UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 6-K

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

For the month of: October, 2016

Platinum Group Metals Ltd.
(SEC File No. 001-33562)

Suite 788 — 550 Burrard Street, Vancouver BC, V6C 2B5, CANADA
Address of Principal Executive Office

Indicate by check mark whether the registrant files or will file annual reports under cover:

Form 20-F ☐ Form 40-F ☒

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

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

SIGNATURE

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

Date:	**October 20, 2016**	*"R. Michael Jones"* **R. MICHAEL JONES** **DIRECTOR & CEO**

EXHIBIT INDEX

EXHIBITS 99.2 AND 99.3 INCLUDED WITH THIS REPORT ARE HEREBY INCORPORATED BY REFERENCE INTO THE REGISTRANT'S REGISTRATION STATEMENT ON FORM F-10 (FILE NO. 333-213985), AND TO BE A PART THEREOF FROM THE DATE ON WHICH THIS REPORT IS SUBMITTED, TO THE EXTENT NOT SUPERSEDED BY DOCUMENTS OR REPORTS SUBSEQUENTLY FILED OR FURNISHED.

EXHIBITS 99.4, 99.5, 99.6 AND 99.7 INCLUDED WITH THIS REPORT ARE HEREBY INCORPORATED BY REFERENCE AS EXHIBITS TO THE REGISTRANT'S REGISTRATION STATEMENT ON FORM F-10 (FILE NO. 333-213985), AND TO BE A PART THEREOF FROM THE DATE ON WHICH THIS REPORT IS SUBMITTED, TO THE EXTENT NOT SUPERSEDED BY DOCUMENTS OR REPORTS SUBSEQUENTLY FILED OR FURNISHED.

Exhibit No.	Description
99.1	News Release dated October 19, 2016.
99.2	Material Change Report dated October 20, 2016.
99.3	Independent Technical Report on the Waterberg Project including Mineral Resource Update and Pre-Feasibility Study dated October 19, 2016.
99.4	Consent of Gordon Cunningham dated October 19, 2016.
99.5	Consent of Robert Goosen dated October 19, 2016.
99.6	Consent of Charles Muller dated October 19, 2016.
99.7	Consent of R. Michael Jones dated October 19, 2016.
99.8	Certificate of Qualified Person, Gordon Cunningham, dated October 19, 2016.
99.9	Certificate of Qualified Person, Robert Goosen, dated October 19, 2016.
99.10	Certificate of Qualified Person, Charles Muller, dated October 19, 2016.

Exhibit 99.1

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Vancouver, BC V6C 2B5
P: 604-899-5450
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News Release

No. 16-330
Oct 19, 2016

Platinum Group Metals Ltd. Announces Positive Independent Pre-Feasibility Study for the Waterberg PGM Project

Annual Planned Production Rate of 744,000 Ounces of Platinum, Palladium, Rhodium and Gold plus 23 million Pounds Nickel and Copper

Initial Probable Mineral Reserve of 12.3 Million Ounces of Platinum, Palladium, Rhodium and Gold

Fully Mechanized Underground Mine Planned for 600,000 Tonnes per Month Would be One of the Larger and Lowest Cash Cost PGM Mines Globally

(Vancouver/Johannesburg) **Platinum Group Metals Ltd.** (PTM-TSX; PLG-NYSE MKT) ("Platinum Group" or the "Company") announces positive results from an Independent Pre-Feasibility Study ("PFS") on the Waterberg PGM Project completed by international and South African engineering firm WorleyParsons RSA (Pty) Ltd. trading as Advisian. Platinum Group holds a 58.62% effective interest in the Waterberg Project with the Japan, Oil, Gas and Metals National Corporation ("JOGMEC") holding a 28.35% interest. Empowerment partner Mnombo Wethu Consultants (Pty) Ltd. ("Mnombo") holds the balance of the joint venture. JOGMEC funding is in place to advance the project through completion of a Feasibility Study ("FS").

Platinum Group Metals plans to continue drilling the deposit and to advance the project to completion of a FS and a construction decision. The Company also plans to file a mining right application, with Joint Venture approval, based substantially on the results of the PFS.

Highlights of the PFS include:

- Validation of the 2014 Waterberg Preliminary Economic Assessment ("PEA") results for a large scale, shallow, decline accessible, mechanized platinum, palladium, rhodium and gold ("4E") mine.
- Annual steady state production rate of 744,000 4E ounces in concentrate.
- A 3.5 year construction period.
- On site life-of-mine average cash cost of US$248 per 4E ounce including by-product credits and exclusive of smelter discounts.
- After-tax Net Present Value ("NPV") of US$320 million, at an 8% discount rate, using three-year trailing average metal prices.
- After-tax NPV of US$507 million, at an 8% discount rate, using investment bank consensus average metal prices.
- Estimated capital to full production of approximately US$1.06 billion including US$67 million in contingencies. Peak project funding estimated at US$914 million.

- After-tax Internal Rate of Return ("IRR") of 13.5% using three-year trailing average price deck.
- After-tax IRR of 16.3% at investment bank consensus average metal prices.
- Probable reserves of 12.3 million 4E ounces.
- Indicated resources updated to 24.9 million 4E ounces (2.5 g/t 4E cut-off) and deposit remains open on strike to the north and below a 1,250 meter arbitrary depth cut-off.

R. Michael Jones, CEO and co-founder of Platinum Group said, "The completion of the PFS significantly increases the Company's attributable 4E reserves and is an important milestone for the project and the Company. The PFS has a similar approach, similar peak funding in US dollar terms with increased production, compared to the PEA.

Waterberg is designed to be a low cost, multi-decline, fully mechanized, mining complex along an initial 13 km deposit strike length with two 300,000 tonne per month mills built in close sequence. At 744,000 ounces annual steady state production and a modelled 18 year mine life, Waterberg is very large and offers excellent exposure to the essential metals of platinum, palladium, rhodium and gold. Amazingly, the deposit is still open. The PFS covers only the first 218 million tonnes in Indicated resources to date.

With the full support of our joint venture partners JOGMEC and Mnombo, we look forward to advancing Waterberg during the remainder of 2016 and 2017 with more drilling, a FS on the initial complex, and the submission of a mining right application. From an original US$20 million commitment by JOGMEC in 2015, approximately US$8 million of further project funding remains to be spent. We are very appreciative of JOGMEC's continued commitment and support.

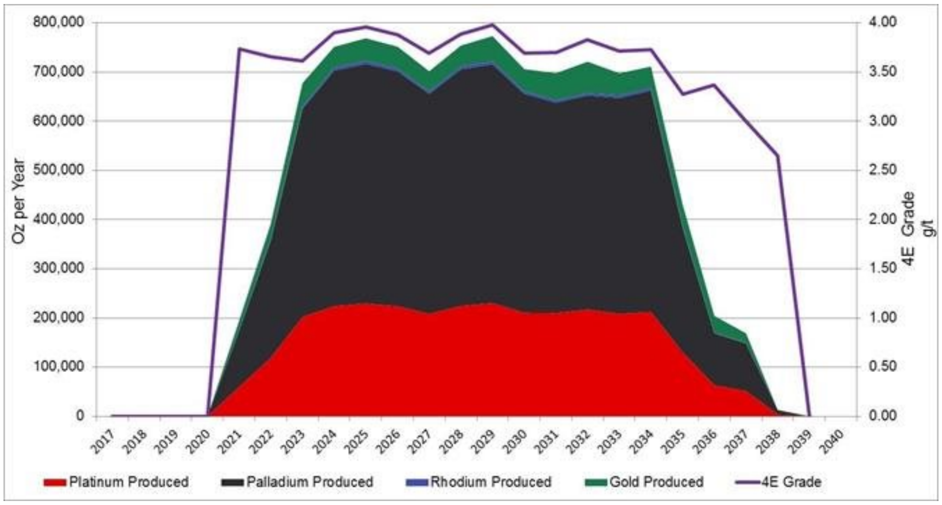
We look forward to growing and advancing Waterberg so that we may fully understand this new part of the Bushveld complex that we and our joint venture partners have discovered. We will work hard to maximize the value of the resource for all stakeholders including shareholders, employees, the Government of South Africa and the local communities."

This press release has been prepared by the qualified persons named herein. This press release is qualified in its entirety by reference to the PFS, which is expected to be filed shortly on SEDAR at www.sedar.com and the SEC's EDGAR site at www.sec.gov. Investors should refer to the PFS for further information.

SCALE AND PGM INDUSTRY POSITION

As a result of the shallow depth, good grades and a fully mechanized mining approach, the Waterberg Project has the opportunity to be a safe mine within the lowest quartile of the industry cost curve. The project resources consist of 60% palladium and the PFS estimates that Waterberg will produce 472,000 ounces of palladium annually; more palladium than the Stillwater Mine produced in 2015, or about 6% of the world's palladium production in 2015. Producing approximately 744,000 4E ounces per year, Waterberg would be one of the largest platinum group metals mine complexes in South Africa based on 2015 production numbers.

It is estimated that Waterberg will create approximately 3,361 new primary highly trained jobs with transferable skills. The increased safety, improved working conditions, low costs and decline access for rapid development all provide attractive features compared to traditional platinum and palladium mines in South Africa. The project is in an area prioritized for economic development. Relations with the small rural community in the area have been business like and positive.



Above: Total Ounces Produced — Life of Mine

KEY RESULTS

Resource Update

Additional drilling since the April 2016 Resource Report has updated the resources as follows:

T-Zone 2.5 g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade Rh g/t	Grade 4E g/t	Grade Cu %	Grade Ni %	Metal 4E Kg	Metal 4E Moz
Indicated	2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122,375	3.934
Inferred	2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75,485	2.427

F-Zone 2.5 g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade Rh g/t	Grade 4E g/t	Grade Cu %	Grade Ni %	Metal 4E Kg	Metal 4E Moz
Indicated	2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651,670	20.952
Inferred	2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260,484	8.375

The Total Mineral Resource is summarized below:

Waterberg Total 2.5 g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Pd g/t	Au g/t	Rh g/t	4E g/t	Cu %	Ni %	Metal 4E Kg	Moz
Indicated	2.5	218.265	1.06	2.18	0.26	0.04	3.55	0.08	0.15	774,045	24.886
Inferred	2.5	97.212	1.03	2.10	0.30	0.03	3.46	0.06	0.11	335,969	10.802

4E = Platinum Group Elements (Pt+Pd+Rh+Au). The cut-offs for mineral resources have been established by a qualified person after a review of potential operating costs and other factors. The mineral resources stated above are shown on a 100% basis, that is, for the Waterberg Project as a whole entity. Conversion Factor used — kg to oz = 32.15076. Numbers may not add due to rounding. Resources do not have demonstrated economic viability. A 5% and 7% geological loss has been applied to the Indicated and Inferred categories respectively. Effective Date Oct 17, 2016. Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance for the assessment of resources; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, US$984/oz Rh, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15.

Total aggregate mineral resources at Waterberg on a 100% project basis have increased slightly since those reported in April 2016. Inferred category resources have decreased to an estimated 10.8 million 4E ounces from 11.71 million ounces 4E Inferred in April, 2016. Indicated category resources have increased to an estimated 24.9 million 4E ounces, from 23.9 million 4E ounces Indicated in April 2016:

1. The mineral resources are classified in accordance with the SAMREC standards. There are certain differences with the "CIM Standards on Mineral Resources and Reserves"; however, in this case the QP believes the differences are not material and the standards may be considered the same. Mineral resources that are not mineral reserves do not have demonstrated economic viability and Inferred resources have a high degree of uncertainty.

2. The mineral resources are provided on a 100% project basis and Inferred and Indicated categories are separate and the estimates have an effective date of 17 October 2016.

3. A cut-off grade of 2.5 g/t 4E for both the T and the F-Zones is applied to the selected base case mineral resources.

4. Cut off for the T and the F-Zones considered costs, smelter discounts, concentrator recoveries from previous engineering work completed on the property by the

Company. The resource model was cut-off at an arbitrary depth of 1,250 meters, although intercepts of the deposit do occur below this depth.

5. Mineral resources were completed by Mr. CJ Muller of CJM Consulting.

6. Mineral resources were estimated using kriging methods for geological domains created in Datamine from 303 original holes and 483 deflections. A process of geological modelling and creation of grade shells using indicating kriging was completed in the estimation process.

7. The estimation of mineral resources has taken into account environmental, permitting and legal, title, and taxation, socio-economic, marketing and political factors.

8. The mineral resources may be materially affected by metals prices, exchange rates, labor costs, electricity supply issues or many other factors detailed in the Company's Annual Information Form.

9. The data that formed the basis of the estimate are the drill holes drilled by Platinum Group, which consist of geological logs, the drill hole collars surveys, the downhole surveys and the assay data. The area where each layer was present was delineated after examination of the intersections in the various drill holes.

10. There is no guarantee that all or any part of the mineral resource not included in the current reserves will be upgraded and converted to a mineral reserve.

Reserves

Reserves are stated on a 100% Project Basis. Reserves are a subset of the Indicated resources and the mine plan was developed from the October 2016 resource model above and includes mine modifying factors such as geological losses, dilution, development overbreak, mine design factors, in stope losses and the extraction ratio from the mining methods applied to the T and F-Zones.

The independent Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian). The table below shows the prill splits, which are calculated using the individual metal grades reported as a percentage of the total 4E grade. There are no Inferred mineral resources included in the reserves.

Prill Splits

	Prill Split				Grade	
	Pt	Pd	Au	Rh	Cu	Ni
Zone	%	%	%	%	%	%
T-Zone	29	49	21	1	0.16	0.08
F-Zone	30	64	5	1	0.07	0.16

Probable Mineral Reserve at 2.5 g/t 4E Cut-off— Tonnage and Grades

Waterberg Probable Mineral Reserve — Tonnage and Grades

Zone	Mt	Cut-off grade (g/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)	Cu (%)	Ni (%)
T-Zone	16.5	2.5	1.14	1.93	0.83	0.04	3.94	0.16	0.08
F-Zone	86.2	2.5	1.11	2.36	0.18	0.04	3.69	0.07	0.16
Total	**102.7**	**2.5**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**	**0.08**	**0.15**

Probable Mineral Reserve at 2.5 g/t 4E Cut-off— Contained Metal

Waterberg Probable Mineral Reserve — Contained Metal

Zone	Mt	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	4E (Moz)	4E content (kg)	Cu (Mlb)	Ni (Mlb)
T-Zone	16.5	0.61	1.03	0.44	0.02	2.09	65,097	58.21	29.10
F-Zone	86.2	3.07	6.54	0.51	0.10	10.22	318,007	132.97	303.94
Total	**102.7**	**3.67**	**7.57**	**0.95**	**0.12**	**12.32**	**383,103**	**191.18**	**333.04**

Reasonable prospects of economic extraction were determined with the following assumptions: Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance for the assessment of resources and reserves; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, US$984/oz Rh, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15. Smelter payability of 85% was estimated for 4E and 73% for Cu and 68% for Ni. The effective date is October 17, 2016. A 2.5 g/t Cut-off was used and checked against a pay-limit calculation. Independent Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian). The mineral reserves may be materially affected by changes in metals prices, exchange rates, labor costs, electricity supply issues or many other factors. See Risk Factors in 43-101 report on www.sedar.com and the Company's Annual Information Form. The reserves are estimated under SAMREC with no material difference to the CIM 2014 definitions in this case.

The estimation of mineral reserves has taken into account environmental, permitting and legal, title, taxation, socio-economic, marketing and political factors. Based on the cut-off grade and a maximum depth cut-off of 1,250 meters the Probable reserve will support an 18 year mine life.

PROJECT PFS RESULTS

The PFS results validate the PEA with similar capital costs in USD, increased production profile (from 655,000 3E ounces/yr PEA to 744,000/yr 4E ounces PFS) and an increase in sustaining capital. Optimization of the mine plan and working on reducing underground sustaining development capital will be part of the upcoming Feasibility Study.

PROJECT MODEL TIMELINE

The project time line includes a construction decision following the completion of a FS and first production 3 years later. Under the PFS model, first production is estimated as mid-2021, if the FS is completed at the end of 2017 and a mining right and other permits are granted as planned. Final reef tonnes are scheduled to be mined in 2038.

CAPITAL COSTS

Capital costs to full production and peak funding of the project are estimated in Rand 2016 terms. Peak Funding is estimated at US$914 million. The costs are estimated in USD at 15R/1USD with a flat exchange rate. Escalation of costs in Rand terms are estimated to be mostly offset over time by future Rand devaluation.

TOTAL CAPITAL

Facility Code	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
2000	Underground Mining	6,092	9,766	406	651
3000	Concentrator	2,850	159	190	11
4000	Shared Services & Infrastructure	1,063	43	71	03
5000	Regional Infrastructure	2,566	0	171	0
6000	Site Support Services	691	67	46	04
7000	Project Delivery Management	1,399	147	93	10
8000	Other Capitalised Costs	246	83	16	06
9000	Contingency	999	1,202	67	80
	Total Capital	**15,906**	**11,468**	**1,060**	**765**

The estimates for the scope of work, within the given battery limits, and subject to the qualifications, assumptions and exclusions contained in the PFS, are considered to be within the accuracy range required for a PFS of ±25%. Monte Carlo simulation was used to provide a 12% contingency that was used in the estimates.

Waterberg 2016 PFS Results Details

Item	Units	Total
Mined and Processed	Mtpa	7.20
Platinum	g/t	1.11
Palladium	g/t	2.29
Gold	g/t	0.29
Rhodium	g/t	0.04
4E	g/t	3.73
Copper	%	0.08
Nickel	%	0.15
	Recoveries	
Platinum	%	82.5%
Palladium	%	83.2%
Gold	%	75.3%
Rhodium	%	59.4%
4E	%	82.1%
Copper	%	87.9%
Nickel	%	48.8%
	Produced in Concentrate	
Concentrate	ktpa	285
Platinum	g/t	24.2
Palladium	g/t	51.5

Item	Units	Total
Gold	g/t	4.9
Rhodium	g/t	0.6
4E	g/t	81
Copper	%	1.9
Nickel	%	1.8

Recovered Metal in Concentrate

Platinum	kozpa	222
Palladium	kozpa	472
Gold	kozpa	45
Rhodium	kozpa	6
4E	kozpa	744
Copper	Mlbpa	11
Nickel	Mlbpa	12

KEY ASSUMPTIONS

Economic Assumptions

Parameter	Unit	3 Year Trailing Average	Investment Bank Consensus Price
Platinum	USD/oz	1,212	1,213
Palladium	USD/oz	710	800
Gold	USD/oz	1,229	1,300
Rhodium	USD/oz	984	1,000
T and F Combined Basket (4E)	USD/oz	899	960
Nickel	USD/lb	6.10	7.50
Copper	USD/lb	2.56	2.90
Base Metals Refining Charge	% Gross Sales Pay	85%	
Copper Refining Charge	% Gross Sales Pay	73%	
Nickel Refinery Charge	% Gross Sales Pay	68%	

FINANCIAL RESULTS

Average Life of Mine ("LOM") Operating Cost Rates and Totals per Area in ZAR and USD

	Average LOM (ZAR/t)	Total LOM (ZAR M)	Average LOM (USD/t)	Total LOM (USD M)
Mining	R 271.90	R 27,915	$ 18.13	$ 1,861
Engineering & Infrastructure	R 107.49	R 11,036	$ 7.17	$ 736
General & Admin	R 40.71	R 4,180	$ 2.71	$ 279
Process	R 154.52	R 15,864	$ 10.30	$ 1,058
Total OPEX Cost	**R 574.62**	**R 58,994**	**$ 38.31**	**$ 3,933**

4E Cash Costs before and after Credits and Costs

Item	US$/oz 4E in Concentrate Life-of-Mine Average	5-Year Average 2022 - 2026	10-Year Average 2022 - 2031
Mine Site Cash Cost	389	390	374
Nickel Credits	98	97	98
Copper Credits	42	40	40
Total Mine Cash Costs After Credits	**248**	**253**	**236**
Realisation cost (smelter "cost", transport)	232	**224**	**231**
Total Cash Costs After Credits	**481**	**477**	**467**

Financial Results Three Year Trailing Average Price Deck 15R/1USD Flat

Item	Discount Rate	ZAR Millions (Before Tax)	ZAR Millions (After Tax)	USD Millions (Before Tax)	USD Millions (After Tax)
	Undiscounted	36,096	25,042	2,406	1,669
	4.0%	18,213	11,883	1,214	792
	6.0%	12,666	7,808	844	520
Net Present Value	8.0%	8,565	4,805	571	320
	10.0%	5,519	2,584	368	172
	12.0%	3,249	939	217	62
	14.0%	1,555	-278	104	-19
Internal Rate of Return		16.6%	13.5%	16.6%	13.5%
Project Payback Period (Years) from 2017		10	10	10	10

Investment Bank Consensus Price Deck

Item	Discount Rate	Before Tax (ZAR M)	After Tax (ZAR M)	Before Tax (USD M)	After Tax (USD M)
	Undiscounted	45,781	31,946	3,052	2,130
	4.0%	24,180	16,184	1,612	1,079
	6.0%	17,426	11,263	1,162	750
Net Present Value	8.0%	12,402	7,610	827	507
	10.0%	8,641	4,884	576	325
	12.0%	5,812	2,842	387	189
	14.0%	3,676	1,311	245	87
Internal Rate of Return		19.8%	16.3%	19.8%	16.3%
Project Payback Period (Years) from 2017		9	9	9	9

Sensitivity Analysis — Post Tax Three Year Trailing Average Price Deck 15R/1USD Flat

Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter
Metal Prices	-20%	-10%	0%	10%	20%
IRR (post-tax)	5%	10%	13.5%	17%	20%
NPV (8% Discount) (R000)	-2,467	1,211	4,805	8,344	11,854
NPV (8% Discount) ($000)	-164	67	320	556	790
Head Grade	-20%	-10%	0%	10%	20%
IRR (post-tax)	6%	10%	13.5%	16%	19%
NPV (8% Discount) (R000)	-1,513	1,562	4,805	7,562	10,505
NPV (8% Discount) ($000)	-101	104	320	504	700
Capex	-20%	-10%	0%	10%	20%
IRR (post-tax)	17%	15%	13.5%	12%	10%
NPV (8% Discount) (R000)	8,161	6,484	4,805	3,109	1,395
NPV (8% Discount) ($000)	544	432	320	207	93
Opex	-20%	-10%	0%	10%	20%
IRR (post-tax)	18%	16%	13.5%	12%	10%
NPV (8% Discount) (R000)	7,435	6,121	4,805	3,246	2,124
NPV (8% Discount) ($000)	496	408	320	216	142

CONSTRUCTION AND DEVELOPMENT METHODOLOGY

The immediate next steps for the Waterberg Project are the development of a mining right application and working towards a FS. The PFS specifies that the project will utilize an EPCM contractor and a mining contractor for the development of the mine, overseen by an experienced in-house Platinum Group owners' team. Initial development is planned with contractors with a later takeover for mining by an owners' team. Equipment operators will be trained at an existing MQA certified training center developed and owned by Platinum Group.

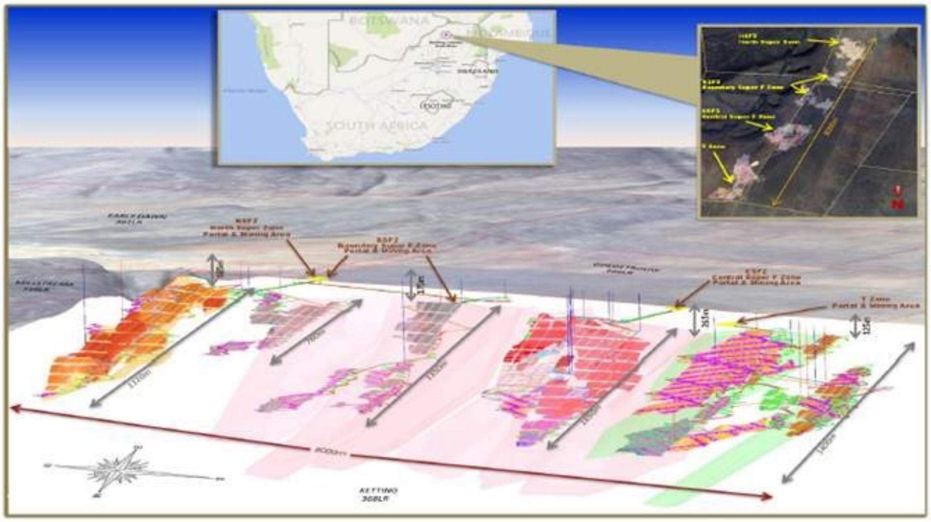
PLANNED MINING METHODS

The mining blocks of the Waterberg deposits occur at depths from 140 meters to 1,250 meters along 8,000 meters of strike length of reserves. The deposit is known from drill intercepts to continue below 1,250 meters and to extend for over 13,000 meters of strike length.

Access to the mining complex is planned by three decline ramp clusters. Decline ramps have advantages over vertical shafts in terms of capital cost, and importantly, time. Declines to the depths of the top of the Waterberg deposit can be developed over 24-36 months whereas vertical shafts, shaft infrastructure and equipping can take six to seven years.

Mining will be completed by safe, efficient fully mechanized methods and the dip and thickness of the zones are driving the mining method selection. A fleet of approximately 400 trackless machines including drill rigs, loaders, dump trucks and other trackless machines will be used for mining and development. A minimum mining width has been set at three meters so that all mining can be fully mechanized, safe and efficient.

Mining Method	Vertical Mining Height	Dip of the Reef	Key Advantages
Blind Longitudinal Retreat	3 - 15m	$\leq 35°$	On reef development. Good grade, extraction.
Sub-level Open Stoping - Longitudinal	3 - 15m	>35°	On reef development. Good extraction. Bulk and low cost.
Sub-level Open Stoping - Transverse	>15m	>35°	Bulk method. High efficiency, large tonnage, good extraction, ultra-low cost.



Above: Portal and Underground Layouts (October 17, 2016)

The mine utilizes a large supply of new mechanized trackless mobile equipment for mining and feed ore onto large conveyors from underground to the processing plant. The mine will rely on both the T and the F reef from more than one portal to make the tonnage profile steady state of 600,000 tonnes per month.

METALLURGICAL RECOVERY AND PROCESSING

The flotation test work indicated that the Waterberg ores are amenable to treatment by conventional flotation without the need for re-grinding. A standard flotation concentrator can be used to produce a saleable concentrate, at a 4E grade of no less than 80 g/t, with no deleterious products. A 4E recovery rate in excess of 80% is expected at the proposed mill feed grades.

Metallurgical test work on the Waterberg ores by SGS, Mintek and DRA has focused on recovery of 4E platinum group elements and copper-nickel sulphides with the objective of producing a high grade concentrate attractive for smelting in South Africa.

The processing plant is designed in two 300,000 tonne/month Mill-Float in two cycles “MF-2” standard platinum industry modules for ramp-up and operational flexibility. A JOGMEC reagent circuit with some opportunity for increased recovery has also been tested. The plant modules are designed to accept T reef, F reef or a blended combination. The ore types can be co-mingled without negatively affecting recoveries.

INFRASTRUCTURE

The main infrastructure requirements for the Waterberg Project are access roads, tailings storage, water management, power supply and process plant to service and treat the targeted mine production.

The Waterberg Project is situated in a remote area and will require approximately 32 km of existing unpaved roads to be surfaced.

BULK WATER SUPPLY

A combination of sewage effluent together with groundwater is considered the most viable solution to meet the proposed mining schedule. Several available options were considered including a pipeline to be developed for several users to the south, including for other proposed and active mines. This option was not chosen as it is considered to have greater risk due to the large number of parties involved. Sufficient water sources for the project were identified and early discussions for the preferred arrangements were positive.

BULK POWER SUPPLY

The updated electricity supply plan compiled by Eskom provides for the establishment of two 77 km long 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation.

The development of the abovementioned infrastructure will be done in conjunction with Eskom on a self-build basis and this work is already in an advanced stage including the application for permits for the proposed power line.

METALS MARKETS AND OFFTAKE

The Waterberg Project will produce a flotation concentrate from the processing plant which is assumed to be sold or toll treated into the local South African market.

Production of up to 285,000 tonnes of concentrate per annum will be available at peak production. The concentrate will contain approximately 80 g/t 4E's plus copper at between 1% and 9.2% and nickel at between 1.1% and 5% copper.

The concentrate does not contain any penalty elements such as chrome and is rich in sulphur, thus making it a desirable concentrate to blend with other high chrome concentrates.

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal indications from smelters are that the concentrate is attractive.

Based on the large volume of concentrate and the significantly lower operating cost of the metal without the smelter discount, the consideration of production of an onsite smelter matte or combination with other Northern Limb material for further critical mass is recommended. The company will consider this recommendation with its partners JOGMEC and Mnombo.

LABOUR, SOCIAL AND PERMITTING

The Waterberg Project will create safe long term jobs with transferable mechanized equipment operations skills for a large part of the work force. The increased safety and ability to create good paying well trained jobs is an attractive community benefit. The social license to operate has been a focus of the company with ongoing positive meetings and interactions. The next stage of the project will involve the development of a Social and Labour Plan as part of the mining right application. The project involves normal measures for the protection of the environment similar to other platinum mining operations.

2016 AND 2017 PROGRAMS FUNDED

During 2015, JOGMEC committed to fund US$20 million of project work at Waterberg. Approximately US$8 million of that commitment remains to be completed and will fund 100% of the costs for the balance of 2016 and into 2017.

QUALIFIED PERSONS

The following Qualified Persons have completed work in preparation of the PFS and are responsible for the contents:

- **Independent Engineering Qualified Person:**
  Mr. Robert L Goosen
  (B.Eng. (Mining Engineering)) Pr. Eng. (ECSA)
  Advisian/WorleyParsons Group

- **Independent Geological Qualified Person:**
  Mr. Charles J Muller
  (B.Sc. (Hons) Geology) Pr. Sci. Nat.
  CJM Consulting (Pty) Ltd

- **Independent Engineering Qualified Person:**
  Mr. Gordon I. Cunningham
  B. Eng. (Chemical), Pr. Eng. (ECSA), Professional association to FSAIMM
  Turnberry Projects (Pty) Ltd.

This press release has been reviewed and approved by R. Michael Jones, P.Eng., a non-independent Qualified Person and the CEO of the Company. He has verified the technical information for disclosure in this press release by reviewing the work of the QPs on a test basis, visiting the site and meeting with the project QPs through the development of the PFS.

DATA VERIFICATION, QUALITY ASSURANCE AND CONTROL

Scientific and Technical Information in this Press Release related to mineral resources has been reviewed and approved by Charles J Muller, (BScHons) Pr Sci Nat (Reg. No 400201/04), an independent consulting geologist and resource estimator of CJM Consulting, an independent qualified person as defined in National Instrument 43-101 -Standards of Disclosure for Mineral Projects (“NI 43-101”). He has verified the data by reviewing the detailed assay and geological information and metallurgical work on the Waterberg deposit. He is satisfied that the data is appropriate for the resource estimate by reviewing the core, assay certificates and quality control information as well as reviewing the procedures on sampling, chain of custody and data base records of the Platinum Group exploration team.

Base metals and other major elements were determined by multi acid digestion with Inductively Coupled Plasma (“ICP”) finish and PGEs were determined by conventional fire assay and ICP finish. Setpoint Laboratories is an experienced ISO 17025 SANAS accredited laboratory in assaying and have utilized a standard quality control system including the use of standards. Bureau Veritas South Africa and Genalysis of Australia with similar standards and approaches have been used for assays and umpire checks. Platinum Group utilized a well-documented system of inserting blanks and standards into the assay stream and has a strict chain of custody and independent lab re-check system for quality control. Details are available in the NI 43 101 reports on the project at www.sedar.com and www.platinumgroupmetals.net

The independent QPs for the PFS (CJ Muller, GI Cunningham and RL Goosen) have visited the Waterberg property for personal inspection during 2016. Mr. RL Goosen last visited the site on 13 October 2016, Mr. GI Cunningham on 13 October 2016 and Mr. CJ Muller on 29 March 2016. They all have undertaken due diligence with respect to the PTM data. Other than as specified below they jointly take responsibility for the report.

- Charles J Muller — Geology and Mineral Resource Estimation
- Robert L Goosen — Reserve Estimation, Mining and Infrastructure
- Gordon I Cunningham — Metallurgy, Metals Markets, Offtake, Capital cost and financial model

The QPs have verified the data sufficiently for the reporting of resources, reserves and this Pre-Feasibility Study. The QPs have reviewed and approved their relevant section of this press release.

OPPORTUNITIES

- The company plans to work towards optimization of the mine plan, development plan and waste development plan for ventilation with the objective of reducing sustaining capital in the FS stage.
- Further drilling will be completed with the objective to upgrade some of the resources and the deposit remains open. A longer mine life will also be targeted in the high grade T reef areas. High grade thickness areas in the F reef will also receive targeted drilling with the objective of increasing definition for the FS mine plan.
- Further metallurgical work will be completed in the FS including the potential for increased recoveries using the JOGMEC circuit.
- The Waterberg concentrate is attractive for smelting and is of large strategic scale importance to the industry.

RISKS

- The project at a PFS stage has all of the normal mining projects risks including but not limited to, estimation risk for the resources and reserves, recovery risks, capital cost and operating cost estimation risks, permitting and community and surface rights access risk.
- Government regulation stability and amendments of the fiscal regime is an additional risk.
- Waterberg is a large green-fields project and final off-take of the proposed metal is yet to be negotiated for a large volume of concentrate.

ABOUT PLATINUM GROUP METALS LTD.

Platinum Group, based in Johannesburg, South Africa and Vancouver, Canada, has a successful track record with more than 20 years of experience in exploration, mine discovery, mine construction and mine operations.

Formed in 2002, Platinum Group holds significant mineral rights in the Bushveld Igneous Complex of South Africa, which is host to over 70% of the world’s primary platinum production. The Company is currently focused on ramping up the Maseve Mine, its first near-surface platinum mine, to commercial production.

Platinum Group has expanded its exploration and development efforts on the North Limb of the Bushveld Complex on the Waterberg Project. Waterberg represents a new bulk type of platinum, palladium and gold deposit.

On behalf of the Board of
Platinum Group Metals Ltd.

"R. Michael Jones"
President and CEO

For further information, contact:
R. Michael Jones, President
or Kris Begic, VP, Corporate Development
Platinum Group Metals Ltd., Vancouver
Tel: (604) 899-5450 / Toll Free: (866) 899-5450
www.platinumgroupmetals.net

Disclosure

The Toronto Stock Exchange and the NYSE MKT LLC have not reviewed and do not accept responsibility for the accuracy or adequacy of this news release, which has been prepared by management.

This press release contains forward-looking information within the meaning of Canadian securities laws and forward-looking statements within the meaning of U.S. securities laws (collectively "forward-looking statements"). Forward-looking statements are typically identified by words such as: believe, expect, anticipate, intend, estimate, plans, postulate and similar expressions, or are those, which, by their nature, refer to future events. All statements that are not statements of historical fact are forward-looking statements. Forward-looking statements in this press release include, without limitation, the projections and assumptions relating to future events that are contained in the PFS, including, without limitation NPV, IRR, costs, potential production of the Waterberg Project and other operational and economic projections with respect to the Waterberg Project; future activities at Waterberg and the funding of such activities; trends in metal prices; potential future market conditions; the Company's overall capital requirements and future capital raising activities, plans and estimates regarding exploration, studies, development, construction and production on the Company's properties, other economic projections and the Company's outlook. Statements of mineral resources and mineral reserves also constitute forward-looking statements to the extent they represent estimates of mineralization that will be encountered on a property and/or estimates regarding future costs, revenues and other matters. Although the Company believes the forward-looking statements in this press release are reasonable, it can give no assurance that the expectations and assumptions in such statements will prove to be correct. The Company cautions investors that any forward-looking statements by the Company are not guarantees of future results or performance, and that actual results may differ materially from those in forward-looking statements as a result of various factors, including; the Company's capital requirements may exceed its current expectations; the uncertainty of cost, operational and economic projections; the ability of the Company to negotiate and complete future funding transactions; variations in market conditions; the nature, quality and quantity of any mineral deposits that may be located; metal prices; other prices and costs; currency exchange rates; the Company's ability to obtain any necessary permits, consents or authorizations required for its activities; the Company's ability to produce minerals from its properties successfully or profitably, to continue its projected growth, or to be fully able to implement its business strategies; and other risk factors described in the Company's shelf prospectus and registration statement, Form 40-F annual report, annual information form and other filings with the Securities and Exchange Commission and Canadian securities regulators, which may be viewed at www.sec.gov and www.sedar.com, respectively.

This press release also includes a reference to mineral resources and mineral reserves. The estimation of resources and reserves is inherently uncertain and involves judgement. Mineral resources that are not reserves do not have demonstrated economic viability. Judgements associated with geology, tonnage grades in place and that can be mined may prove to be unreliable and inaccurate. Fluctuations in metals prices, exchange rates, labour costs and government regulations among other things may materially affect resources and reserves. The company does not yet have a right to mine the reported resources and reserves and there can be no assurance that the company will convert its prospecting permits to a mining right.

Cautionary Note to U.S. and other Investors

Estimates of mineralization and other technical information included or referenced in this press release have been prepared in accordance with NI 43-101. The definitions of proven and probable reserves used in NI 43-101 differ from the definitions in SEC Industry Guide 7. Under SEC Industry Guide 7 standards, a "final" or "bankable" feasibility study is required to report reserves, the three-year historical average price is used in any reserve or cash flow analysis to designate reserves and the primary environmental analysis or report must be filed with the appropriate governmental authority. As a result, the reserves reported by the Company in accordance with NI 43-101 may not qualify as "reserves" under SEC standards. In addition, the terms "mineral resource", "measured mineral resource", "indicated mineral resource" and "inferred mineral resource" are defined in and required to be disclosed by NI 43-101; however, these terms are not defined terms under SEC Industry Guide 7 and normally are not permitted to be used in reports and registration statements filed with the SEC. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Investors are cautioned not to assume that any part or all of the mineral deposits in these categories will ever be converted into reserves. "inferred mineral resources" have a great amount of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an inferred mineral resource will ever be upgraded to a higher category. Under Canadian securities laws, estimates of inferred mineral resources may not form the basis of feasibility or pre-feasibility studies, except in rare cases. Additionally, disclosure of "contained ounces" in a resource is permitted disclosure under Canadian securities laws; however, the SEC normally only permits issuers to report mineralization that does not constitute "reserves" by SEC standards as in place tonnage and grade without reference to unit measurements. Accordingly, information contained or referenced in this press release containing descriptions of the Company's mineral deposits may not be comparable to similar information made public by U.S. companies subject to the reporting and disclosure requirements of United States federal securities laws and the rules and regulations thereunder.

Exhibit 99.2

FORM 51-102F3
MATERIAL CHANGE REPORT

ITEM 1. NAME AND ADDRESS OF COMPANY

PLATINUM GROUP METALS LTD. (the "Company" or "Platinum Group") 788 — 550 Burrard Street Vancouver BC, V6C 2B5
Telephone: (604) 899-5450 Facsimile: (604) 484-4710

ITEM 2. DATE OF MATERIAL CHANGE

October 19, 2016

ITEM 3. NEWS RELEASE

A news release was disseminated on October 19, 2016 to the TSX as well as through various other approved public media and was SEDAR filed with the British Columbia, Alberta, Saskatchewan, Manitoba, Ontario, Québec, New Brunswick, Nova Scotia, Prince Edward Island and Newfoundland Securities Commissions.

ITEM 4. SUMMARY OF MATERIAL CHANGE

Platinum Group announced positive results from an independent pre-feasibility study ("PFS") on the Waterberg Project contained in a technical report dated October 19, 2016 and titled "Independent Technical Report on the Waterberg Project Including Mineral Resource Update and Pre-Feasibility Study" (the "Technical Report") prepared by WorleyParsons RSA (Pty) Ltd. trading as Advisian. Platinum Group is to hold a 58.62% effective interest in the Waterberg Project (including through its minority interest in Mnombo) with the Japan, Oil, Gas and Metals National Corporation ("JOGMEC") holding a 28.35% interest. Empowerment partner Mnombo Wethu Consultants (Pty) Ltd. ("Mnombo") will hold the balance of the joint venture.

Highlights of the PFS include:

- Validation of the 2014 Waterberg Preliminary Economic Assessment ("PEA") results for a large scale, shallow, decline accessible, mechanized platinum, palladium, rhodium and gold ("4E") mine.

- Annual steady state production rate of 744,000 4E ounces in concentrate.

- A 3.5 year construction period.

- On site life-of-mine average cash cost of US$248 per 4E ounce including by-product credits and exclusive of smelter discounts.

- After-tax Net Present Value ("NPV") of US$320 million, at an 8% discount rate, using three-year trailing average metal prices.

- After-tax NPV of US$507 million, at an 8% discount rate, using investment bank consensus average metal prices.

- Estimated capital to full production of approximately US$1.06 billion including US$67 million in contingencies. Peak project funding estimated at US$914 million.

- After-tax Internal Rate of Return ("IRR") of 13.5% using three-year trailing average price deck.

- After-tax IRR of 16.3% at investment bank consensus average metal prices.
- Probable reserves of 12.3 million 4E ounces (2.5 g/t 4E cut-off).
- Indicated resources updated to 24.9 million 4E ounces (2.5 g/t 4E cut-off) and deposit remains open on strike to the north and below a 1,250 meter arbitrary depth cut-off.

Platinum Group Metals plans to continue drilling the deposit and to advance the project to completion of a feasibility study and a construction decision. The Company also plans to file a mining right application, with joint venture approval, based substantially on the results of the PFS.

ITEM 5. FULL DESCRIPTION OF MATERIAL CHANGE

5.1 Full Description of Material Change

Platinum Group announced positive results from an independent pre-feasibility study ("PFS") of the Waterberg Project contained in a technical report dated October 19, 2016 and titled "Independent Technical Report on the Waterberg Project Including Mineral Resource Update and Pre-Feasibility Study" (the "Technical Report") prepared by WorleyParsons RSA (Pty) Ltd. trading as Advisian. Platinum Group is to hold a 58.62% effective interest in the Waterberg Project (including through its minority interest in Mnombo) with the Japan, Oil, Gas and Metals National Corporation ("JOGMEC") holding a 28.35% interest. Empowerment partner Mnombo Wethu Consultants (Pty) Ltd. ("Mnombo") will hold the balance of the joint venture.

Highlights of the PFS include:

- Validation of the 2014 Waterberg Preliminary Economic Assessment ("PEA") results for a large scale, shallow, decline accessible, mechanized platinum, palladium, rhodium and gold ("4E") mine.
- Annual steady state production rate of 744,000 4E ounces in concentrate.
- A 3.5 year construction period.
- On site life-of-mine average cash cost of US$248 per 4E ounce including by-product credits and exclusive of smelter discounts.
- After-tax Net Present Value ("NPV") of US$320 million, at an 8% discount rate, using three-year trailing average metal prices.
- After-tax NPV of US$507 million, at an 8% discount rate, using investment bank consensus average metal prices.
- Estimated capital to full production of approximately US$1.06 billion including US$67 million in contingencies. Peak project funding estimated at US$914 million.
- After-tax Internal Rate of Return ("IRR") of 13.5% using three-year trailing average price deck.
- After-tax IRR of 16.3% at investment bank consensus average metal prices.
- Probable reserves of 12.3 million 4E ounces (2.5 g/t 4E cut-off).

- Indicated resources updated to 24.9 million 4E ounces (2.5 g/t 4E cut-off) and deposit remains open on strike to the north and below a 1,250 meter arbitrary depth cut-off.

Platinum Group Metals plans to continue drilling the deposit and to advance the project to completion of a FS and a construction decision. The Company also plans to file a mining right application, with Joint Venture approval, based substantially on the results of the PFS.

The following is the extracted summary section from the Technical Report, which is incorporated by reference herein. For full technical details, reference should be made to the complete text of the Technical Report.

The following summary does not purport to be a complete summary of the Technical Report. The summary is subject to all of the assumptions, qualifications and procedures set out in the Technical Report, and is qualified in its entirety with reference to the full text of the Technical Report. Readers should read this summary in conjunction with the Technical Report. Readers are directed to review the full text of the report, available for review under the Company's profile on SEDAR, at www.sedar.com, and on the SEC's EDGAR website, at www.sec.gov.

1. Summary

The following items are the main components forming the purpose of the Pre-Feasibility Study:

- To update the Mineral Resource estimate to October 2016 and to publish the results of the PFS;
- To determine the optimal techno-economic solution that considers all opportunities and risks, that exceeds the investment criteria hurdle and is aligned with the Company's strategy;
- To justify the expenditure for a Feasibility Study of one selected project option;
- To compile a work programme, budget and schedule baseline for the development of the scope and deliverables of the Feasibility Study;
- To provide a framework of project options as a converging view, to demonstrate that all the discarded project options have been studied to the degree that they are clearly identified as inferior and will not re-emerge as potential options;
- To optimize the project size, scope, technical and production parameters by evaluating all the alternative technology and implementation options, as well as the project costs and benefits
- To determine targets for further value enhancement and risk reduction.
- To provide the basis for a Mining Rights Application.

1.1 Introduction

This report was prepared in compliance with National Instrument 43—101, Standards of Disclosure for Mineral Projects (NI 43—101), and documents the results of ongoing exploration and project work.

The project is the development of large greenfield platinum mine and concentrator plant north of the town of Mokopane in the Province of Limpopo.

A Preliminary Economic Assessment (PEA) on the original Waterberg JV was completed and announced in February 2014.

The resource estimate includes the T Zone, F South, F Central, F Boundary and F North with the shallowest edge of the known deposit on the T-Zone at approximately 140m below surface. The resource estimate has been cut off at an arbitrary depth of 1,250m vertical. Drill intercepts well below 1,250m vertical indicate the deposit continues and is open down dip from this depth. The deposit is 13km long and remains open along strike to the north.

The key features of the Waterberg 2016 PFS include:

- Development of a large, mechanized, underground mine that is planned at a 7.2Mtpa throughput scenario;
- Planned steady state annual production rate of 744 koz of platinum, palladium, rhodium and gold (4E) in concentrate;
- Estimated Capital to full production requirement of approximately ZAR15,906 billion (US$1,060 million), including ZAR999 million (US$67 million) in contingencies;
- Peak funding ZAR13,694 million (US$914 million);

- After-tax Net Present Value (NPV) of ZAR4,805 million (US$320 million), at an 8% discount rate (three year trailing average price desk 31 July 2016 US$1,212/oz Pt, US$710/oz Pd, US$984/oz Rh, US$1,229/oz Au, US$/ZAR 15);

- After-tax Net Present Value (NPV) of ZAR7,610 million (US$507 million), at an 8% discount rate (Investment Bank Consensus) Price US$1,213/ozPt, US$800/oz Pd, US$1,000 Rh, US$1,300/oz Au, US$/ZAR 15

- After-tax Internal Rate of Return (IRR) of 13.5% (three year trailing average price deck); and

- Internal Rate of Return (IRR) of 16.3% After-tax (Investment Bank Consensus Price).

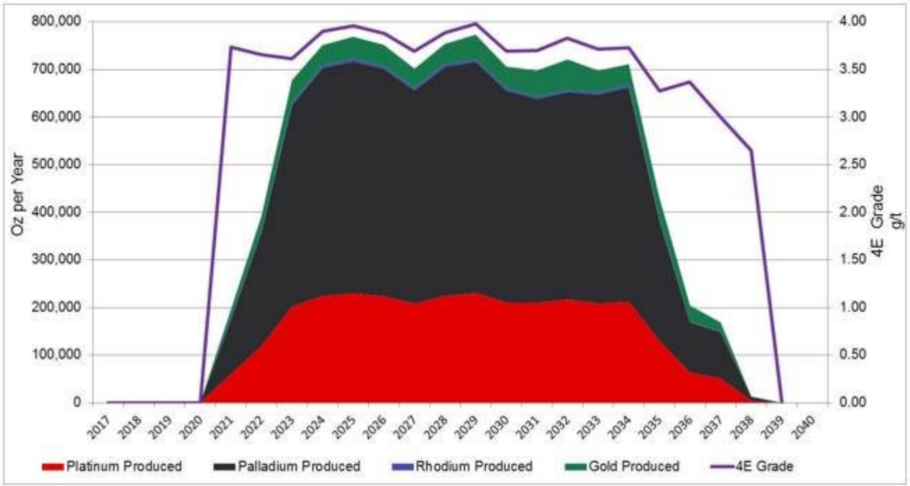


Figure 1-1: Total Ounces Produced

Mine production is shown in Figure 1-2 and the after tax cash flow is shown in Figure 1-3.

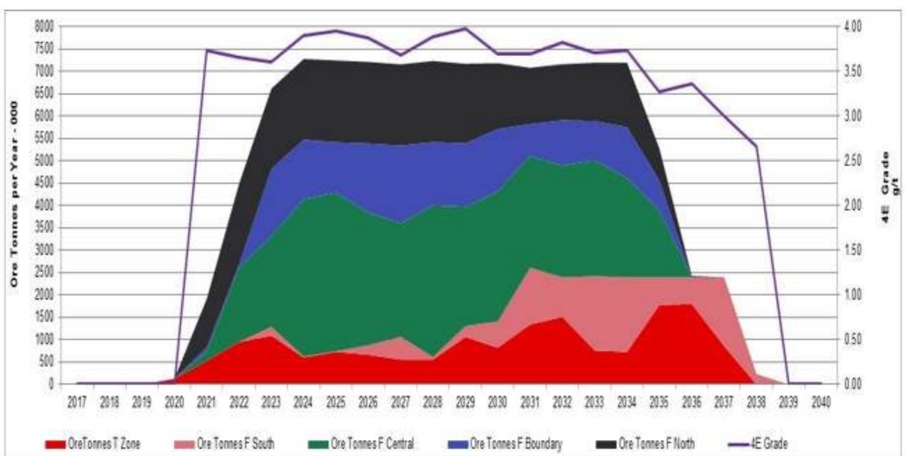


Figure 1-2: Total Mine Production

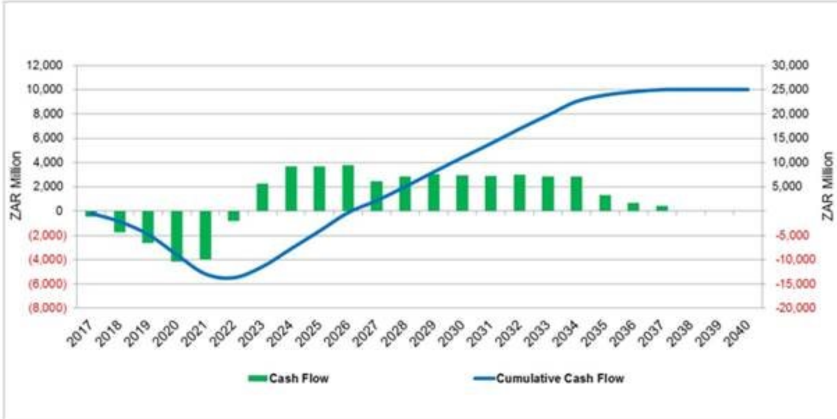


Figure 1-3: Annual Cashflow after Tax

1.2 Ownership

The ownership structure consists of:

- Platinum Group Metals (RSA) (Pty) Ltd, abbreviated to PTM (45.65% directly)
- JOGMEC (28.35%)
- BEE partner Mnombo Wethu Consultants (26%).

Because of PTM's 49.90% ownership in Mnombo, the Company has a direct and indirect 58.62% overall interest in the project. Platinum Group Metals is the operator.

The size and scale of the Waterberg Project represents a significant alternative to narrow width, conventional, Merensky and UG2 mining on the Western and Eastern Limbs of the Bushveld Complex.

The government of South Africa holds the mineral rights to the project properties under the Mineral and Petroleum Resources Development Act (Act, 28 of 2002). The mineral rights are held through a mining right under the Mineral and Petroleum Resources Developm

1.3 Location and Access

The Waterberg Mineral Project is located approximately 85km north of the town of Mokopane in the Province of Limpopo, South Africa as shown in Figure 1-4.

Platinum Group Metals has been granted prospecting rights covering the Waterberg and Waterberg Extension Project of 111,882 ha. The prospecting rights are approximately 40km north south and 40 km east west centered at 23°22'01" south latitude and 28°49'42" east longitude. The project is accessible by paved and dirt roads by vehicle.

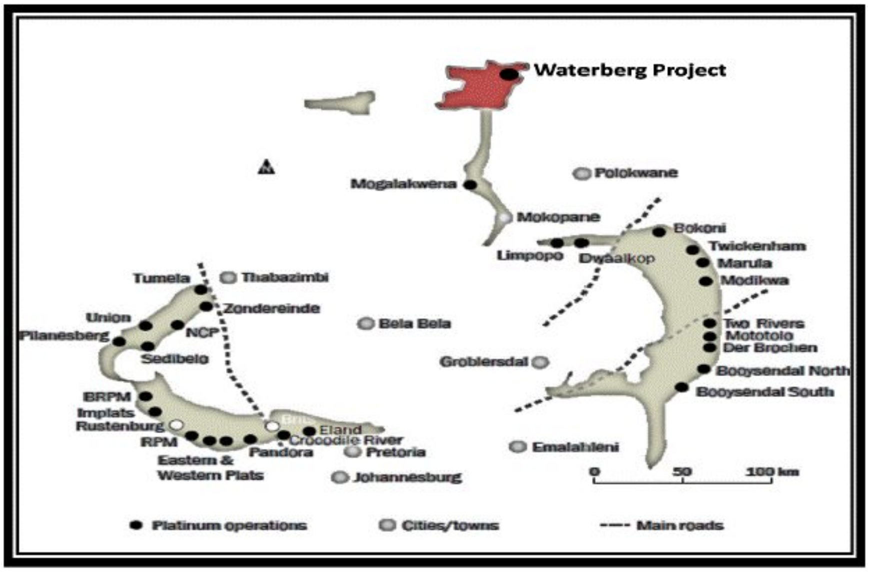


Figure 1-4: Location of Waterberg Project within the Bushveld Complex in the Republic of South Africa

1.4 Geological Setting, Deposit Type and Mineralisation

The Bushveld and Molopo Complexes in the Kaapvaal Craton are two of the most well known mafic/ultramafic layered intrusions in the world. The Bushveld complex was intruded about 2,060 million years ago into rocks of the Transvaal Supergroup, largely along an unconformity between the Magaliesberg quartzite of the Pretoria Group and the overlying Rooiberg felsites. It is estimated to exceed 66,000km ^{2} in extent, of which about 55% is covered by younger formations. The Bushveld Complex hosts several layers rich in Platinum Group Metals (PGM), chromium and vanadium, and constitutes the world's largest known resource of these metals.

The Waterberg Project is situated off the northern end of the previously known Northern Limb, where the mafic rocks have a different sequence to those of the Eastern and Western Limbs.

PGM mineralization within the Bushveld package underlying the Waterberg Project is hosted in two main layers: the T-Zone and the F-Zone.

The T-Zone occurs within the Main Zone just beneath the contact of the overlaying Upper Zone. Although the T-Zone consists of numerous mineralized layers, three potential economical layers were identified, T1, T2HW and T2 layers. They are composed mainly of anorthosite, pegmatoidal gabbros, pyroxenite, troctolite, harzburgite, gabbronorite and norite.

The F-Zone is hosted in a cyclic unit of olivine rich lithologies towards the base of the Main Zone towards the bottom of the Bushveld Complex. This zone consists of alternating units of harzburgite, troctolite and pyroxenites.

The F-Zone was divided into the FH and FP layers. The FH layer has significantly higher volumes of olivine in contrast with the lower lying FP layer, which is predominately pyroxenite. The FH layer is further subdivided into six cyclic units chemically identified by their geochemical signature, especially chrome. The base of these units can also be lithologically identified by a pyroxenite layer.

1.5 Geology

The Waterberg Project is located along the strike extension of the Northern Limb of the Bushveld Complex. The geology consists predominantly of the Bushveld Main Zone gabbros, gabbronorites, norites, pyroxenites and anorthositic rock types with more mafic rock material such as harzburgite and troctolites that partially grade into dunites towards the base of the package. In the southern part of the project area, Bushveld Upper Zone lithologies such as magnetite gabbros and gabbronorites do occur as intersected in drill hole WB001 and WB002. The Lower Magnetite Layer of the Upper Zone was intersected on the south of the project property (Disseldorp) where drill hole WB001 was drilled and intersected a 2.5m thick magnetite band.

On the property, the Bushveld package strikes south-west to northeast with a general dip of34º-38º towards the west is observed from drill hole core for the layered units intersected on Waterberg property within the Bushveld Package (Figure 1-5). However, some structural blocks may be tilted at different angles depending on structural and /or tectonic controls.

The Bushveld Upper Zone is overlain by a 120m to 760m thick Waterberg Group, which is a sedimentary package predominantly, made up of sandstones, and within the project area that sedimentary formations known as the Setlaole and Makgabeng Formations constitute the Waterberg Group. The Waterberg package is flat lying with dip angles ranging from to 2º to 5º. Figure 1-5 gives an overview of interpreted geology for the Waterberg Project.

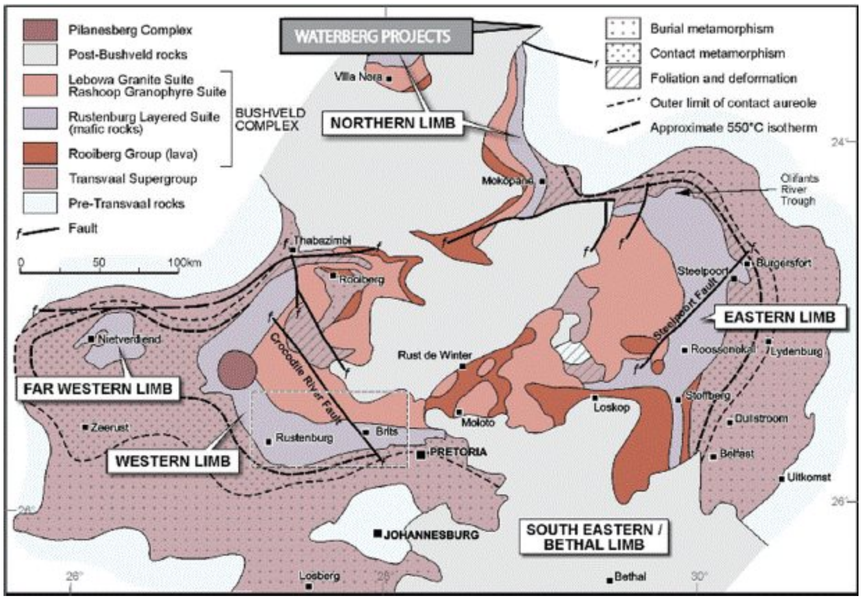


Figure 1-5: Regional Geology

1.6 Exploration Status

The Waterberg Project is at an advanced project that has undergone preliminary economic evaluations, which have warranted further work. Drilling to date has given the confidence to classify Mineral Resources as Inferred and Indicated.

1.7 Sample Preparation

The sampling methodology concurs with PTM protocol based on industry best practice. The quality of the sampling is monitored and supervised by a qualified geologist. The sampling is done in a manner that includes the entire potentially economic unit, with sufficient shoulder sampling to ensure the entire economic zones are assayed.

1.8 Analysis

For the present database, field samples have been analyzed by three different laboratories. The primary laboratory is currently Set Point laboratories (South Africa). Genalysis (Australia) is used for referee test work to confirm the accuracy of the primary laboratory. Analysis was also completed at Bureau Vertitas in Rustenberg.

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

Samples are analyzed for Pt (ppm), Pd (ppm) Rh (ppm) and Au (ppm) by standard 25g lead fire-assay using a silver collector. Rh (ppm) is assayed using the same method but with a palladium collector and only for selected samples. After pre-concentration by fire assay, the resulting solutions are analyzed using ICP-OES (Inductively Coupled Plasma—Optical Emission Spectrometry).

The base metals (copper, nickel, cobalt and chromium) are analyzed using ICP-OES (Inductively Coupled Plasma — Optical Emission Spectrometry) after a multi-acid digestion.

This technique results in “almost” total digestion. The drilling, sampling and analytical aspects of the project are considered to have been undertaken to industry standards. The data is considered reliable and suitable for mineral resource estimation.

The company completes a Quality Control and Assurance review on all of the laboratory samples including a review of the lab quality control samples and the company inserted standards. Issues that are detected beyond acceptable levels are requested for re-analysis.

1.9 Drilling

The data from which the structure of the mineralized horizons was modelled and grade

Values estimated were derived from 298 538m of diamond drilling. This report updates the mineral resource estimate using this dataset. The initial database for this mineral resource estimate was received on July 7, 2016. The raw database consists of 303 drill holes with 483 deflections totaling 300,875 m.

The management of the drilling programmes, logging and sampling has been undertaken from two facilities: one at the town of Marken in Limpopo Province, South Africa and the other on the farm Goedetrouw 366LR within the prospecting right area.

Drilled core is cleaned, de-greased and packed into metal core boxes by the drilling company. The core is collected from the drilling site on a daily basis by PTM personnel and transported to the core yard. Before the core is taken off the drilling site, core recovery and the depths are checked. Core logging is done by hand on a pro-forma sheet by qualified geologists under supervision of the Project Geologist.

1.10 Quality Control and Quality Assurance

PTM have instituted a complete QA/QC programme including the insertion of blanks and certified reference materials as well as referee analyses. The programme is being followed and is considered to be to industry standard. The data is as a result, considered reliable in the opinion of the Qualified Person.

1.11 Mineral Resource Estimate

This report documents the mineral resource estimate - Effective Date: 17 October 2016. The Mineral Resources are reported in the table below. Infill drilling over portions of the project area and new estimation methodology has made it possible to estimate a new mineral resource estimate and upgrade portions of the mineral resource to the Indicated category. The Mineral Resource Statement is summarized below:

Table 1-1: T-Zone Mineral Resource at 2.5g/t 4E Cut-off

T-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade Rh g/t	Grade 4E g/t	Grade Cu %	Grade Ni %	Metal 4E Kg	Metal 4E Moz
Indicated	2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122,375	3.934
Inferred	2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75,485	2.427

Table 1-2: F-Zone Mineral Resource at 2.5g/t 4E Cut-off

F-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade Rh g/t	Grade 4E g/t	Grade Cu %	Grade Ni %	Metal 4E Kg	Metal 4E Moz
Indicated	2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651,670	20.952
Inferred	2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260,484	8.375

4E = platinum Group Elements (Pd+Pt+Rh) and Au The cut-offs for Mineral Resources have been established by a qualified person after a review of potential operating costs and other factors. The Mineral Resources stated above are shown on a 100% basis, that is, for the Waterberg Project as a whole entity. Conversion Factor used — kg to oz = 32.15076. Numbers may not add due to rounding. Resources do not have demonstrated economic viability. A 5% and 7% geological loss have been applied to the indicated and inferred categories respectively. Effective Date Oct 17, 2016. Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance was used for the assessment of Resources; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, Rh, US$984/oz, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15.

The combined Mineral Resource Statement is summarized below:

Table 1-3: Total Mineral Resource at 2.5g/t 4E Cut-off

Waterberg Total 2.5g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade Rh g/t	Grade 4E g/t	Grade Cu %	Grade Ni %	Metal 4E Kg	Metal 4E Moz
Indicated	2.5	218.265	1.06	2.18	0.26	0.04	3.55	0.08	0.15	774,045	24.886
Inferred	2.5	97.212	1.03	2.10	0.30	0.03	3.46	0.06	0.11	335,969	10.802

Mineral Resources at Waterberg on a 100% project basis have decreased to an estimated 10.8 million ounces 4E in the Inferred category but increased to 24.9 million ounces 4E in the Indicated category, from 23.9 million ounces 4E Indicated in April 2016:

1. The Mineral Resources are classified in accordance with the SAMREC standards. There are certain differences with the "CIM Standards on Mineral Resources and Reserves"; however, in this case the QP believes the differences are not material and the standards may be considered the same. Mineral resources that are not mineral reserves do not have demonstrated economic viability and Inferred resources have a high degree of uncertainty.

2. The Mineral Resources and are provided on a 100% project basis and Inferred and Indicated categories are separate and the estimates have an effective date of 17 October 2016.

3. A cut-off grade of 2.5g/t 4E for both the T and the F Zones is applied to the selected base case Mineral Resources. Previously a 2g/t 4E cut-off was applied to the resources.

4. Cut off for the T and the F Zones considered costs, smelter discounts, concentrator recoveries from previous engineering work completed on the property by the Company. The Resource model was cut-off at an arbitrary depth of 1250m, although intercepts of the deposit do occur below this depth.

5. Mineral Resources were completed by Charles Muller of CJM Consulting.

6. Mineral Resources were estimated using Kriging methods for geological domains created in Datamine from 303 original holes and 483 deflections. A process of geological modelling and creation of grade shells using indicating kriging was completed in the estimation process.

7. The estimation of Mineral Resources has taken into account environmental, permitting and legal, title, and taxation, socio-economic, marketing and political factors.

8. The Mineral Resources may be materially affected by metals prices, exchange rates, labor costs, electricity supply issues or many other factors detailed in the Company's Annual Information Form.

The data that formed the basis of the estimate are the drill holes drilled by PTM, which consist of geological logs, the drill hole collars, the downhole surveys and the assay data. The area where each layer was present was delineated after examination of the intersections in the various drill holes.

There is no guarantee that all or any part of the Mineral Resource not included in the current reserves will be upgraded and converted to a Mineral Reserve.

1.12 Mineral Reserves Estimates

The effective date for the mineral Reserve estimate contained in this report is 17 October 2016.

On review by the Qualified Person for Reserves, Robert L Goosen (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves. The final access to the minerals will require permits from the Department of Mineral Resources ("DMR"), acquisition of surface rights, water use license, securing of power and a social license to operate as established in a Social and Labor Plan.

The QPs are not aware of unique characteristics related to this Project that would prevent the granting of such permits and satisfied with progress towards the timing of submission of these applications where applicable. The mineral rights are held under Prospecting Permits with the exclusive right to apply for a Mining Right.

The Mineral Reserve statement for the Waterberg project is based on the South African Code for the Reporting of Exploration Results, Mineral Resource and Mineral Reserves (SAMREC code). There is no material difference between the SAMREC and CIM 2014 code for Mineral Reserve estimation in this case.

Figure 1-6 sets out the framework for classifying tonnage and grade estimates to reflect different levels of geoscientific confidence and the different degrees of technical and economic evaluation. Mineral Resources can be estimated based on geoscientific information with input from relevant disciplines.

Mineral Reserves, which are a modified sub-set of the Indicated and Measured Mineral Resources in order of increasing confidence, are converted into Probable Mineral Reserves and Proven Mineral Reserves (shown within the dashed outline in Figure 1-6), require consideration of factors affecting extraction, including mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors ('modifying factors'), and should in most instances be estimated with input from a range of disciplines.

A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve, which is the economically mineable part of an Indicated Resource, and in some circumstances a Measured Resource. This is demonstrated by at least a Pre-Feasibility Study ("PFS") including adequate information on mining, processing, metallurgical, and economic and other factors that demonstrate, at the time of reporting, the economic extraction can be justified.

A Proven Reserve is the economically mineable part of a Measured Resource demonstrated by the same factors as above. A Proven Mineral Reserve implies that there is a high degree of confidence. Not all mining and permit approvals need be in place for the declaration of Reserves.

Abridged definitions are given below in Section 2.5.

The SAMREC code definition of a Mineral Reserve is:

"A 'Mineral Reserve' is the economically mineable material derived from a Measured, or Indicated Mineral, resource or both. It includes diluting and contaminating materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a Pre-Feasibility Study for a project and a Life of Mine Plan for an operation must have been completed, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors (the modifying factors). Such modifying factors must be disclosed."

Mineral Reserves are reported as inclusive of diluting and contaminating uneconomic and waste material delivered for treatment or dispatched from the mine without treatment.

The CIM 2014 code definition for a Mineral Reserve:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported."

For this technical report, the Mineral Reserves for the Waterberg project have been stated under the SAMREC Code with no material difference to the CIM 2014 standards. The point of reference is ore delivery to the RoM silo at the processing plant.

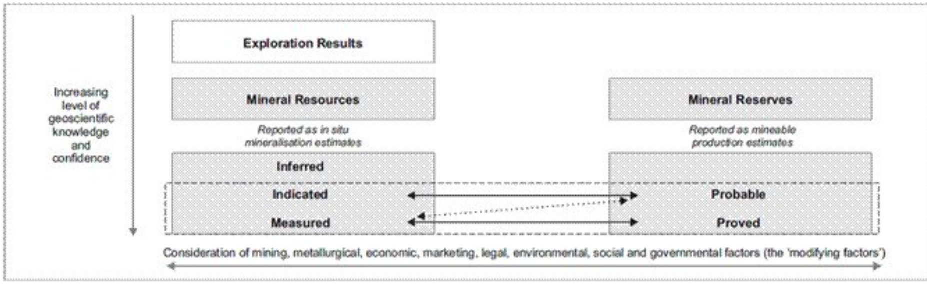


Figure 1-6: Relationship between Mineral Resources and Mineral Reserves

The conversion to Mineral Reserves was undertaken initially at 3.0g/t and the 2.5 g/t 4E stope cut-off grade for both for the T and the F-Zones, which considered costs, smelter discounts, concentrator recoveries from the previous and ongoing engineering work completed on the property by the Company and its independent engineers. Spot and three-year trailing average prices and exchange rates are considered for the cut-off considerations. Initial mine plans were developed based on a 3 g/t 4E cut-off. At the end of the mine life material that was available at a 2.5 g/t 4E cut-off was considered in the full life of mine.

From the Mineral Resource as estimated in this report, each stope has been fully diluted, comprising of a planned dilution and additional dilution for all aspects of the mining process. There are no inferred Mineral Resources included in the Reserves.

The Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian).

Table 1-4 show the Prill splits which are calculated using the individual metal grades reported as a percentage of the total 4E grade.

Table 1-4: Prill Splits

	Prill Split				Grade	
	Pt	Pd	Au	Rh	Cu	Ni
Zone	%	%	%	%	%	%
T-Zone	29	49	21	1	0.16	0.08
F-Zone	30	64	5	1	0.07	0.16

Table 1-5 and Table 1-6 show the total diluted and recovered Probable Mineral Reserve for the Waterberg project.

Table 1-5: Probable Mineral Reserve at 2.5g/t 4E Cut-off — Tonnage and Grades

Waterberg Probable Mineral Reserve — Tonnage and Grades

Zone	Mt	Cut-off grade (g/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)	Cu (%)	Ni (%)
T-Zone	16.5	2.5	1.14	1.93	0.83	0.04	3.94	0.16	0.08
F-Zone	86.2	2.5	1.11	2.36	0.18	0.04	3.69	0.07	0.16
Total	**102.7**	**2.5**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**	**0.08**	**0.15**

Table 1-6: Probable Mineral Reserve at 2.5g/t 4E Cut-off — Contained Metal

Waterberg Probable Mineral Reserve — Contained Metal

Zone	Mt	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	4E (Moz)	4E Content (kg)	Cu (Mlb)	Ni (Mlb
T-Zone	16.5	0.61	1.03	0.44	0.02	2.09	65 097	58.21	29.10
F-Zone	86.2	3.07	6.54	0.51	0.10	10.22	318 007	132.97	303.94
Total	**102.7**	**3.67**	**7.57**	**0.95**	**0.12**	**12.32**	**383 103**	**191.18**	**333.04**

Reasonable prospects of economic extraction were determined with the following assumptions: Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance was used for the assessment of Resources and Reserves; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, US$984/oz Rh, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15. Smelter payability of 85% was estimated for 4E and 73% for Cu and 68% for Ni. The effective date is October 17, 2016. A 2.5 g/t Cut-off was used and checked against a pay-limit calculation. Independent Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian). The mineral reserves may be materially affected by changes in metals prices, exchange rates, labor costs, electricity supply issues or many other factors. See Risk Factors in 43-101 report on www.sedar.com and on the Company's Annual Information Form. The reserves are estimated under SAMREC with no material difference to the CIM 2014 definitions in this case.

1.13 Geotechnical Investigations

1.13.1 Ground Conditions

The site is covered by five identified soil profiles (Kalahari sand, ferruginised Kalahari sand, colluvium, alluvium and strongly cemented calcrete) across the proposed site.

The DCP test results confirm that the transported material layer found from 0.5m below ground level has an allowable bearing capacity of at least 50kPa.

The permanent water table was not encountered during this investigation.

The transported Aeolian material encountered on the site is generally suitable for use in engineered layer work applications. Further testing would be necessary if proposed for use.

Soft to medium hard rock sandstone and strongly cemented calcrete pan can be expected at shallow depth below ground level. Some variation can be expected over the site. Blasting may be required to maintain the lines and levels of services and foundations depending on the design depths.

The sidewalls of the trial pits were relatively stable during the investigations.

1.13.2 Foundations

According to the trial pits/rotary core drilling investigation and the laboratory test results, the site is classified as a "H1/S2/C2/R" site in the NHBRC Classification, with an expected range of total soil movements more than 20mm. The assumed differential movement is 50%.

1.13.2.1 Light Structures* (100 — 150kPa)

Remove the soil to a depth of 1.6m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 150kPa.

1.13.2.2 Medium Structures* (150 — 250kPa)

Remove the soil to a depth of 3m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 250kPa.

1.13.2.3 Heavy Structures* (250 - 500kPa)

Remove the soil to a depth of 4m below surface or up to the bedrock. The excavation must then be backfilled with G5 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 500kPa.

Notes*: Soil raft foundation with good site drainage is recommended. Ninety-three percent compaction is a reasonable expectation. Anything above that might not be achievable during construction. Soil mattresses will have to be found on dense sand (>100kPa) as a minimum.

1.13.2.4 Primary and Secondary Surface Crushers

Spread foundations founded on the bedrock are considered feasible. Allowable bearing capacity of at least 5MPa, which is generally suitable for a crusher structure, was confirmed with the point load test results. The recommended founding level was identified at 4.21m depth below natural ground level in the borehole WB130. Good founding material (medium hard rock sandstone) will have to be validated by a competent person during construction.

1.14 Mine Plan

1.14.1 Geotechnical Factors

Prior to the commencement of the PFS, additional geotechnical data was obtained through core logging of recently drilled boreholes. The revised geotechnical, database, which includes laboratory strength test results, was used to determine rock properties and classify the rock mass. This information was used together with available geological information to construct a 3-dimensional geotechnical rock mass model. The geotechnical rock mass model together with other pertinent information informed aspects of mine design. Input parameters derived from this work were used in idealized numerical models to evaluate various mining configurations and mine sequencing and to augment the empirical evaluations that were conducted.

Some elementary geological interpretations were made to help inform mine design.

The potential for surface displacement resulting from underground mining was assessed with elementary numerical models and it was found that the likelihood of surface subsidence is very low.

The potential for raisebore instability was assessed based on a few boreholes not necessarily near any proposed ventilation raise bore location. There could be challenges, however better informed assessments can only be made based on dedicated geotechnical boreholes at each location.

The two mining methods proposed, BLR and SLOS were assessed and are substantially feasible as long as control is exercised diligently.

Critical hydraulic radii were calculated for open span designs and pillar dimensions were determined based on empirical methods and numerical modelling. In an attempt to optimize extraction, the designs for Waterberg are in a “transition” zone between indefinite stability on the one hand and definite caving on the other.

Based on the rock mass classification and using the Q-system, guidelines for ground support in main access excavations, main and secondary on reef roadways and on reef drifts have been developed.

All the work contributed to the development of a set of rock mechanics parameters for mine design.

Current risks and opportunities to the project associated with mine design have been identified and listed and a set of recommendations for the way forward have been compiled.

1.14.2 Mining Methods selected

The wireframes resulting from the MSO runs were used to create artificial footwall and hanging wall contact zones from which the mine design could be digitized.

Three mining methods Blind Longitudinal Retreat, "BLR" Transverse Sub-level open stoping "TSLOS" and Longitudinal Sub-level open stoping "LSLOS") were selected for the project as they satisfy the following design criteria:

- Minimize the schedule required to achieve full production with stope sequencing;
- Required production volumes;
- Opex/Capex cost;
- Optimize recovery and minimize dilution;
- Maximize flexibility and adaptability based on size, shape, and distribution of target mining areas; and
- Prevent surface subsidence from underground mining.

The criteria for each of these methods are detailed below, but can be resumed by the following table:

Table 1-7: Mining Method Criteria

Mining Method	Dip	Vertical Thickness
BLR	≤ 35°	3 - 15m
LSLOS	> 35°	3 - 15m
TSLOS		> 15m

The MSO wireframes provided the boundaries to which each mining method is applied. These boundaries along with the artificial contact zones were used in Studio 5D Planner to create the detailed mined design.

The design maximized the recovery of material identified from MSO while keeping to geotechnical guidelines proposed by rock engineering, thus all geotechnical losses were designed for and would not require additional factors.

To obtain initial tonnage and grades, the mine design was evaluated against the block model and the results were exported to EPS for scheduling and reporting.

From the Mineable Shape Optimizer model, ore bodies were delineated by resource characteristics and potential mining methods were selected and derived for each defined mining area through a process of option identification and ranking, and adapted to the rock conditions, including:

- Geometry of orebody;
- Geological complexities;
- Geotechnical properties of the country rock and orebody; and
- Depth below surface of extraction.

The mine is designed to initially develop the high-grade zones to minimize pre-production development capital and maximize early revenues. Further optimization for grade is an opportunity with more detailed mine designs in the Definitive Feasibility stage. Final resource to Reserve reconciliation checks was completed. The QP is satisfied with the Reserve data and has verified the data for the Reserve estimate.

1.14.3 Mine Design Access

The top of mining zones in the current Waterberg mine plan occur at depths ranging from 170m to approximately 350m below surface.

The majority of development is done by mechanized equipment on the ore horizon due to the orebody and various mining methods.

Access to the mine will be via three decline shafts, to service the various zones namely:

- T-Zone : Portal Position - South;
- F Central : Portal Position — Central;
- F Boundary and F North : Portal Position — North.

The design philosophy applied to the Waterberg project followed an approach of proven designs and results of various trade-off studies and was designed to accommodate a mine plan, which ramps up to 7.2 Mtpa.

Practical consideration of the real estate purchases and protection of heritage resources were considered in the selection of surface infrastructure.

The study has concluded that the dual decline option has lower capital cost and lower long-term operating costs and provides a more flexible and easily expandable solution for initial mine access and production ramp-up, as well as an opportunity to achieve higher production rates in the event that resource growth is confirmed.

Other key access design objectives met are:

- To access the workings in a way this minimizes capital development; and
- To facilitate an aggressive production build up, targeting the high-grade areas as quickly as possible.

Various ventilation holes from surface will also be required to provide a ventilation egress point.

1.14.3.1 Portal and Declines

Initial access into the mine would be via portals that service the twin declines.

The dimensions of the main access declines are 6.0 m (W) x 6.0 m (H), while the main conveyor declines have dimensions of 5.5 m (W) x 5.5 m (H). The declines will dip at -9°, generally in an easterly direction. Figure 1-7 shows the position of the portals in relation to the surface infrastructure. The dimensions have been based on the conveyor design, ventilation intake requirements and sizes of equipment.

Positioning the portal as shown, will facilitate quick access to the shallower parts of the ore body, which will reduce the time to ‘first ore’. In addition, the portal position allows quick access to the higher-grade areas of the Waterberg mining area.

Portal designs were created based on professional experience in similar ground environment and geotechnical information gathered from the inspection of four boreholes drilled near the proposed portals location.

Laboratory tests were conducted to confirm the on-site investigation and establish preliminary engineering parameters for the soils and rocks.

The suggested preliminary portals designs will have to be supported and approved with the finite element and limit equilibrium methods during the Definitive Feasibility Study (DFS) to reach an acceptable Factor of Safety (FoS) determined for the project.

The proposed portals designs were conducted in a manner consistent with the level of care and skill ordinarily exercised by members of the geotechnical profession practicing under similar conditions in the locality of the project.

- Portals T-Zone and F Central

  The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 37° inclination into soil material. The high wall is 20m high from the footwall position. The overall slope angles are 41° and 50° for the sidewalls and highwall respectively in the preliminary portal design. The top two benches have a height of 4m. The remaining benches are 6m high. The catch berms have a width of 3m across the highwall and sidewalls.

- Portal F North Zone

  The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 36° inclination into soil material. The high wall is 35m high from the footwall position. The overall slope angles are 38° and 44° for the sidewalls and highwall respectively in the preliminary portal design. The first bench has a height of 5m. The remaining benches are 6m high. The catch berms have a width of 3m across the highwall and sidewalls.

  Each mining method requires a different underground infrastructure, such as access development to sub-levels, loading points, ventilation shafts and silos. Together, they form intricate network of openings, drifts, ramps, shafts and slot raises, each with its designated function.

1.14.3.2 Mining Rates

The PTM Waterberg Project requires significant underground development in order to optimally access the ore body. Access to the high-grade areas of the mine is required as soon as reasonably possible in order to attain a maximized potential project value.

A mining cycle scheduling operation, derived from first principles, for cleaning, supporting, drilling and blasting was completed for various mining systems and face arrangements. This was done to test the theoretical possibility of attaining the required 100m per month system advance, which has been planned, whilst not conservative, is a consistently achievable target from both a theoretical and actual benchmarked operations perspective.

There is significant opportunity to increase the planned system advance rate in areas should it be possible to achieve multi-blast conditions during the course of the mine development. This would entail establishing an independent ventilation district that solely ventilates the development and is removed from stoping operations.

Figure 1-7 gives an overview of the portal positions and extent of strike and dip of the orebody.

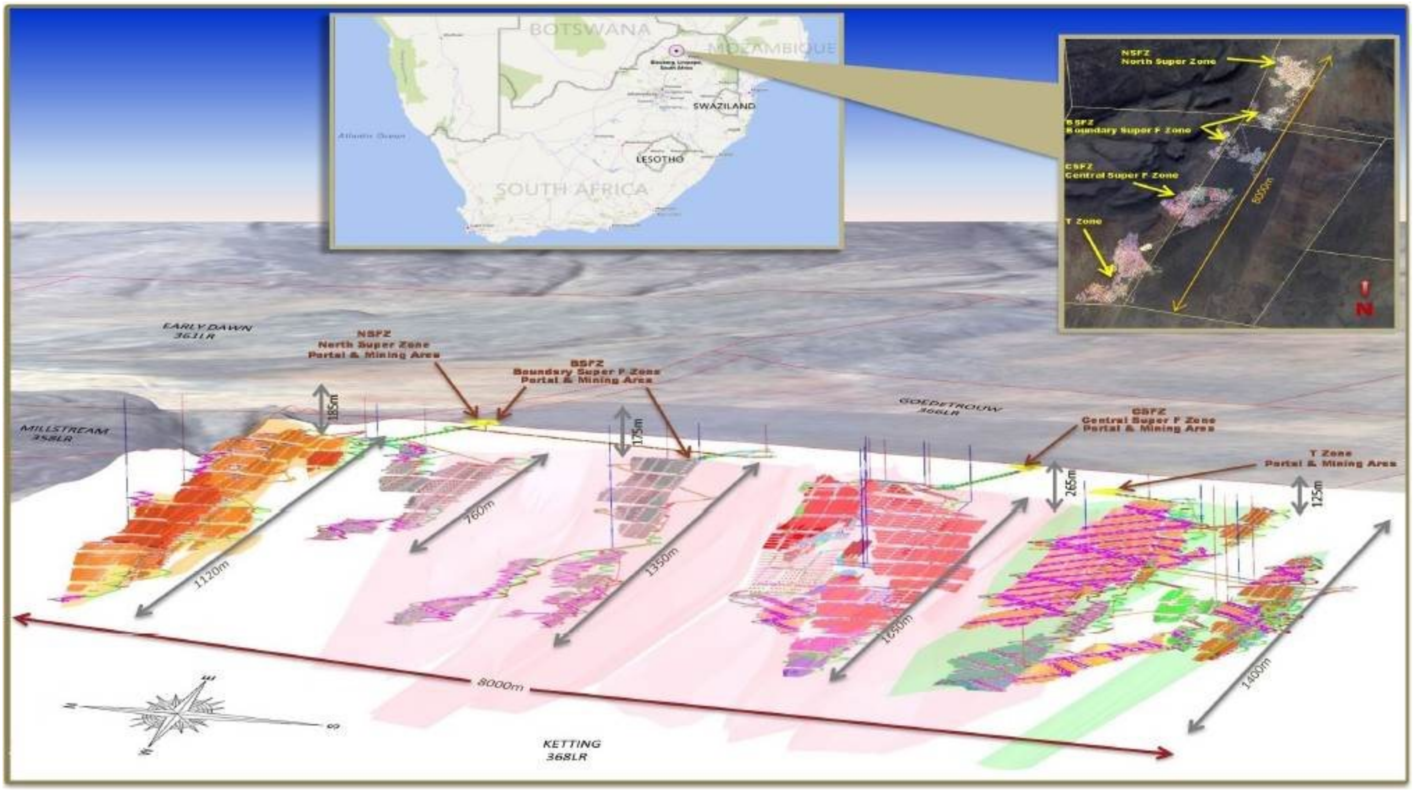


Figure 1-7: Portal and Underground Layouts

1.14.4 Production summary and schedule

The key average annual production results over the 18-year mine life are shown in Table 1-8

Table 1-8: Production Summary

Item	Units	Total
Mined and Processed	Mtpa	7.20
Platinum	g/t	1.11
Palladium	g/t	2.29
Gold	g/t	0.29
Rhodium	g/t	0.04
4E	g/t	3.73
Copper	%	0.08
Nickel	%	0.15
Recoveries		
Platinum	%	82.5%
Palladium	%	83.2%
Gold	%	75.3%
Rhodium	%	59.4%
4E	%	82.1%
Copper	%	87.9%
Nickel	%	48.8%
Concentrate Produced		
Concentrate	ktpa	285
Platinum	g/t	24.2
Palladium	g/t	51.5
Gold	g/t	4.9
Rhodium	g/t	0.6
4E	g/t	81
Copper	%	1.9
Nickel	%	1.8
Recovered Metal in Concentrate		
Platinum	kozpa	222
Palladium	kozpa	472
Gold	kozpa	45
Rhodium	kozpa	6
4E	kozpa	744
Copper	Mlbpa	11
Nickel	Mlbpa	12

Year 4 bases the mine plan on a multiple ramp access underground mining operation ramping up to 600ktpm where it remains for the majority of the LoM until the lower grade end period.

The current status of Life of Mine (LOM) throughput is based on an initial 3g/t 4E cut-off; thereafter, 2.5 g/t 4E will be applied in the final years of the mine life.

The tail of the production schedule for the Waterberg production starts in 2035 and final reef tonnes are scheduled for 2038.

The recommended throughput option for the Waterberg process plant is two modules of 300ktpm each. This configuration is sufficiently flexible to cater for the portal development scenarios and further provides flexibility to cater for both large and small mining operations if selected in future.

Total Mine production with the average grade is shown in Figure 1-8.

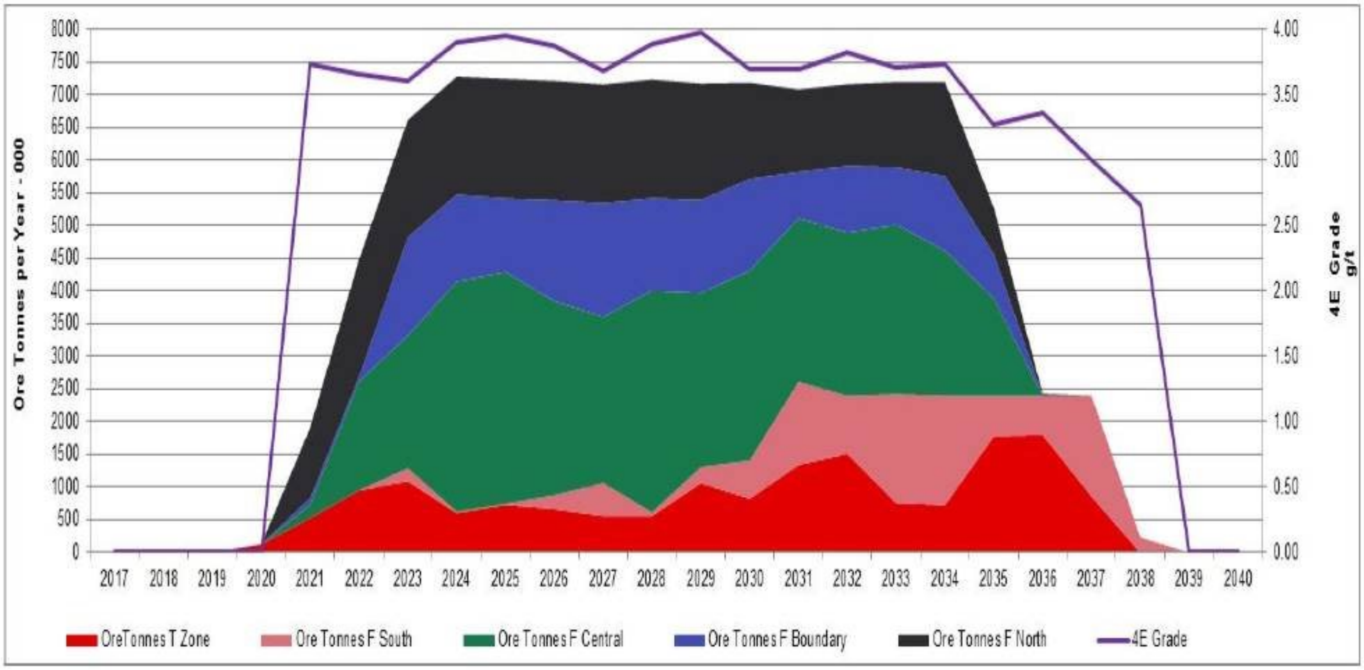


Figure 1-8: Mining Method Total Mine Production

1.14.5 Ventilation

The ventilation and cooling systems consider safety and health requirements in accordance with the Mine Health and Safety Act [MHSA, Act 29 of 1996].

Ventilation and cooling system designs are based on the production and development tonnage profiles and diesel fleet provided by the mine design team. The mining plan is based on steady state production of 600 000 reef tons per month, ventilation and cooling requirements for each mining area is phased-in accordingly over LoM.

Diesel equipment will be a significant heat source accounting for almost 40% of mine heat, in comparison heat flow from rock will account for less than 10% [maximum Virgin Rock Temperature VRT 46.0°C]. The balance will come from auto-compression and other sources including electrical. In mechanized mines, to a depth of approximately 700 mbs this heat can usually be removed by ventilation used to dilute exhaust gasses. However, beyond this depth, heat flowing into the mine from rock and other sources combined with heat from the diesel equipment means that generally, air alone cannot adequately cool the mine and additional mechanical cooling is required. It is confirmed that at depth T-Zone, F1 South, F2 Central, F4 Boundary North and F5 North additional cooling will be required

1.15 Metallurgical Test Work and Recovery

Various metallurgical test work campaigns have been conducted throughout the course of 2013 to 2016 to determine the optimum flowsheet for treatment of the various Waterberg ore lithologies. Metallurgical test work focused on maximizing recovery of PGEs and base metals, mainly copper and nickel, while producing an concentrate product of an acceptable grade for further processing and/or sale to a third party.

In 2013, preliminary metallurgical test work was undertaken at SGS (Booysens, South Africa) using two samples, F-Central and T-zone, taken from the Waterberg deposit as part of the Preliminary Economic Assessment. The results indicated that a potentially saleable concentrate could be produced. The results from the PEA test work program is summarized in the previous PEA technical report, filed in February 2014.

Further investigative test work was performed on an F-Central composite sample, under the management of JOGMEC during the course of 2013 to 2014. The results indicated that a concentrate product in excess of 100 g/t 4E could be produced at acceptable recoveries with the inclusion of Oxalic acid and Thiourea in the reagent suite.

As part of the PFS, extensive metallurgical test work was conducted at MINTEK, which focused on characterizing the various Waterberg lithologies in terms of mineralogical composition, comminution parameters, and flotation response.

Comminution tests have classified the Waterberg ores as hard to very hard and not suitable for Semi-Autogenous Grinding (SAG) milling.

Two flotation flowsheets were tested on each Waterberg lithology, a MF1 circuit utilizing Oxalic acid and Thiourea as part of the reagent suite and a MF2 circuit utilizing typical Southern African PGM reagents, such as SIBX as a collector. Batch open circuit flotation test work as well as locked cycle flotation test work was conducted. Encouraging results were obtained from both flowsheets. Test work results have demonstrated that some of the ore types respond better to a particular configuration. However, superior recoveries were obtained for the mine blend samples using the MF2 configuration, leading to the selection of the MF2 circuit for the process design.

It was noted that extensive scavenging and cleaning was required in the MF2 circuit to maximize recoveries, while lower mass pulls in the high grade and low grade circuits where essential to ensure acceptable concentrate grades were achieved and the product grade specification were met. Flotation work indicated that the optimum final grind for the F-zone ores are 80% passing 75µm; whilst there are evidence that the T-zone material could achieve higher recoveries at finer grinds of 85-90% passing 75µm. Further test work to investigate the optimization of the T-zone final grind is recommended.

The flotation test work indicated that the Waterberg ores are amenable to treatment by conventional flotation without the need for re-grinding. A standard flotation concentrator can be used to produce a saleable concentrate, at a 4E grade of no less than 80 g/t, with no deleterious products. 4E recoveries in excess of 80% are expected at the proposed mill feed grades.

1.16 Process Plant Design

The process design for the Waterberg Concentrator Plant has been developed based on the extensive metallurgical test work results, as well as other desktop level studies completed by the project team. A trade off study was conducted to determine the optimal production ramp up and steady state production. Based on the outcome of the study the plant steady state capacity of 7.2Mtpa will be achieved by the construction of the plant in two phases. Each phase consisting of a 3.6Mtpa concentrator module,

The Phase 1 3.6 Mtpa concentrator module and associated infrastructure, is planned to start production in month 36. Phase 2 includes the construction of the second 3.6 Mtpa module to take the total production to 7.2 Mtpa in month 53. The second concentrator module is designed as a copy of the first module, with minor exceptions with regards to shared infrastructure.

Each of these modules comprises a three-stage crushing circuit, feeding crushed material to the primary milling circuits. Primary milling is achieved in a ball mill with closed-circuit classification followed by a primary rougher flotation bank. The primary rougher concentrate is further upgraded in the primary cleaning/re-cleaning circuit to produce a high grade concentrate product. The primary rougher tailings is further liberated in the secondary milling circuit which consist of a ball mill with closed-circuit classification, before reporting to the secondary rougher and scavenger flotation circuit. The secondary rougher concentrate product reports to the secondary cleaning/re-cleaning stages to produce a medium grade concentrate, whilst the scavenger flotation concentrate is upgraded in the scavenger cleaning circuit to produce a low grade concentrate product. Each of the concentrate products are combined in the concentrate thickener for dewatering, followed by filtration. The flotation tailings products are thickened prior to beings disposed to the residue storage facility.

Refer to Figure 1-9 for an illustration of the above.

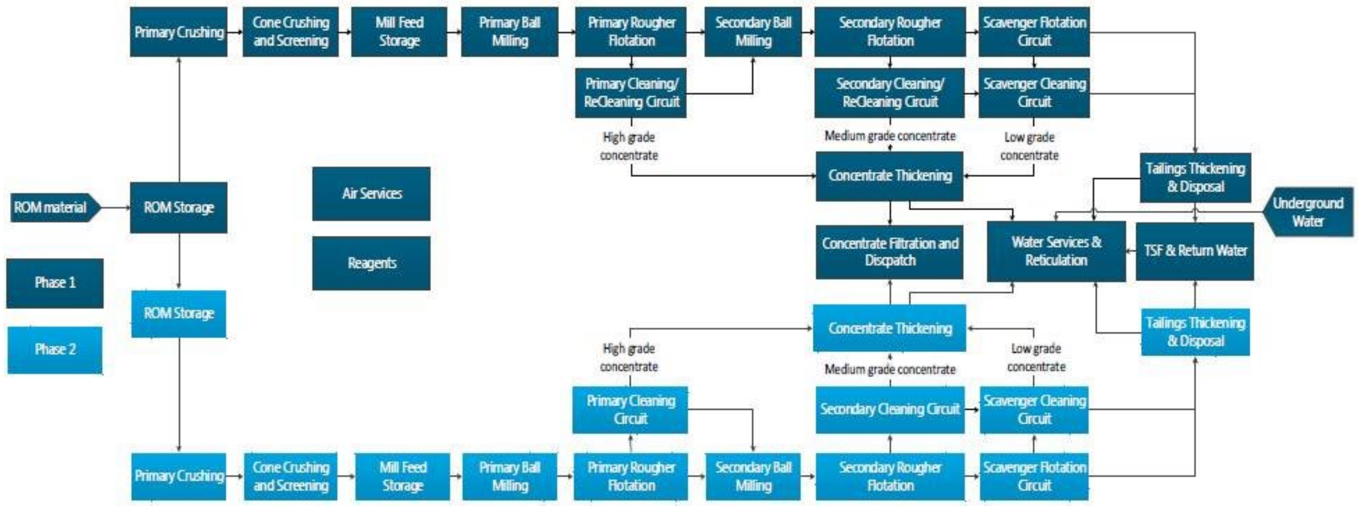


Figure 1-9: Waterberg Concentrator Block Flow Diagram

1.17 Infrastructure

The design philosophy applied to the Waterberg project followed an approach of proven designs and results of various trade-off studies.

The infrastructure was designed to accommodate a mine plan, which ramps up to 7.2Mtpa. Locations and sizing of infrastructures were significantly influenced by the geographical area. Real estate associated with cost, social, and cultural heritage considerations allowed little leeway for selection of locations. A site layout plan covering site facilities is shown in Figure 1-10.

The key infrastructure includes regional infrastructure, local infrastructure, central shared services, portal infrastructure as well as mine ventilation and refrigeration surface infrastructure as described in Section 18.

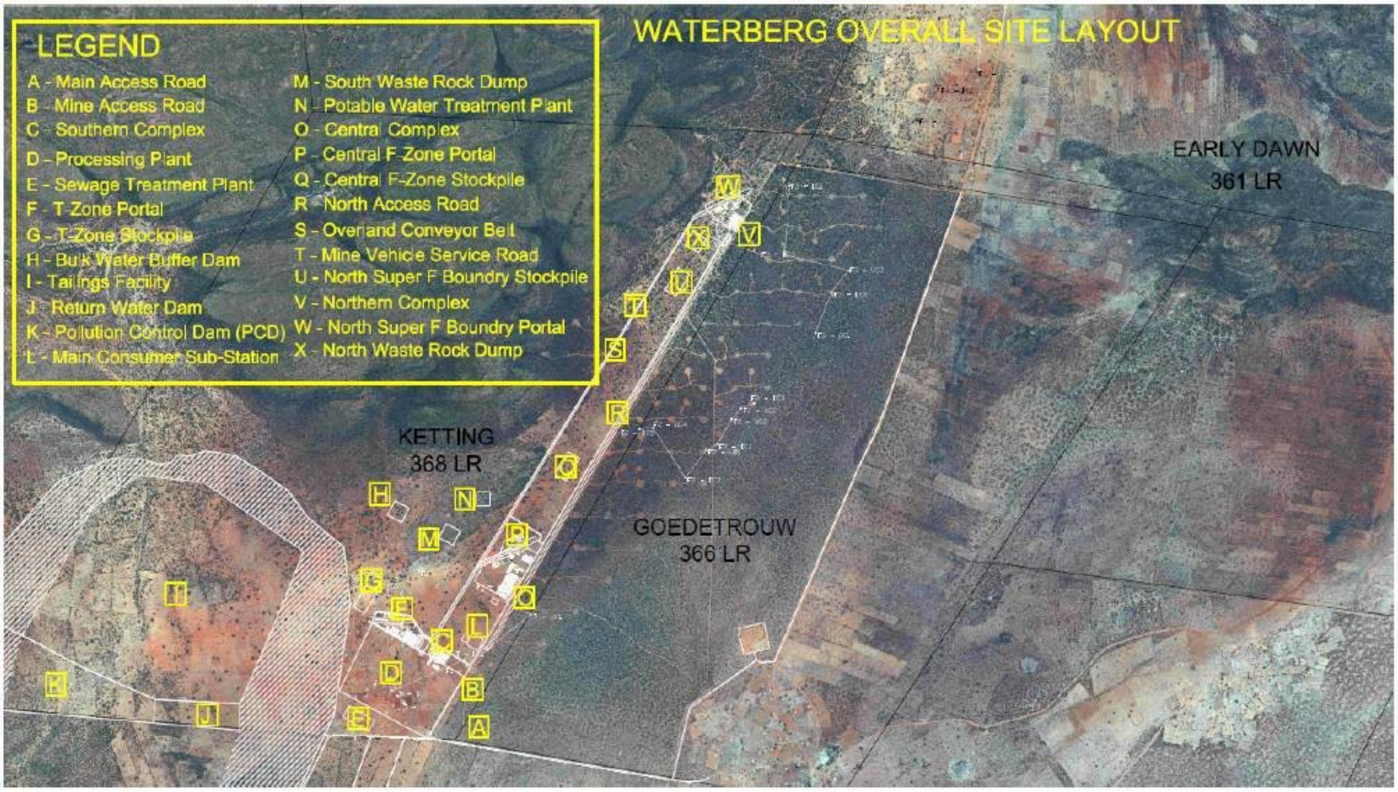


Figure 1-10: General Site Layout

1.17.1 Bulk Water Supply

South Africa is a country of relatively low rainfall and, in particular, the Limpopo province will require significant additional water capacity to meet the growing demand from the mining, agricultural, and domestic sectors. The Government has committed to addressing this shortage in the interest of developing the region. However, there are major planning, infrastructural design, and funding challenges that need to be addressed in order to ensure that sufficient bulk water supply is achieved.

The Olifants River Water Resource Development Project (ORWRDP) has been designed to deliver water to the Eastern Limb and Northern Limb of the Bushveld Igneous Complex (BIC) of South Africa. The ORWRDP consists of the new De Hoop Dam, the raising of the wall of the Flag Boshielo Dam, and related pipeline infrastructure, which will ultimately deliver water via Pruissen to Sekuruwe, located some 30kms to the north of Mokopane and 60kms south of PTM Waterberg Project. From this point, PTM Waterberg will need to develop their own pipeline project to take water to their site.

Implementation of the Flag Boshielo Pruizen pipeline has been put on hold because of funding issues and withdrawal of commitments from some mines due to low commodity prices. The PTM Waterberg project is located on the northern extremity of the ORWRDP area, the delay in implementation will result in PTM Waterberg not meeting their development schedule, and other options would need to be considered.

During the Pre-Feasibility Study, other bulk water supply options were considered. Other options considered were Glen Alpine Dam, transfer of water from Lephalala River, groundwater and effluent from various Waste Water Treatment Works (WWTW) including Louis Trichardt / Makhado and Seshego. The present water balance model simulations showed that the average bulk water supply requirement over the life of the mine would be 10.6 Ml/d.

Of all the water supply options considered a combination of sewage effluent and groundwater is considered the most viable and least risk solution to meet the proposed mining schedule. Wellfields with mainly poor water quality will be targeted so as not to compete with domestic water uses in the area.

From existing borehole information and limited exploration, drilling done to date about 0.5Ml/day of potable water or more could be developed around the mine site. Poor quality groundwater developed within 35kms east of the mine towards Bochum (about 5,5Ml/day) and to the south of the mine, some 4.3Ml/day is thought to be available. Non-potable groundwater resources up to 35kms from the mine could yield up to 9.9Ml/day.

1.17.2 Ground Water

The PTM Waterberg Project site and surrounding area is underlain by the Waterberg Group, Bushveld Igneous Complex and the Archaean Granite/Gneiss rocks. The Waterberg Group overlies the Bushveld Igneous Complex and comprise predominantly of sandstones. The base of the Bushveld Main Zone is characterized by the presence of a transitional zone that constitutes a mixed zone of Bushveld and altered sediments/quartzites before intersecting the Archaean granite basement. The Waterberg Sedimentary package has been intersected by numerous crisscrossing dolerite or granodiorite sills or dykes and act as preferential flow path for groundwater.

Groundwater abstraction in the area is mainly used for domestic consumption at the villages. Water levels in the area vary between artesian and 52m below ground level (mbgl). The groundwater quality does not always comply with the drinking water standards due mainly to the high salt content. Borehole yields vary considerably over the area with yields of up to 10l/s found along major structures in the Waterberg sediments and in the highly weathered and fractured Gneisses. However, due to the low rainfall, recharge to the aquifers is low with the average annual recharge estimated to be only about 12mm per annum.

Inflow into the proposed mine workings has been estimated to be between 3.6Ml/day and 9.4Ml/day depending on hydraulic conductivity of the deeper fault zones and the number of faults intersected. A conservative figure of 3.3Ml/day has been used in the water balance. These inflows will result in an impact zone around the mining lease area of about 6kms. Production boreholes serving communities within this zone could be affected.

From information available at this stage local groundwater around the mine could yield up to 0.5 Ml/day of potable water or more. Non-potable groundwater resources up to 35kms from the mine could yield up to 9.9Ml/day.

1.17.3 Bulk Power supply

The bulk electricity supply for the project is being planned to cater for mining and plant production rates of up to 600ktpm, which correspond to an electrical load of up to 160MVA. A temporary electrical supply is being planned for the construction stage.

Existing 66kV and 132kV networks approach to within 25km from the project site, however, it has been determined that the capacities of these networks are inadequate to supply the project load. The updated electricity supply plan compiled by Eskom therefore provides for the establishment of new 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation, which is located approximately 77km south of the project site. Eskom has confirmed in principle the availability of capacity from this system to supply the mine.

The proposed bulk electricity supply infrastructure comprises the following:

- Two 77km long 132kV overhead lines from Burotho transmission substation;
- Two 132kV line feeder bays for these new lines at Burotho transmission substation; and
- A 132kV switching substation and step-down substation located on the project site.

The development of the abovementioned infrastructure is being done in conjunction with Eskom on a Self-Build basis in terms of which Waterberg JV Resources is responsible for most of the development work.

This work is already in an advanced stage; with line route planning and environmental impact assessment work having progressed well (refer Figure 1-11, which shows some of the 132kV overhead line route options).

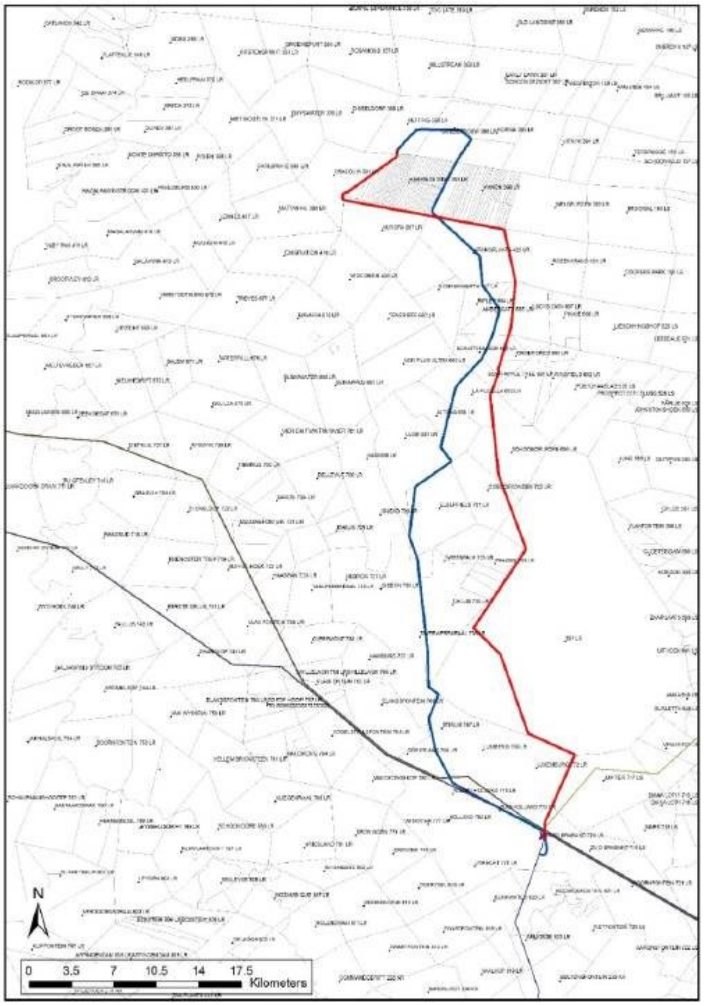


Figure 1-11: Proposed Overhead Line Route

1.17.4 Process Plant

Further to the equipment described in Section 1.16, the following permanent installations are also included to support the processing plant:

- Return water columns from the residue storage facility to the processing plant
- Plant services, i.e. compressed air and raw water
- Plant potable water storage and reticulation
- Plant electrical supply and reticulation, from the plant consumer substation.
- Plant offices
- Plant store
- Plant workshop
- Plant weighbridge

The plant infrastructure includes storm water berms and drains to divert rainwater from the plant and to collection rainwater falling in the plant in a pollution control dam, this water will be captured for use in the process plant and not intended to be discharged to the environment.

1.17.5 Residue Storage facility

A Pre-Feasibility Design (PFD) of the Residue Disposal Facility (RDF) and its associated infrastructure was undertaken. The design of the RDF comprising:

- A Residue Storage Facility (RDF) that accommodates 140 000 000 dry tonnes over a 20 year Life of Mine (LoM);
- A Return Water Dam (RWD) and/or Storm Water Dam (SWD associated with the RDF;
- The associated infrastructure for the RDF (i.e. perimeter slurry deposition pipeline, storm water diversion trenches, perimeter access road, etc.);
- Estimation of the capital costs to an accuracy of ±25%, operating costs associated with these facilities to an accuracy of ±25% and closure costs to an accuracy of ±35%; and
- Estimation of the costs over the life of the facility.

1.17.5.1 Site Selection

A site selection study was undertaken to find the most favorable site. The study found that Ketting farm was the most favorable.

1.17.5.2 Depositional Trade-off Study

A trade-off study was undertaken to determine a suitable depositional methodology as well as to highlight the advantages and disadvantages of each methodology. The following methodologies were investigated:

- Conventional/thickened tailings;
- Cycloned tailings;
- Paste tailings; and
- Dry-filtered tailings.

The following conclusions were drawn from the study:

- Paste disposal is untested in the platinum industry and would pose a significant risk and require an extensive testing regime to consider implementing;
- Dry stacking is a possible option and the potential water recoveries could make this option feasible, however the high capital and operational costs associated with dry stacking could make this potion infeasible compared to a conventional tailings dam;
- Cycloned tailings may provide a cost saving due to the higher rates of rise achievable, however test work is required prior to recommending this option;
- Conventional/thickened tailings are the safest option, well understood in the platinum industry, and have been regarded as the preferred option for Waterberg.

1.17.5.3 Economic Depositional Methodology Trade-off Assessment

Further to this, an Economic Assessment of the various depositional methodologies was undertaken to determine which methodology would provide a cost effective solution given that the scarcity of water at the site. The purpose of this assessment was to determine which option would result in the most cost effective solution in terms of water cost; therefore, the costs were only taken to a conceptual level. The results show that filtered tailings will only be feasible if the water cost exceeds R60/m^3.

Therefore, conventional/thickened tailings were taken forward as the preferred option for Waterberg.

- Key Design Features:

  The key design features of the RDF in Figure 1-12 are as follows:

  - The RDF will be constructed as an upstream, spigotting facility;
  - A compacted earth fill starter wall at elevation 1000m.a.m.s.l.;
  - A penstock system will be used to decant water from the RDF;
  - A RWD with sufficient capacity for the 1 in 50 year storm event (340 000m^3);
  - The RDF has a total footprint area of 297Ha, a maximum height of 55m and a final rate of rise of <3m/year;
  - A concrete lined solution trench to convey seepage water to the RWD;
  - Lined toe paddocks to collect contaminated run-off water from the RDF side slopes; and
  - A slurry spigot pipeline along the crest of the RDF.

Figure 1-12: RDF Layout

1.17.6 Access Roads

The Waterberg Project is located some 85km north of the town of Mokopane (formerly Potgietersrus) in Seshego and Mokerong, districts of the Limpopo Province. Although the bulk of the roads surrounding the site are provincial roads under the jurisdiction of the Roads Agency Limpopo (RAL), some of the minor roads are the responsibility of either the Capricorn District Municipality or the three relevant Local Municipalities.

The Waterberg Project is situated some 56km from the N11 national road that links Mokopane with the Groblers Bridge border post to Botswana. Access to the project area from Mokopane in Figure 1-13 (112km), and Polokwane in Figure 1-14 (94km) includes about 32km of unpaved roads.

It has been assumed in this study that this portion of the access route will remain unsurfaced but provision has been made for re-profiling and adequate drainage run-off along the route and a maintenance contract to maintain the road to an acceptable standard for the life of mine.

The balance of the route will have to be assessed to determine additional costs that may be incurred to upgrade and repair. The transport of the concentrate has been assumed to be done by contract haul and a rate per tonne component has been included in the financial model.

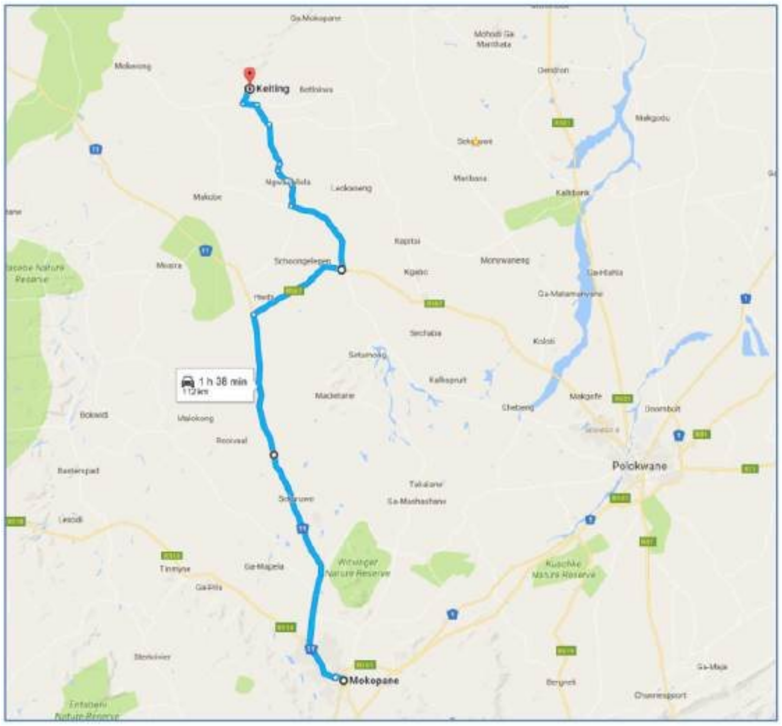


Figure 1-13: Access Route from Mokopane (112km)

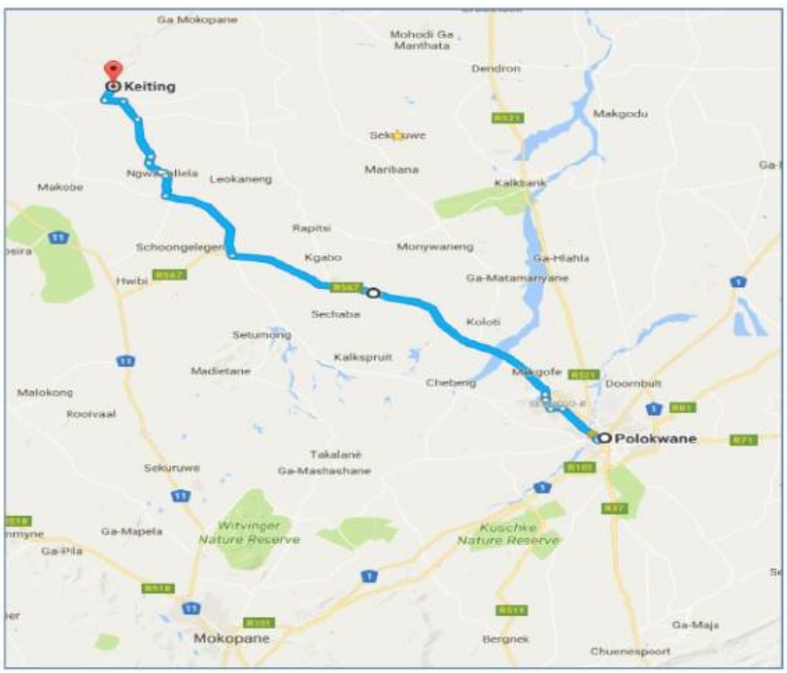


Figure 1-14: Access Route from Polokwane (94km)

1.18 Market Studies and Contracts

Either the Waterberg project will produce a flotation concentrate from the processing plant, which is assumed to be sold, or toll treated into the local South African market.

Production of up to 285 000 tonnes of concentrate per annum will be available at peak production. The concentrate will contain approximately 80g/t 4E's plus copper at between 1% and 9.2% and nickel at between 1.1% and 5%.The concentrate does not contain any penalty elements such as chrome and is rich in Sulphur, thus making it a desirable concentrate to blend with other high chrome concentrates.

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal indications from smelters are that the concentrate is attractive.

Based upon industry data, it is expected that the payability for the concentrate sold to a local smelter operator will be up to 85% for the PGE's, 73% for contained copper and 68% for contained nickel. It is expected that the metal will be available from the refinery after 16 weeks. Opportunity exists to have payment terms with "pipeline' finance facilities and these have been included in the study for the life of the mine.

1.18.1 Metal Prices

The Waterberg Project level financial model begins on 1 July 2016. It is presented in 2016 constant dollars, cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken. The base case real discount factor applied to the analyses is 8%. No allowance for inflation has been made in the analyses.

The following prices, based on a 3-year trailing average in accordance with U.S. Securities and Exchange Commission ("SEC") guidance, was used for the assessment of resources and Reserves.

The exchange rate between the ZAR and the USD is fixed at ZAR15.00:USD1.00 in the financial model throughout the LoM. The pricing and exchange rates above results in the estimated basket prices shown in Table 1-9 below.

Table 1-9: Average Three Year Trailing Metal Prices used in Financial Model

Parameter	Unit	Financial Analysis Assumptions
3 Year Trailing Average Price (Date: 31 July 2016)		
Platinum	US$/oz.	1 212
Palladium	US$/oz.	710
Gold	US$/oz.	1 229
Rhodium	US$/oz.	984
Nickel	US$/lb	6.10
Copper	US$/lb	2.56
Base Metals Refining Charge	% Gross Sales pay	85%
Copper Refining Charge	% Gross Sales pay	73%
Nickel Refining Charge	% Gross Sales pay	68%

Parameter	Unit	Financial Analysis Assumptions
Investment Bank Consensus Price (Date: 16 September 2016)		
Platinum	US$/oz.	1 213
Palladium	US$/oz.	800
Gold	US$/oz.	1 300
Rhodium	US$/oz.	1 000
Nickel	US$/lb	7.50
Copper	US$/lb	2.90

Investment Bank Consensus Sept, 2016 PGMs and base metals.

1.19 Environmental and Impact Assessment Studies

Preliminary environmental baseline studies has been completed for the Waterberg Project and measures have been incorporated in the development of the layouts, designs and operational practices to mitigate potential environmental risks.

The baseline studies included the following:

- Ground Water.
- Air Quality.
- Noise.
- Bio-Diversity.
- Soil.
- Visual Impact.
- Heritage Impact.
- Surface Water.
- Traffic.
- Blasting.

Prior to construction and operation of an underground mine, the following local legislative authorizations would be required:

- In support of a Mining Right Application (MRA), authorization in terms of Section 22 of the Mineral and Petroleum Resources Development Act, 2002 (Act No. 28 of 2002) (MPRD Act) by the Department of Mineral Resources (DMR) is required.
- Environmental Authorization as per the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEMA) and the Environmental Impact Assessment (EIA) Regulations (GNR. 543, 544 and 545 of 18 June 2010) from the Limpopo Department of Economic Development, Environment and Tourism (LEDET).
- A water use license in terms of Section 21 of the National Water Act, 1998 (Act No. 36 of 1998) from the Department of Water and Sanitation (DWS).
- A Waste Management License for categorized waste activities in terms of the National Environmental Management Waste Act, 2008 (Act No. 59 of 2008) (NEMWA) from the National Department of Environmental Affairs (DEA).

There have been discussions with the local communities and stakeholders regarding the environmental protection measures proposed to be undertaken.

The communities that are located within a 5km radius from the proposed project site are:

- Ga-Ngwepe.
- Setlaole.
- Ga-Masekwa.
- Ga-Raweshe.
- Ketting.

Consultations have also been held with the Regulatory Departments on various aspects of the Project and detailed discussions will continue throughout the permitting process and project execution.

A project risk assessment was carried out as part of the Pre-Feasibility Study to identify environmental sensitivities. The key risks potentially affecting the achievement of the project objectives were identified, together with their root causes and potential consequences. Primary mitigating strategies currently in place to address the risks were documented and where the current risk rating was considered unacceptably high, additional action items agreed to reduce it to an acceptable level.

1.20 Community Social Impact Assessment Studies

A social impact assessment is being conducted with the local communities to establish the social understanding within the area of the Waterberg mining operations. The project has maintained a positive open working relationship with the small communities in the area of the project including regular well documented meetings.

The communities that are located within a 5km radius from the proposed prospecting site are Ga-Ngwepe, Setlaole, Ga-Masekwa, Ga-Raweshe, and Ketting.

1.21 Capital and Operating Costs

1.21.1 Capital Costs

Project capital costs total ZAR 27,374M, consisting of the following:

- Initial Capital Costs — includes all costs to develop the property to a sustainable production of 600ktpm. Initial capital costs total ZAR 15,906M and are expended over a 72 month period from January 2017 to Dec 2022 including the pre-production construction and commissioning period; and
- Sustaining Capital Costs — includes all costs over the 16-year mine life related to expansion of production from the initial 300ktpm to 600ktpm and the acquisition, replacement, or major overhaul of assets required to sustain operations. Sustaining capital costs total ZAR 11,468 M and are expended in operating years from Jan 2023 to Jul 2038.
- The peak funding required for the project is estimated at ZAR13,694M (US$914M) in year 2022.

The costs are presented in ZAR 2016 and United States dollars (USD) market terms. It is presented in real money terms and no escalation was added. The base date for the Capital Estimate shall be 31 July 2016 and will be used to qualify the estimate in terms of governing laws, duties, taxes and tariffs.

The exchange rate between the ZAR and the USD will be fixed at ZAR15.00:USD1.00 in the Financial Model throughout the LoM.

The expected order of accuracy of the final estimate is in the range of ±25%

A 12% contingency allowance has been based on an assessment of the risk around the accuracy of the design information, quantities and rates applied using a Monte Carlo statistic process.

The estimate is presented in such a way that it is seamlessly incorporated into the financial model as an input, expressed in monthly cash flows for each WBS Level 1 facility code. Table 1-10 presents the PTM Waterberg capital at Level 1 WBS facility code.

Table 1-10: Total CAPEX

Facility Code	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
2000	Underground Mining	6,092	9,766	406	651
3000	Concentrator	2,850	159	190	11
4000	Shared Services & Infrastructure	1,063	43	71	3
5000	Regional Infrastructure	2,566	—	171	—
6000	Site Support Services	691	67	46	4
7000	Project Delivery Management	1,399	147	93	10
8000	Other Capitalised Costs	246	83	16	6
9000	Contingency	999	1,202	67	80
Total Capital		**15,906**	**11,468**	**1,060**	**765**

The facility level summary of the capital as well as the capital expenditure for LOM is depicted in Figure 1-15.

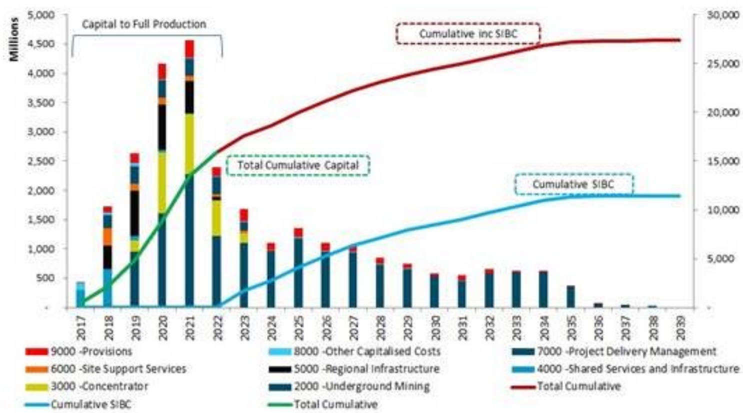


Figure 1-15: Total CAPEX Cashflow

1.21.2 Operating Costs

For the study, OPEX has been defined as:

- All on-reef development as soon as first stoping tonnes are achieved,
- Off-reef development associated with ongoing access and Reserve generation within, when first stoping tonnes are achieved. (These include sub-level off reef, lateral ventilation and other access development),
- All ongoing production related activities after first stoping ore is mined,
- Operating costs associated with the mobile mining equipment and fixed engineering equipment,
- Maintenance of mobile mining equipment and fixed engineering equipment.

Initially the mine will be contractor operated and once first stoping ore is mined for a particular mining zone, it will become owner operated. This excludes some contracted services over LoM such as raise bore, ventilation raises, silo and vertical dams, main access, primary conveyor decline and material decline development. The RDF facility will also be contracted out. The owner-mined operation per zone will coincide with when operating costs starts being incurred. All costs not associated to a particular mining zone will be reported under shared services and will include general, administration, and processing cost.

The operating cost model was developed by following the typical steps and processes prescribed by the Advisian RSA OPEX Estimation standards and methodologies. Methodologies utilized includes first principle costing for the labor, lifecycle costing for all equipment, infrastructure and fleet, zero-based costing for mining consumables and fixed/variable costing for the remainder of operating cost items.

The estimate methodology is aligned to preliminary engineering designs and budgetary quotations for major equipment and consumable cost and conforms to the +-25% accuracy level of a Pre-Feasibility Study. The operating cost estimate is modelled annually in ZAR. Costs reported in USD were converted from ZAR by using and exchange rate of R 15 per USD. A base date of July 2016 was used as costing basis. Costs are reported in real money terms with no escalations or contingency modelled. Quotes and cost rates were sourced from South African suppliers with foreign component cost not having an impact on the operating costs estimate.

The average LoM operating cost for the Waterberg Pre-Feasibility Study project is estimated at R 574.62 per ore tonnes broken (USD 38.31 /t). As indicated in Table 1-11, the total LoM cost amounts to R 58,99 billion (USD 3,93 billion). Average LoM costs are also detailed on a high level per area in ZAR and USD.

Table 1-11: Average LoM Operating Cost Rates and Totals per Area in ZAR and USD

		Average LOM (ZAR/t)		**Total LOM (ZAR M)**		**Average LOM (USD/t)**		**Total LOM (USD)**
Mining	R	271.90	R	27 915	$	18.13	$	1 861
Engineering & Infrastructure	R	107.49	R	11 036	$	7.17	$	736
General & Admin	R	40.71	R	4 180	$	2.71	$	279
Process	R	154.52	R	15 864	$	10.30	$	1 058
Total OPEX Cost	**R**	**574.62**	**R**	**58 994**	**$**	**38.31**	**$**	**3 933**

The information in the table above is visually represented in Figure 1-16 to provide a better understanding of the breakdown per area of the LoM operating cost.

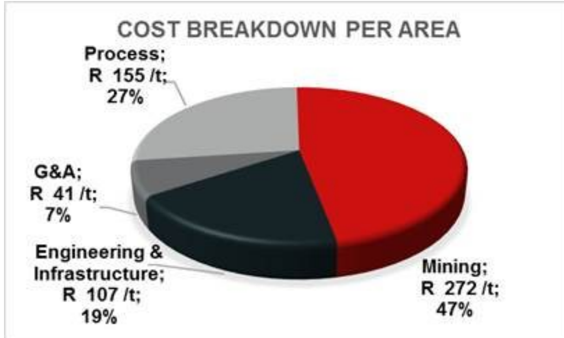


Figure 1-16: LoM Average R/t Operating Cost Breakdown per Area

From the figure, it is evident that mining comprise the bulk of the operating cost at 47%, followed by process at 27% and engineering and infrastructure at 19%. General and administration cost contributes a small portion (7%) of the total operating cost. The mining cost mostly driven by the large materials and supplies cost which is associated to development and production fleet maintenance (R 87/t) and consumables such as fuel (R 30/t). The process cost can be mostly attributed to the high power cost at R 64/t and consumable costs at R 60/t.

Figure 1-17 provides an overview of the operating cost per cost category over LoM. From the graphical representation, it is evident that the majority of costs remain constant. As expected, materials and supplies, cost will vary, as it is the directly related to the production profile.

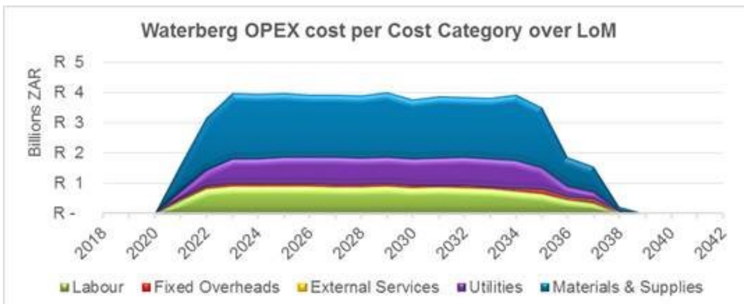


Figure 1-17: Operating Cost broken down per Cost Category over LoM

Figure 1-18 presents the total operating costs over LoM overlaid with the ore tonnage profile. The cost increase observed in 2022 is due to the start of the second process plant in November 2022 (month 53) combined with an increase in tonnage. Steady state is observed around 2024 when the process plant will process 7,2 Mtpa. The process, general, administration, engineering, and infrastructure operating cost remain constant throughout the LoM, whilst the mining operating cost closely resembles the tonnage profile. The two-phased ramp down starting in year 2035 is clearly visible towards the end of LoM.

The dip in operating cost displayed in year 2036 is a result of only one process plant being operational to process 200 ktpm for duration of approximately 17 months, until ore tonnes are depleted.

The operating cost model was developed to enable reporting per zone (e.g. F South), per area (e.g. mining) and per cost category (e.g. labor).

Figure 1-18: Operating Cost per Zone over LoM relative to Ore Tonnes

The operating cost model was developed to enable reporting per zone (e.g. F South), per area (e.g. mining) and per cost category (e.g. labor). For more operating cost detail and results, refer to Section 21.3.

1.22 Summary of Economic Analysis

The results of the financial analysis show an After Tax NPV 8% of ZAR4,805M. The case exhibits an after tax IRR of 13.5% and a payback period of around eleven years. The estimates of cash flows have been prepared on a real basis as at 1 July 2016 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 1-12.

The cumulative cash flow after tax is depicted in Figure 1-19.

Table 1-12: Financial Results Base Case Three Year Trailing Average

Item	Discount Rate	ZAR Millions (Before Taxation)	ZAR Millions (After Taxation)	USD Millions (Before Taxation)	USD Millions (After Taxation)
	Undiscounted	36,096	25,042	2,406	1,669
	4.0%	18,213	11,883	1,214	792
	6.0%	12,666	7,808	844	520
Net Present Value	8.0%	8,565	4,805	571	320
	10.0%	5,519	2,584	368	172
	12.0%	3,249	939	217	62
	14.0%	1,555	-278	104	-19
Internal Rate of Return		16.6%	13.5%	16.6%	13.5%
Project Payback Period (Years)		10	10	10	10

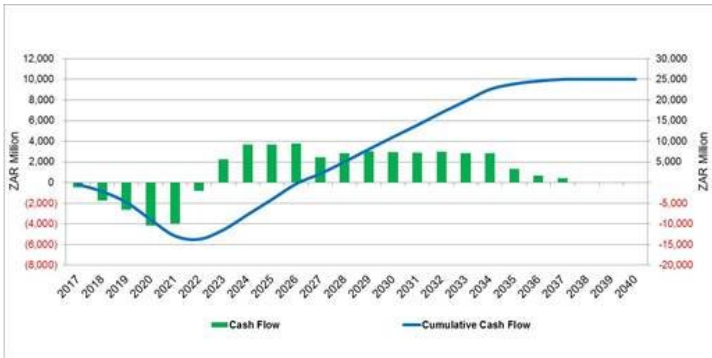


Figure 1-19: Annual Cashflow after Tax

Table 1-13: Investment Bank Consensus Price

Item	Discount Rate	Before Taxation (ZAR)	After Taxation (ZAR)	Before Taxation (USD)	After Taxation (USD)
	Undiscounted	45,781	31,946	3,052	2,130
	4.0%	24,180	16,184	1,612	1,079
	6.0%	17,426	11,263	1,162	750
Net Present Value	8.0%	12,402	7,610	827	507
	10.0%	8,641	4,884	576	325
	12.0%	5,812	2,842	387	189
	14.0%	3,676	1,311	245	87
Internal Rate of Return		19,8%	16.3%	19,8%	16.3%
Project Payback Period (Years)		9	9	9	9

1.23 Mineral Tenure, Surface Rights and Royalties

Currently there are no royalties, back-in rights, payments or other encumbrances that could prevent PTM from carrying out its plans or the trading of its rights to its license holdings at the Waterberg Project. JOGMEC or its nominee has the exclusive right to direct the marketing of the mineral products of the other participants for a 10-year period from first commercial production on an equivalent to commercially competitive arm's length basis and has the first right of refusal to purchase at prevailing market prices any mineral products taken by another participant as its share of joint venture output.

A summary of the mineral exploration and mining rights regime for South Africa is provided in Table 1-12. It should be noted that PTM have a Prospecting Right which allows them should they meet the requirements in the required time, to have the sole mandate to file an application for the conversion of the registered Prospecting Right to a Mining Right.

1.24 Conclusions

Results of this PFS demonstrate that the Waterberg Project warrants development due to its positive, robust economics, large production volume and opportunity relative to the PGM price deck.
.

It is the conclusion of the QPs that the PFS summarized in this technical report contains adequate detail and information to support a Pre-Feasibility level analysis.

Infill drilling over portions of the project area and new estimation methodology has made it possible to estimate a new mineral resource estimate and upgrade portions of the mineral resource to the Indicated category.

A Mineral Resource and Reserves may be declared for the PTM Waterberg project and reported in the tables below:

Table 1-14: T Zone Mineral Resource at 2.5 g/t 4E Cut-off

T-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E	Tonnage	Grade Pt	Grade Pd	Grade Au	Grade Rh	Grade 4E	Grade Cu	Grade Ni	Metal 4E	Metal 4E
	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
Indicated	2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122 375	3.934
Inferred	2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75 485	2.427

Table 1-15: F Zone Mineral Resource at 2.5 g/t 4E Cut-off

F-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E	Tonnage	Grade Pt	Grade Pd	Grade Au	Grade Rh	Grade 4E	Grade Cu	Grade Ni	Metal 4E	Metal 4E
	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
Indicated	2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651 670	20.952
Inferred	2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260 484	8.375

Table 1-16: Probable Reserve at 2.5 g/t 4E Cut-off

Zone	Mt	Moz	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)
T Zone	16.50	2.09	1.14	1.93	0.83	0.04	3.94
F South	10.32	1.26	1.14	2.42	0.19	0.04	3.78
F Central	36.75	4.24	1.08	2.30	0.18	0.04	3.59
F Boundary	16.08	1.94	1.12	2.40	0.19	0.04	3.75
F North	23.02	2.79	1.13	2.42	0.19	0.04	3.78
Total	**102.67**	**12.32**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**

The following prices, based on a 3-year trailing average in accordance with U.S. Securities and Exchange Commission ("SEC") guidance, was used for the assessment of Resources and Reserves.

The Investment Bank Consensus price and spot price were also used for the Sensitivity analysis.

Table 1-17: Key Economic assumptions

Parameter	Unit	3 Yr Trailing Average 31 Jul 2016	Spot Price 6 Oct 2016	Investment Bank Consensus Price Deck 16 Sep 2016
Platinum	US$/oz.	1.212	964	1.213
Palladium	US$/oz.	710	668	800
Gold	US$/oz.	1.229	1.255	1.300
Rhodium	US$/oz.	984	675	1.000
Basket (4E)	US$/oz.	899	798	960
Nickel	US$/lb	6.10	4.52	7.50
Copper	US$/lb	2.56	2.17	2.90
Base Metals Refining Charge	% Gross Sales	85%		
Copper Refining Charge	% Gross Sales	73%		
Nickel Refinery Charge	% Gross Sales	68%		

The key features of the Waterberg 2016 PFS include:

- Planned steady state total and annual production and recoveries for the Mining zones are depicted in the table below.

Table 1-18: Waterberg 2016 PFS Production results.

Item	Unit	Total LOM	LOM Annual Avg
Ore Production			
Mineral Reserve	Mt	103	7.2
Ore Milled	Mt	103	7.2
T-Zone	g/t	3.94	3.94
F South	g/t	3.78	3.78
F Central	g/t	3.59	3.59
F Boundary	g/t	3.75	3.75
F North	g/t	3.78	3.78
4E	g/t	3.73	3.73
Copper	%	0.08	0.08
Nickel	%	0.15	0.15
Recoveries			
Platinum	%	82.5	82.5
Palladium	%	83.2	83.2
Gold	%	75.3	75.3
Rhodium	%	59.4	59.4
4E	%	82.1	82.1
Copper	%	87.9	87.9
Nickel	%	48.8	48.8
Recovered Metal			
Platinum	koz	3,029	222

Item	Unit	Total LOM	LOM Annual Avg
Palladium	koz	6,297	482
Gold	koz	715	45
Rhodium	koz	73	6
4E	koz	10,114	744
Copper	Mlb	168	11
Nickel	Mlb	163	12

Waterberg Key financial metrics are depicted in the table below:

Table 1-19: Waterberg 2016 PFS Results

Item	Units	Total
Key Financial Results (3 Year Trailing Price Deck — US$/ZAR 15) - 31 July 2016		
Life of Mine	years	19
Capital to Full Production	US$M	1060
Mine Site Cash Cost	US$/oz 4E	389
Total Mine Cash Costs After Credits	US$/oz 4E	248
Total Cash Costs After Credits	US$/oz 4E	481
All in Costs After Credits	US$/oz 4E	661
Site Operating Costs	US$/t Milled	38
After Tax NPV @ 8%	US$M	320
After Tax IRR	%	13.5
Project Payback Period (Start First Capital)	years	10
Investment Bank Consensus Price Deck- 16 September 2016		
After Tax NPV8	US$M	507
After Tax IRR	%	16.3

Standard industry practices, equipment and design methods were used in this PFS Study. The report authors are unaware of any unusual or significant risks, or uncertainties that would affect project reliability or confidence based on the data and information made available. For these reasons, the path going forward must continue to focus on drilling activities and obtaining the necessary permitting approval, while concurrently advancing key activities in the FS that will reduce project execution time.

Risk is present in any mineral development project. Feasibility engineering formulates design and engineering solutions to reduce that risk common to every project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, political risks, schedule and cost overruns, and labor sourcing. Opportunities include further optimization of the mine plan and potential reduction of development sustaining capital. The company indicates they will be focused on these aspects in the DFS phase.

The project provides attractive returns when compared to competitive projects in the Bushveld Complex in the Western or Northern Limb. Based on the competitive returns the project is recommended to proceed to the Definitive Feasibility Stage, ("DFS"). Drilling for measured resources should continue and be designed and budgeted along with the scoping process for the DFS.

1.24.1 Geology and Mineral Estimates

A Mineral Resource may be declared for the PTM Waterberg project. This Resource comprises an Indicated Resource of 31 Million tonnes at 3.88g/t 4E for the T-zone; and 186 Million tonnes at 3.49 g/t 4E for the F-zone. Additional Inferred Resources of 19 Million tonnes at 3.79g/t 4E for the T-zone and 77 Million tonnes at 3.37g/t 4E for the F-zone. These Resources are reported at a 4E grade cut-off of 2.5 g/t. Only Indicated resources are included in the mine plan and financial analysis.

1.24.2 Geotechnical and Rock Engineering

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

- The general geotechnical conditions are suitable for the planned infrastructure and the soil and rock is capable of supporting the planned structures.
- The geotechnical database was adequate for this level of study.
- The mining methods that have been identified as most suited are Blind Longitudinal Retreat (BLR) and Sub-Level Open Stoping (SLOS). These mining methods offer flexibility and with proper sequencing of mining cuts and support strategies, regional stability can be improved.

1.24.3 Mining

The mine design and production schedules presented are deemed as reasonable for a PFS level of confidence. Although, the BLR mining method is not widely utilized, it is the view of the project study team that the layouts and schedule rates are not overly aggressive.

A number of potential optimization opportunities have been identified and can be further quantified and expanded in the DFS.

1.24.4 Metallurgy

Sufficient test work to support the Waterberg Platinum pre-feasibility study has been undertaken.

Extensive metallurgical test work has been conducted on two different flowsheets, namely the MF1 and MF2 flowsheets, with encouraging results obtained from both. Test results have demonstrated that some of the ore types respond better to a particular configuration.

Bench scale test work conducted, on the Waterberg ores types and blends, has demonstrated that a saleable final concentrate containing at least 80 g/t 4E can be produced by applying a MF2 flowsheet and using standard Southern African PGM reagents. No deleterious elements are expected in the final concentrate, whilst 4E recoveries in excess of 80% are expected for the selected process design.

1.24.5 Infrastructure

For the purposes of this PFS, a range of options were considered for the on site and regional infrastructure.

The main infrastructure requirements for the Waterberg Project are access roads, residue disposal, water management, power supply and process plant to service and treat the targeted mine production.

The Waterberg Project is situated in a remote area and will require approximately 32km of existing unpaved roads to be surfaced.

A combination of sewage effluent together with groundwater is considered the most viable solution to meet the bulk water requirements of the proposed mining schedule. Wellfields with poor water quality will be targeted so as not to compete with domestic water uses in the area.

The bulk electricity supply for the project is being planned to cater for mining and plant production rates of up to 600ktpm, which correspond to an electrical load of up to 160MVA. A temporary electrical supply is being planned for the construction stage. Eskom has been engaged in the design process.

The availability of skilled labor resources, for both construction and operational phases, is limited and the training and skills development program will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Waterberg Project. The company plans to use its accredited training center.

1.24.5.1 Residue Storage Facility

The following conclusions were drawn from the study:

- A pre-feasibility design of the Residue Disposal Facility (RDF) for the Waterberg Project has been undertaken, in which:
  - A suitable site for the RDF has been identified;
  - conventional/thickened tailings is the safest option and well understood in the platinum industry and has been regarded as the preferred option for Waterberg;
  - a conventional/thickened RDF has been shown to be the most cost effective option for Waterberg in terms of water costs; and
  - The total LoM cost associated with the Waterberg RDF over the duration of the project life (Feasibility Study to Post Closure) is estimated at R1,057 million.

1.24.5.2 Bulk Water Supply

Of all the options considered, a combination of sewage effluent together with groundwater is considered the most viable solution to meet the proposed mining schedule.

Consider the bulk water source options as described is Section 19.3. The option of wellfields in combination with an effluent water pipeline from Bochum (Senwabarwama Ponds) is the most favorable with the least risk and is considered the base case. This infrastructure would allow the collection of water from various sources along the way, thereby ensuring a more sustainable bulk water supply to the Waterberg site.

The wellfields in combination with Waste Water Treatment Works (WWTW) pipeline from Bochum also creates the following opportunities:

- Access to groundwater from various wellfield areas along the route to supplement supply. This water is considered unsuitable for human consumption and would therefore have little impact on community water requirements;
- collection of water from smaller WWTW at Mogwadi;
- possible future expansion of the pipeline to collect effluent from Makhado WWTW

1.24.5.3 Bulk Power Supply

The updated electricity supply plan compiled by Eskom provides for the establishment of new 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation,

The development of the abovementioned infrastructure will be done in conjunction with Eskom on a Self-Build basis and this work is already in an advanced stage.

1.24.6 Market Studies and Contracts

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal contact by the Company is reported to indicate capacity and interest by two smelters. This will need to be confirmed in the DFS stage. Based on a comparison with the Merensky style of concentrate the Waterberg concentrate is considered attractive.

Based upon industry data, it is expected that the payability for the concentrate sold to a local smelter operator will be up to 85% for the PGE's, 73% for contained copper and 68% for contained nickel. It is expected that the payment terms will be full payment after 16 weeks for all metals, but with financing arrangements, these terms can be improved, but with significant interest charges for the up-front payment.

1.24.7 Environmental Impact Assessment Studies

The environmental permit, not yet approved, is of paramount importance, and delays from the company plan will increase project execution time. Without the permit advancement to a mining right with approval, the Project cannot proceed and failure to secure the necessary permits could stop or delay the Project. The project design considers the environment and local communities.

1.24.8 Community Social Impact Assessment Studies

The Community Social Impact Assessment Study is underway. It is focusing on all the three farms affected by the mining operations. This study is important because it assists in the compilations of the Social and Labor Plan (SLP). The SLP forms part of the Mining Right application process. Detailed consultation has been on going and is well documented.

The process for completing a Mining Right Application is underway. Discussions have been positive and business like. Both the community and the company have arranged experienced mining lawyers to facilitate the negotiations. The small community of approximately 100 homes will have to be relocated to the farm next to Ketting, which is also owned by the same community. This will require relocations costs. The MRPDA provides for a right of access and fair compensation will be required.

Allowance for land purchase and relocation costs was provided for the SLP in the Financial Model.

1.25 Recommendations

The QPs recommend that the Waterberg project advance to the DFS stage. The project financial model, including low capital cost per annual ounce of production and low operating costs provides the basis for further investment and refinement of the project design. The QPs recommend that based on the large scale PGM production profile of the project at 744,000 4E ounces per year that the project owners initiate discussions with smelters an investigate a standalone smelting option. The QPs also recommend that the owners initiate work towards an application for a Mining Right including the development of a Social and Labor Plan and environmental permits.

1.25.1 Geology and Mineral Estimates

It is recommended that exploration drilling continue in order to advance the geological confidence in the deposit through infill drilling. This will provide more data for detailed logging and refined modelling. This is expected to confirm the geological continuity and allow the declaration of further Indicated Mineral Resources.

Given the results of the diamond drilling on the northern area and the extent of target areas generated by geophysical surveys, the completion of the planned exploration drilling is recommended north of the location of the current exploration programme. The objective of the exploration drilling would be to find the limit of the current deposit, confirm the understanding of the F Zone and improve the confidence for a selected part of the deposit to the measured category for the DFS. Geotechnical and Rock Engineering

1.25.2 Geotechnical and Rock Engineering

The following is a list of work that will be required for a feasibility level of study. Although the list is comprehensive is by no means exhaustive.

- Additional trial pits should be excavated at the exact positions of the proposed structures during the Definitive Feasibility Study at the next stage. A diamond drilled triple tubes borehole should be undertaken at each surface crusher up to a depth of 45m or 10m into medium hard rock sandstone or stronger (>25MPa). Appropriate soil and rock laboratory testing should be part of the geotechnical investigation at this stage, including falling head permeability test of the in situ material for the clay/geosynthetic liner of the tailing dam.

- The T-Reef should be explored geotechnically in more detail.

- Sufficient data should be collected to allow for rigorous analyses of joints. This will include oriented core.

- A representative number of boreholes should be logged at selected locations to derive a more complete rock mass model that will inform designs of excavations away from the orebody as well as the main on-reef declines.

- With improved understanding of the model input parameters and the mining configuration, the assessment of the stability of the BLR designs, SLOS stopes and SLOS pillars can be conducted with greater confidence.

1.25.3 Mining

It is recommended that the opportunities mentioned in Section 16.12.2 be investigated further. This could be done prior to the next phase of the study or at least during the next DFS study phase.

- The mine design of underground access infrastructure, other underground excavations and production areas should be prepared to higher level of confidence for the DFS.
- Scheduling rates for development and production should be revisited to ensure that the rates planned remain realistic and achievable.
- Compile a detailed Bill of Quantities of the mine design and involve relevant mining contracting companies so that accurate cost estimates can be prepared.
- Conduct a simulation exercise that considers all underground logistics. It is recommended that this be done using an appropriate software package.
- Review the risks mentioned so that where possible adequate mitigating factors can be incorporated into the mine design and schedule.
- Complete a value engineering exercise on development and mining designs to reduce dilution and increase head grades.
- Waste development in sustaining capital should be studied for reduction with investigation and further detailing of the ventilation plan

1.25.4 Metallurgy

It is recommended that the opportunities mentioned in Section 17 be investigated further. This could be done prior to the next phase of the study or at least during the next study phase.

The following is also recommended for the next study phase:

- Flotation test work using water from the envisaged raw water sources to ensure the flotation performance is not negatively affected.
- Testing of the MF2 circuit using an Oxalic acid and Thiourea reagent scheme
- Comminution variability test work on the individual ore types
- Comminution variability test work on various possible mine blends
- Flotation open circuit batch variability test work on the individual ore types
- Flotation open circuit batch variability test work on various possible mine blends
- Concentrate thickening and filtration test work

Geotechnical investigation of the plant site to accurately determine founding conditions in the plant area and inform the design of the civil engineering works is also recommended.

The Definitive Feasibility Study would be completed using the test work results to optimize the process and infrastructure design and allow a more accurate assessment of the capital cost, operating cost and risks.

1.25.5 Infrastructure

Progress in-depth further infrastructure studies associated with access roads, supply and logistics, RDF design methodologies and any other areas of the Project where studies and confidence levels are lacking and for which information is required to support permitting and feasibility studies.

The Infrastructure component outlines a series of recommendations for the Project including progression to the Feasibility Study phase in order to assess the Waterberg development further including:

1.25.5.1 Residue Storage facility

For the Residue Disposal Facility Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- A geotechnical investigation of the RDF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials.
- A seepage analysis and slope stability study be undertaken to confirm the seepage regimes through the RDF as well as to confirm the RDF stability. The results of these analyses could affect greatly on the geometry of the RDF walls and ultimate height of the facility.
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample.
- Possible further optimization of the RDF preparatory works in terms of layout, footprint extent, etc. including any changes to the mine plan.
- Review the construction rates with a contractor to price the facility with representative rates.
- Compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy; and a hydrological study of potential flood lines near the RDF.

1.25.5.2 Bulk Water Supply

Due to the scarcity of water in the area, it will be critical to conduct more detailed hydrogeological investigations in order to identify in detail the potential groundwater resources that can be developed for mine supply and to predict the mine inflows and impact zone accurately. This will also be important to determine external bulk water requirements and the timing thereof. These hydrogeological investigations should include a numerical model, which will also assist the mine with monitoring and water management during the life of mine.

1.25.5.3 Bulk Power Supply

The electrical supply for the construction phase will involve the strengthening of an existing 22kV rural overhead line until the permanent supply infrastructure is in place. The 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation and the associated infrastructure would form part of the permanent supply infrastructure

1.25.6 Market Studies and Contracts

It is recommended that the local smelter operators be formally approached to better understand the appetite to consume the significant concentrate production once the mine is at steady state. A competitive process could be developed with the Japanese partner JOGMEC.

In addition, during the Definitive Feasibility Study, it is recommended that a Scoping Study be completed into the potential for the inclusion of a Waterberg Project Smelter on site. The product from this smelter could be a furnace matte or a convertor matte, which could be treated locally or exported for refining.

1.25.7 Environmental Impact Assessment Studies

The future development and delivery of the Waterberg Project will be underpinned by a programme of work for the mitigation of social and environmental impacts; creating value through good governance practices.

PTM has a programme of work in place to comply with the necessary environmental, social and community requirements, which include:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA;
- Public Participation Process (PPP) in accordance with the NEMA Guidelines;
- Specialist investigations in support of the ESIA;
- Integrated Water Use License Application (IWULA) in compliance with the National Water Act (NWA); and
- Integrated Waste Management License in compliance with the National Environmental Management Waste Act (NEMWA).

1.25.8 Community Social Impact Assessment Studies

The community impact assessment studies are being conducted and Platinum Group Metals and detailed documentation of the process is recommended to continue with appropriate specialists and counsel.

QUALIFIED PERSONS

The following Qualified Persons have completed work in preparation of the PFS and are responsible for the contents:

- **Independent Engineering Qualified Person:**
  Mr. Robert L Goosen
  (B.Eng. (Mining Engineering)) Pr. Eng. (ECSA)
  Advisian/WorleyParsons Group

- **Independent Geological Qualified Person:**
  Mr. Charles J Muller
  (B.Sc. (Hons) Geology) Pr. Sci. Nat.
  CJM Consulting (Pty) Ltd

- **Independent Engineering Qualified Person:**
  Mr. Gordon I. Cunningham
  B. Eng. (Chemical), Pr. Eng. (ECSA), Professional association to FSAIMM
  Turnberry Projects (Pty) Ltd.

This material change report has been reviewed and approved by R. Michael Jones, P.Eng., a non-independent Qualified Person and the CEO of the Company.

Forward-looking Statements

This material change report contains forward-looking information within the meaning of Canadian securities laws and forward-looking statements within the meaning of U.S. securities laws (collectively "forward-looking statements"). Forward-looking statements are typically identified by words such as: believe, expect, anticipate, intend, estimate, plans, postulate and similar expressions, or are those, which, by their nature, refer to future events. All statements that are not statements of historical fact are forward-looking statements. Forward-looking statements in this material change report include, without limitation, the projections and assumptions relating to future events that are contained in the PFS, including, without limitation NPV, IRR, costs, potential production of the

Waterberg Project and other operational and economic projections with respect to the Waterberg Project; future activities at Waterberg and the funding of such activities; trends in metal prices; potential future market conditions; the Company's overall capital requirements and future capital raising activities, plans and estimates regarding exploration, studies, development, construction and production on the Company's properties, other economic projections and the Company's outlook. Statements of mineral resources and mineral reserves also constitute forward-looking statements to the extent they represent estimates of mineralization that will be encountered on a property and/or estimates regarding future costs, revenues and other matters. Although the Company believes the forward-looking statements in this material change report are reasonable, it can give no assurance that the expectations and assumptions in such statements will prove to be correct. The Company cautions investors that any forward-looking statements by the Company are not guarantees of future results or performance, and that actual results may differ materially from those in forward-looking statements as a result of various factors, including; the Company's capital requirements may exceed its current expectations; the uncertainty of cost, operational and economic projections; the ability of the Company to negotiate and complete future funding transactions; variations in market conditions; the nature, quality and quantity of any mineral deposits that may be located; metal prices; other prices and costs; currency exchange rates; the Company's ability to obtain any necessary permits, consents or authorizations required for its activities; the Company's ability to produce minerals from its properties successfully or profitably, to continue its projected growth, or to be fully able to implement its business strategies; and other risk factors described in the Company's shelf prospectus and registration statement, Form 40-F annual report, annual information form and other filings with the Securities and Exchange Commission and Canadian securities regulators, which may be viewed at www.sec.gov and www.sedar.com, respectively. Except as required under applicable securities legislation, the Company undertakes no obligation to publicly update or revise forward-looking statements, whether as a result of new information, future events or otherwise.

This material change report also includes a reference to mineral resources and mineral reserves. The estimation of resources and reserves is inherently uncertain and involves judgement. Mineral resources that are not reserves do not have demonstrated economic viability. Judgements associated with geology, tonnage grades in place and that can be mined may prove to be unreliable and inaccurate. Fluctuations in metals prices, exchange rates, labour costs and government regulations among other things may materially affect resources and reserves. The Company does not yet have a right to mine the reported resources and reserves and there can be no assurance that the Company will convert its prospecting permits to a mining right.

5.2 Disclosure for Restructuring Transactions

N/A

ITEM 6. RELIANCE ON SUBSECTION 7.1(2) OF NATIONAL INSTRUMENT 51-102

N/A

ITEM 7. OMITTED INFORMATION

N/A

ITEM 8. EXECUTIVE OFFICER

The following senior officer of the Issuer is knowledgeable about the material change and may be contacted by the Commission at the following telephone number:

R. Michael Jones, President & CEO Phone: (604) 899-5450

ITEM 9. DATE OF REPORT

October 20, 2016

INDEPENDENT TECHNICAL REPORT ON THE WATERBERG PROJECT INCLUDING MINERAL RESOURCE UPDATE AND PRE-FEASIBILITY STUDY

Project Areas located on the Northern Limb of the Bushveld Igneous Complex, South Africa
(S23°22'01"; E28°49'42")

For

PLATINUM GROUP METALS (RSA) (PTY) LTD
REPUBLIC OF SOUTH AFRICA REGISTERED COMPANY
REGISTRATION NUMBER: 2000/025984/07
A WHOLLY OWNED SUBSIDIARY OF PLATINUM GROUP METALS LTD.
TSX — PTM; NYSE; MKT — PLG
Report Date: October 19, 2016

Effective Date of Mineral Resources and Reserves of this Report is October 17, 2016

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Independent Engineering Qualified Person:

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IMPORTANT NOTICE

The Waterberg Project Technical Report has been evaluated and prepared in accordance with NI 43-101 to comply with the requirements for a Pre-Feasibility Study (PFS). This Technical Report includes updated Inferred and Indicated Resources. Only Indicated resources have been incorporated into the mine plan and financial model. The mineable Reserve represents the portion of the Indicated resource that can be economically mined as delivered to the mill, and as demonstrated in this PFS.

The reader is cautioned to note that the mineral Reserves are included within the Indicated Mineral Resources, and are not in addition to them.

Cautionary Note to U.S. Investors:

The definitions of proven and probable Reserves used in NI 43-101 differ from the definitions in Securities and Exchange Commission (SEC) Industry Guide 7. Under SEC Industry Guide 7 standards, a “final” or “bankable” feasibility study is required to report Reserves. The three-year historical average price is used in any Reserve or cash flow analysis to designate Reserves and the primary environmental analysis or report must be filed with the appropriate governmental authority. As a result, the Reserves reported by the Company in accordance with NI 43-101 may not qualify as “Reserves” under SEC standards. In addition, the terms “mineral resource”, “measured mineral resource”, “indicated mineral resource” and “inferred mineral resource” are defined in and required to be disclosed by NI 43-101; however, these terms are not defined terms under SEC Industry Guide 7 and normally are not permitted to be used in reports and registration statements filed with the SEC. Mineral Resources that are not mineral Reserves do not have demonstrated economic viability. Readers are cautioned not to assume that any part of, or all of the mineral deposits in these categories will ever be converted into Reserves. “Inferred Mineral Resources” have an amount of uncertainty as to their existence, and uncertainty as to their economic and legal feasibility. It cannot be assumed that any part of an inferred mineral resource will ever be upgraded to a higher category. Under Canadian securities laws, estimates of inferred Mineral Resources may not form the basis of feasibility or pre-feasibility studies, except in rare cases.

Additionally, disclosure of “contained ounces” in a resource is permitted disclosure under Canadian Securities laws; however, the SEC normally only permits issuers to report mineralization that does not constitute “Reserves” by SEC standards as in-place tonnage and grade without reference to unit measurements. Accordingly, information contained or referenced in this technical report containing descriptions of the Company’s mineral deposits may not be comparable to similar information made public by U.S. companies subject to the reporting and disclosure requirements of United States federal securities laws and the rules and regulations thereunder.

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Figure List

Maps and Figures are dated as of the date of this report.

1. Summary

The following items are the main components forming the purpose of the Pre-Feasibility Study:

- To update the Mineral Resource estimate to October 2016 and to publish the results of the PFS;
- To determine the optimal techno-economic solution that considers all opportunities and risks, that exceeds the investment criteria hurdle and is aligned with the Company's strategy;
- To justify the expenditure for a Feasibility Study of one selected project option;
- To compile a work programme, budget and schedule baseline for the development of the scope and deliverables of the Feasibility Study;
- To provide a framework of project options as a converging view, to demonstrate that all the discarded project options have been studied to the degree that they are clearly identified as inferior and will not re-emerge as potential options;
- To optimize the project size, scope, technical and production parameters by evaluating all the alternative technology and implementation options, as well as the project costs and benefits
- To determine targets for further value enhancement and risk reduction.
- To provide the basis for a Mining Rights Application.

1.1 Introduction

This report was prepared in compliance with National Instrument 43—101, Standards of Disclosure for Mineral Projects (NI 43—101), and documents the results of ongoing exploration and project work.

The project is the development of large greenfield platinum mine and concentrator plant north of the town of Mokopane in the Province of Limpopo.

A Preliminary Economic Assessment (PEA) on the original Waterberg JV was completed and announced in February 2014.

The resource estimate includes the T Zone, F South, F Central, F Boundary and F North with the shallowest edge of the known deposit on the T-Zone at approximately 140m below surface. The resource estimate has been cut off at an arbitrary depth of 1,250m vertical. Drill intercepts well below 1,250m vertical indicate the deposit continues and is open down dip from this depth. The deposit is 13km long and remains open along strike to the north.

The key features of the Waterberg 2016 PFS include:

- Development of a large, mechanized, underground mine that is planned at a 7.2Mtpa throughput scenario;
- Planned steady state annual production rate of 744 koz of platinum, palladium, rhodium and gold (4E) in concentrate;
- Estimated Capital to full production requirement of approximately ZAR15,906 billion (US$1,060 million), including ZAR999 million (US$67 million) in contingencies;
- Peak funding ZAR13,694 million (US$914 million);

- After-tax Net Present Value (NPV) of ZAR4,805 million (US$320 million), at an 8% discount rate (three year trailing average price desk 31 July 2016 US$1,212/oz Pt, US$710/oz Pd, US$984/oz Rh, US$1,229/oz Au, US$/ZAR 15);

- After-tax Net Present Value (NPV) of ZAR7,610 million (US$507 million), at an 8% discount rate (Investment Bank Consensus Price) US$1,213/ozPt, US$800/oz Pd, US$1,000 Rh, US$1,300/oz Au, US$/ZAR 15

- After-tax Internal Rate of Return (IRR) of 13.5% (three year trailing average price deck); and

- Internal Rate of Return (IRR) of 16.3% After-tax (Investment Bank Consensus Price).

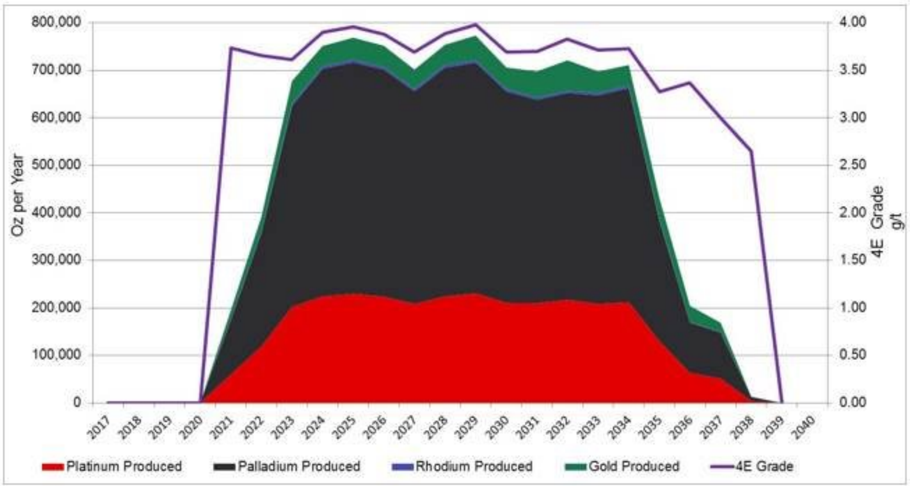


Figure 1-1: Total Ounces Produced

Mine production is shown in Figure 1-2 and the after tax cash flow is shown in Figure 1-3.

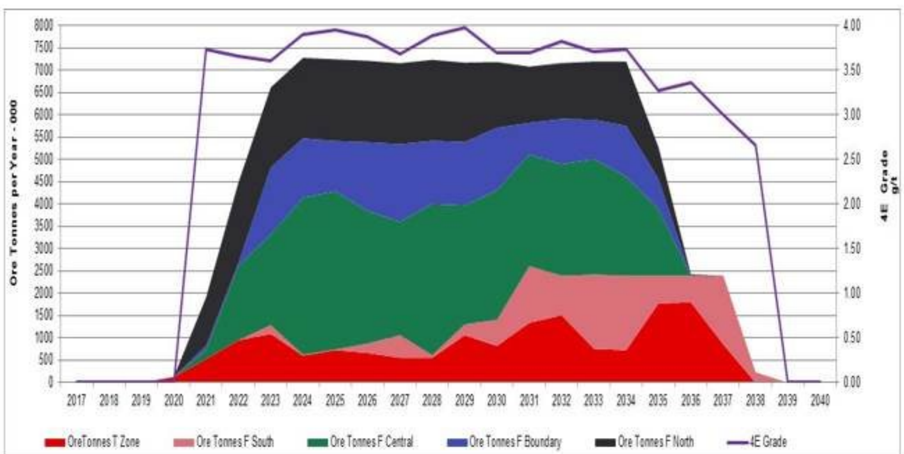


Figure 1-2: Total Mine Production

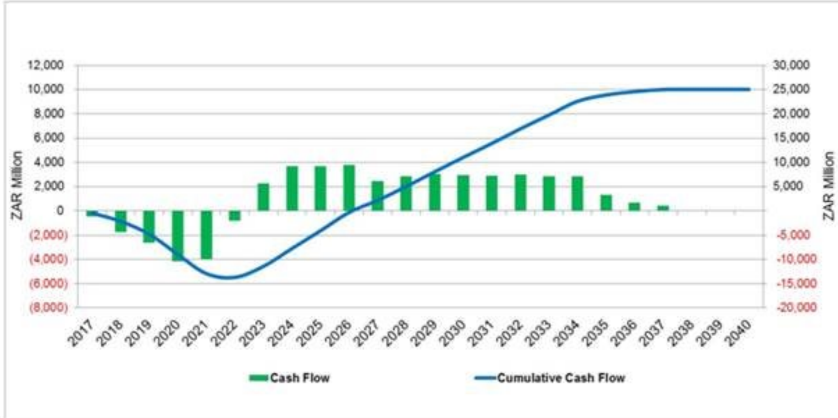


Figure 1-3: Annual Cashflow after Tax

1.2 Ownership

The ownership structure consists of:

- Platinum Group Metals (RSA) (Pty) Ltd, abbreviated to PTM (45.65% directly)
- JOGMEC (28.35%)
- BEE partner Mnombo Wethu Consultants (26%).

Because of PTM's 49.90% ownership in Mnombo, the Company has a direct and indirect 58.62% overall interest in the project. Platinum Group Metals is the operator.

The size and scale of the Waterberg Project represents a significant alternative to narrow width, conventional, Merensky and UG2 mining on the Western and Eastern Limbs of the Bushveld Complex.

The government of South Africa holds the mineral rights to the project properties under the Mineral and Petroleum Resources Development Act (Act, 28 of 2002). The mineral rights are held through a mining right under the Mineral and Petroleum Resources Developm

1.3 Location and Access

The Waterberg Mineral Project is located approximately 85km north of the town of Mokopane in the Province of Limpopo, South Africa as shown in Figure 1-4.

Platinum Group Metals has been granted prospecting rights covering the Waterberg and Waterberg Extension Project of 111,882 ha. The prospecting rights are approximately 40km north south and 40 km east west centered at 23°22'01" south latitude and 28°49'42" east longitude. The project is accessible by paved and dirt roads by vehicle.

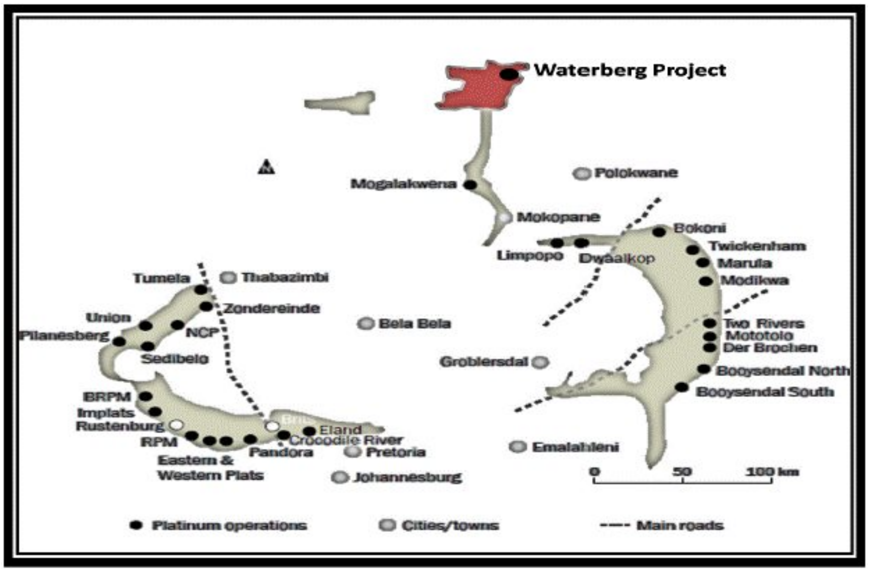


Figure 1-4: Location of Waterberg Project within the Bushveld Complex in the Republic of South Africa

1.4 Geological Setting, Deposit Type and Mineralisation

The Bushveld and Molopo Complexes in the Kaapvaal Craton are two of the most well known mafic/ultramafic layered intrusions in the world. The Bushveld complex was intruded about 2,060 million years ago into rocks of the Transvaal Supergroup, largely along an unconformity between the Magaliesberg quartzite of the Pretoria Group and the overlying Rooiberg felsites. It is estimated to exceed 66,000km ^{2} in extent, of which about 55% is covered by younger formations. The Bushveld Complex hosts several layers rich in Platinum Group Metals (PGM), chromium and vanadium, and constitutes the world's largest known resource of these metals.

The Waterberg Project is situated off the northern end of the previously known Northern Limb, where the mafic rocks have a different sequence to those of the Eastern and Western Limbs.

PGM mineralization within the Bushveld package underlying the Waterberg Project is hosted in two main layers: the T-Zone and the F-Zone.

The T-Zone occurs within the Main Zone just beneath the contact of the overlaying Upper Zone. Although the T-Zone consists of numerous mineralized layers, three potential economical layers were identified, T1, T2HW and T2 layers. They are composed mainly of anorthosite, pegmatoidal gabbros, pyroxenite, troctolite, harzburgite, gabbronorite and norite.

The F-Zone is hosted in a cyclic unit of olivine rich lithologies towards the base of the Main Zone towards the bottom of the Bushveld Complex. This zone consists of alternating units of harzburgite, troctolite and pyroxenites.

The F-Zone was divided into the FH and FP layers. The FH layer has significantly higher volumes of olivine in contrast with the lower lying FP layer, which is predominately pyroxenite. The FH layer is further subdivided into six cyclic units chemically identified by their geochemical signature, especially chrome. The base of these units can also be lithologically identified by a pyroxenite layer.

1.5 Geology

The Waterberg Project is located along the strike extension of the Northern Limb of the Bushveld Complex. The geology consists predominantly of the Bushveld Main Zone gabbros, gabbronorites, norites, pyroxenites and anorthositic rock types with more mafic rock material such as harzburgite and troctolites that partially grade into dunites towards the base of the package. In the southern part of the project area, Bushveld Upper Zone lithologies such as magnetite gabbros and gabbronorites do occur as intersected in drill hole WB001 and WB002. The Lower Magnetite Layer of the Upper Zone was intersected on the south of the project property (Disseldorp) where drill hole WB001 was drilled and intersected a 2.5m thick magnetite band.

On the property, the Bushveld package strikes south-west to northeast with a general dip of34º-38º towards the west is observed from drill hole core for the layered units intersected on Waterberg property within the Bushveld Package (Figure 1-5). However, some structural blocks may be tilted at different angles depending on structural and /or tectonic controls.

The Bushveld Upper Zone is overlain by a 120m to 760m thick Waterberg Group, which is a sedimentary package predominantly, made up of sandstones, and within the project area that sedimentary formations known as the Setlaole and Makgabeng Formations constitute the Waterberg Group. The Waterberg package is flat lying with dip angles ranging from to 2º to 5º. Figure 1-5 gives an overview of interpreted geology for the Waterberg Project.

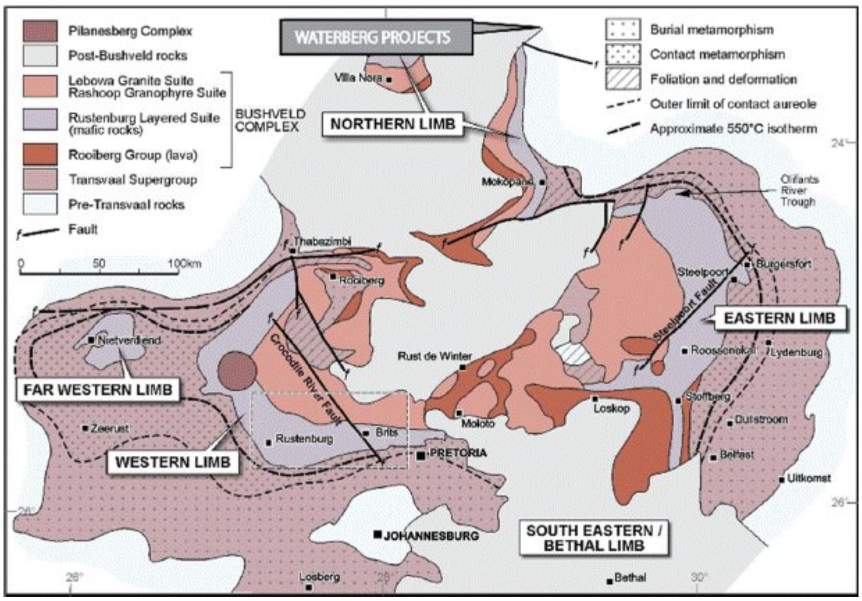


Figure 1-5: Regional Geology

1.6 Exploration Status

The Waterberg Project is at an advanced project that has undergone preliminary economic evaluations, which have warranted further work. Drilling to date has given the confidence to classify Mineral Resources as Inferred and Indicated.

1.7 Sample Preparation

The sampling methodology concurs with PTM protocol based on industry best practice. The quality of the sampling is monitored and supervised by a qualified geologist. The sampling is done in a manner that includes the entire potentially economic unit, with sufficient shoulder sampling to ensure the entire economic zones are assayed.

1.8 Analysis

For the present database, field samples have been analyzed by three different laboratories. The primary laboratory is currently Set Point laboratories (South Africa). Genalysis (Australia) is used for referee test work to confirm the accuracy of the primary laboratory. Analysis was also completed at Bureau Vertitas in Rustenberg.

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

Samples are analyzed for Pt (ppm), Pd (ppm) Rh (ppm) and Au (ppm) by standard 25g lead fire-assay using a silver collector. Rh (ppm) is assayed using the same method but with a palladium collector and only for selected samples. After pre-concentration by fire assay, the resulting solutions are analyzed using ICP-OES (Inductively Coupled Plasma—Optical Emission Spectrometry).

The base metals (copper, nickel, cobalt and chromium) are analyzed using ICP-OES (Inductively Coupled Plasma — Optical Emission Spectrometry) after a multi-acid digestion.

This technique results in “almost” total digestion. The drilling, sampling and analytical aspects of the project are considered to have been undertaken to industry standards. The data is considered reliable and suitable for mineral resource estimation.

The company completes a Quality Control and Assurance review on all of the laboratory samples including a review of the lab quality control samples and the company inserted standards. Issues that are detected beyond acceptable levels are requested for re-analysis.

1.9 Drilling

The data from which the structure of the mineralized horizons was modelled and grade

Values estimated were derived from 298 538m of diamond drilling. This report updates the mineral resource estimate using this dataset. The initial database for this mineral resource estimate was received on July 7, 2016. The raw database consists of 303 drill holes with 483 deflections totaling 300,875 m.

The management of the drilling programmes, logging and sampling has been undertaken from two facilities: one at the town of Marken in Limpopo Province, South Africa and the other on the farm Goedetrouw 366LR within the prospecting right area.

Drilled core is cleaned, de-greased and packed into metal core boxes by the drilling company. The core is collected from the drilling site on a daily basis by PTM personnel and transported to the core yard. Before the core is taken off the drilling site, core recovery and the depths are checked. Core logging is done by hand on a pro-forma sheet by qualified geologists under supervision of the Project Geologist.

1.10 Quality Control and Quality Assurance

PTM have instituted a complete QA/QC programme including the insertion of blanks and certified reference materials as well as referee analyses. The programme is being followed and is considered to be to industry standard. The data is as a result, considered reliable in the opinion of the Qualified Person.

1.11 Mineral Resource Estimate

This report documents the mineral resource estimate - Effective Date: 17 October 2016. The Mineral Resources are reported in the table below. Infill drilling over portions of the project area and new estimation methodology has made it possible to estimate a new mineral resource estimate and upgrade portions of the mineral resource to the Indicated category. The Mineral Resource Statement is summarized below:

Table 1-1: T-Zone Mineral Resource at 2.5g/t 4E Cut-off

T-Zone 2.5g/t Cut-off

Resource	Cut-off 4E	Tonnage	Grade							Metal 4E	
			Pt	Pd	Au	Rh	4E	Cu	Ni		
Category	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
Indicated	2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122,375	3.934
Inferred	2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75,485	2.427

Table 1-2: F-Zone Mineral Resource at 2.5g/t 4E Cut-off

F-Zone 2.5g/t Cut-off

Resource	Cut-off 4E	Tonnage	Grade							Metal 4E	
			Pt	Pd	Au	Rh	4E	Cu	Ni		
Category	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
Indicated	2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651,670	20.952
Inferred	2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260,484	8.375

4E = platinum Group Elements (Pd+Pt+Rh) and Au The cut-offs for Mineral Resources have been established by a qualified person after a review of potential operating costs and other factors. The Mineral Resources stated above are shown on a 100% basis, that is, for the Waterberg Project as a whole entity. Conversion Factor used — kg to oz = 32.15076. Numbers may not add due to rounding. Resources do not have demonstrated economic viability. A 5% and 7% geological loss have been applied to the indicated and inferred categories respectively. Effective Date Oct 17, 2016. Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance was used for the assessment of Resources; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, Rh, US$984/oz, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15.

The combined Mineral Resource Statement is summarized below:

Table 1-3: Total Mineral Resource at 2.5g/t 4E Cut-off

Waterberg Total 2.5g/t Cut-off

Resource	Cut-off		Grade							Metal	
	4E	Tonnage	Pt	Pd	Au	Rh	4E	Cu	Ni	4E	
Category	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
Indicated	2.5	218.265	1.06	2.18	0.26	0.04	3.55	0.08	0.15	774,045	24.886
Inferred	2.5	97.212	1.03	2.10	0.30	0.03	3.46	0.06	0.11	335,969	10.802

Mineral Resources at Waterberg on a 100% project basis have decreased to an estimated 10.8 million ounces 4E in the Inferred category but increased to 24.9 million ounces 4E in the Indicated category, from 23.9 million ounces 4E Indicated in April 2016:

1. The Mineral Resources are classified in accordance with the SAMREC standards. There are certain differences with the “CIM Standards on Mineral Resources and Reserves”; however, in this case the QP believes the differences are not material and the standards may be considered the same. Mineral resources that are not mineral reserves do not have demonstrated economic viability and Inferred resources have a high degree of uncertainty.

2. The Mineral Resources and are provided on a 100% project basis and Inferred and Indicated categories are separate and the estimates have an effective date of 17 October 2016.

3. A cut-off grade of 2.5g/t 4E for both the T and the F Zones is applied to the selected base case Mineral Resources. Previously a 2g/t 4E cut-off was applied to the resources.

4. Cut off for the T and the F Zones considered costs, smelter discounts, concentrator recoveries from previous engineering work completed on the property by the Company. The Resource model was cut-off at an arbitrary depth of 1250m, although intercepts of the deposit do occur below this depth.

5. Mineral Resources were completed by Charles Muller of CJM Consulting.

6. Mineral Resources were estimated using Kriging methods for geological domains created in Datamine from 303 original holes and 483 deflections. A process of geological modelling and creation of grade shells using indicating kriging was completed in the estimation process.

7. The estimation of Mineral Resources has taken into account environmental, permitting and legal, title, and taxation, socio-economic, marketing and political factors.

8. The Mineral Resources may be materially affected by metals prices, exchange rates, labor costs, electricity supply issues or many other factors detailed in the Company’s Annual Information Form.

The data that formed the basis of the estimate are the drill holes drilled by PTM, which consist of geological logs, the drill hole collars, the downhole surveys and the assay data. The area where each layer was present was delineated after examination of the intersections in the various drill holes.

There is no guarantee that all or any part of the Mineral Resource not included in the current reserves will be upgraded and converted to a Mineral Reserve.

1.12 Mineral Reserves Estimates

The effective date for the mineral Reserve estimate contained in this report is 17 October 2016.

On review by the Qualified Person for Reserves, Robert L Goosen (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves. The final access to the minerals will require permits from the Department of Mineral Resources ("DMR"), acquisition of surface rights, water use license, securing of power and a social license to operate as established in a Social and Labor Plan.

The QPs are not aware of unique characteristics related to this Project that would prevent the granting of such permits and satisfied with progress towards the timing of submission of these applications where applicable. The mineral rights are held under Prospecting Permits with the exclusive right to apply for a Mining Right.

The Mineral Reserve statement for the Waterberg project is based on the South African Code for the Reporting of Exploration Results, Mineral Resource and Mineral Reserves (SAMREC code). There is no material difference between the SAMREC and CIM 2014 code for Mineral Reserve estimation in this case.

Figure 1-6 sets out the framework for classifying tonnage and grade estimates to reflect different levels of geoscientific confidence and the different degrees of technical and economic evaluation. Mineral Resources can be estimated based on geoscientific information with input from relevant disciplines.

Mineral Reserves, which are a modified sub-set of the Indicated and Measured Mineral Resources in order of increasing confidence, are converted into Probable Mineral Reserves and Proven Mineral Reserves (shown within the dashed outline in Figure 1-6), require consideration of factors affecting extraction, including mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors ('modifying factors'), and should in most instances be estimated with input from a range of disciplines.

A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve, which is the economically mineable part of an Indicated Resource, and in some circumstances a Measured Resource. This is demonstrated by at least a Pre-Feasibility Study ("PFS") including adequate information on mining, processing, metallurgical, and economic and other factors that demonstrate, at the time of reporting, the economic extraction can be justified.

A Proven Reserve is the economically mineable part of a Measured Resource demonstrated by the same factors as above. A Proven Mineral Reserve implies that there is a high degree of confidence. Not all mining and permit approvals need be in place for the declaration of Reserves.

Abridged definitions are given below in Section 2.5.

The SAMREC code definition of a Mineral Reserve is:

"A 'Mineral Reserve' is the economically mineable material derived from a Measured, or Indicated Mineral, resource or both. It includes diluting and contaminating materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a Pre-Feasibility Study for a project and a Life of Mine Plan for an operation must have been completed, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors (the modifying factors). Such modifying factors must be disclosed."

Mineral Reserves are reported as inclusive of diluting and contaminating uneconomic and waste material delivered for treatment or dispatched from the mine without treatment.

The CIM 2014 code definition for a Mineral Reserve:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported."

For this technical report, the Mineral Reserves for the Waterberg project have been stated under the SAMREC Code with no material difference to the CIM 2014 standards. The point of reference is ore delivery to the RoM silo at the processing plant.

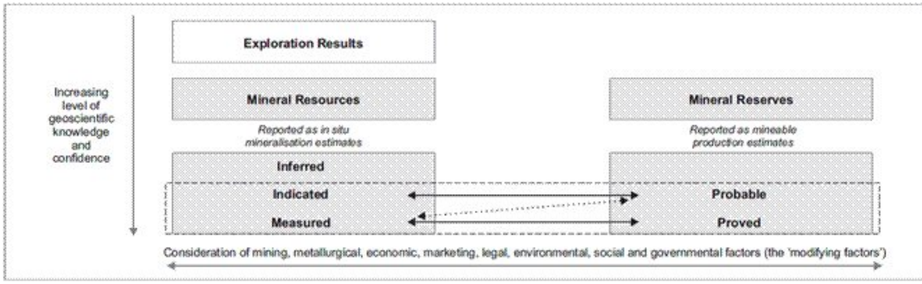


Figure 1-6: Relationship between Mineral Resources and Mineral Reserves

The conversion to Mineral Reserves was undertaken initially at 3.0g/t and the 2.5 g/t 4E stope cut-off grade for both for the T and the F-Zones, which considered costs, smelter discounts, concentrator recoveries from the previous and ongoing engineering work completed on the property by the Company and its independent engineers. Spot and three-year trailing average prices and exchange rates are considered for the cut-off considerations. Initial mine plans were developed based on a 3 g/t 4E cut-off. At the end of the mine life material that was available at a 2.5 g/t 4E cut-off was considered in the full life of mine.

From the Mineral Resource as estimated in this report, each stope has been fully diluted, comprising of a planned dilution and additional dilution for all aspects of the mining process. There are no inferred Mineral Resources included in the Reserves.

The Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian).

Table 1-4 show the Prill splits which are calculated using the individual metal grades reported as a percentage of the total 4E grade.

Table 1-4: Prill Splits

	Prill Split				Grade	
	Pt	Pd	Au	Rh	Cu	Ni
Zone	%	%	%	%	%	%
T-Zone	29	49	21	1	0.16	0.08
F-Zone	30	64	5	1	0.07	0.16

Table 1-5 and Table 1-6 show the total diluted and recovered Probable Mineral Reserve for the Waterberg project.

Table 1-5: Probable Mineral Reserve at 2.5g/t 4E Cut-off — Tonnage and Grades

Waterberg Probable Mineral Reserve — Tonnage and Grades

Zone	Mt	Cut-off grade (g/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)	Cu (%)	Ni (%)
T-Zone	16.5	2.5	1.14	1.93	0.83	0.04	3.94	0.16	0.08
F-Zone	86.2	2.5	1.11	2.36	0.18	0.04	3.69	0.07	0.16
Total	**102.7**	**2.5**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**	**0.08**	**0.15**

Table 1-6: Probable Mineral Reserve at 2.5g/t 4E Cut-off — Contained Metal

Waterberg Probable Mineral Reserve — Contained Metal

Zone	Mt	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	4E (Moz)	4E Content (kg)	Cu (Mlb)	Ni (Mlb
T-Zone	16.5	0.61	1.03	0.44	0.02	2.09	65 097	58.21	29.10
F-Zone	86.2	3.07	6.54	0.51	0.10	10.22	318 007	132.97	303.94
Total	**102.7**	**3.67**	**7.57**	**0.95**	**0.12**	**12.32**	**383 103**	**191.18**	**333.04**

Reasonable prospects of economic extraction were determined with the following assumptions: Metal prices used in the reserve estimate are as follows based on a 3-year trailing average (as at July 31/2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance was used for the assessment of Resources and Reserves; US$1,212/oz Pt, US$710/oz Pd, US$1229/oz Au, US$984/oz Rh, US$6.10/lb Ni, US$2.56/lb Cu, US$/ZAR15. Smelter payability of 85% was estimated for 4E and 73% for Cu and 68% for Ni. The effective date is October 17, 2016. A 2.5 g/t Cut-off was used and checked against a pay-limit calculation. Independent Qualified Person for the Statement of Reserves is Mr. RL Goosen (WorleyParsons RSA (Pty) Ltd Trading as Advisian). The mineral reserves may be materially affected by changes in metals prices, exchange rates, labor costs, electricity supply issues or many other factors. See Risk Factors in 43-101 report on www.sedar.com and on the Company's Annual Information Form. The reserves are estimated under SAMREC with no material difference to the CIM 2014 definitions in this case.

1.13 Geotechnical Investigations

1.13.1 Ground Conditions

The site is covered by five identified soil profiles (Kalahari sand, ferruginised Kalahari sand, colluvium, alluvium and strongly cemented calcrete) across the proposed site.

The DCP test results confirm that the transported material layer found from 0.5m below ground level has an allowable bearing capacity of at least 50kPa.

The permanent water table was not encountered during this investigation.

The transported Aeolian material encountered on the site is generally suitable for use in engineered layer work applications. Further testing would be necessary if proposed for use.

Soft to medium hard rock sandstone and strongly cemented calcrete pan can be expected at shallow depth below ground level. Some variation can be expected over the site. Blasting may be required to maintain the lines and levels of services and foundations depending on the design depths.

The sidewalls of the trial pits were relatively stable during the investigations.

1.13.2 Foundations

According to the trial pits/rotary core drilling investigation and the laboratory test results, the site is classified as a "H1/S2/C2/R" site in the NHBRC Classification, with an expected range of total soil movements more than 20mm. The assumed differential movement is 50%.

1.13.2.1 Light Structures* (100 — 150kPa)

Remove the soil to a depth of 1.6m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 150kPa.

1.13.2.2 Medium Structures* (150 — 250kPa)

Remove the soil to a depth of 3m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 250kPa.

1.13.2.3 Heavy Structures* (250 - 500kPa)

Remove the soil to a depth of 4m below surface or up to the bedrock. The excavation must then be backfilled with G5 materials in in 0.200m thick layers; compacted to 93% mod ASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 500kPa.

Notes*: Soil raft foundation with good site drainage is recommended. Ninety-three percent compaction is a reasonable expectation. Anything above that might not be achievable during construction. Soil mattresses will have to be found on dense sand (>100kPa) as a minimum.

1.13.2.4 Primary and Secondary Surface Crushers

Spread foundations founded on the bedrock are considered feasible. Allowable bearing capacity of at least 5MPa, which is generally suitable for a crusher structure, was confirmed with the point load test results. The recommended founding level was identified at 4.21m depth below natural ground level in the borehole WB130. Good founding material (medium hard rock sandstone) will have to be validated by a competent person during construction.

1.14 Mine Plan

1.14.1 Geotechnical Factors

Prior to the commencement of the PFS, additional geotechnical data was obtained through core logging of recently drilled boreholes. The revised geotechnical, database, which includes laboratory strength test results, was used to determine rock properties and classify the rock mass. This information was used together with available geological information to construct a 3-dimensional geotechnical rock mass model. The geotechnical rock mass model together with other pertinent information informed aspects of mine design. Input parameters derived from this work were used in idealized numerical models to evaluate various mining configurations and mine sequencing and to augment the empirical evaluations that were conducted.

Some elementary geological interpretations were made to help inform mine design.

The potential for surface displacement resulting from underground mining was assessed with elementary numerical models and it was found that the likelihood of surface subsidence is very low.

The potential for raisebore instability was assessed based on a few boreholes not necessarily near any proposed ventilation raise bore location. There could be challenges, however better informed assessments can only be made based on dedicated geotechnical boreholes at each location.

The two mining methods proposed, BLR and SLOS were assessed and are substantially feasible as long as control is exercised diligently.

Critical hydraulic radii were calculated for open span designs and pillar dimensions were determined based on empirical methods and numerical modelling. In an attempt to optimize extraction, the designs for Waterberg are in a “transition” zone between indefinite stability on the one hand and definite caving on the other.

Based on the rock mass classification and using the Q-system, guidelines for ground support in main access excavations, main and secondary on reef roadways and on reef drifts have been developed.

All the work contributed to the development of a set of rock mechanics parameters for mine design.

Current risks and opportunities to the project associated with mine design have been identified and listed and a set of recommendations for the way forward have been compiled.

1.14.2 Mining Methods selected

The wireframes resulting from the MSO runs were used to create artificial footwall and hanging wall contact zones from which the mine design could be digitized.

Three mining methods Blind Longitudinal Retreat, “BLR” Transverse Sub-level open stoping “TSLOS” and Longitudinal Sub-level open stoping “LSLOS”) were selected for the project as they satisfy the following design criteria:

- Minimize the schedule required to achieve full production with stope sequencing;
- Required production volumes;
- Opex/Capex cost;
- Optimize recovery and minimize dilution;
- Maximize flexibility and adaptability based on size, shape, and distribution of target mining areas; and
- Prevent surface subsidence from underground mining.

The criteria for each of these methods are detailed below, but can be resumed by the following table:

Table 1-7: Mining Method Criteria

Mining Method	Dip	Vertical Thickness
BLR	≤ 35°	3 - 15m
LSLOS	> 35°	3 - 15m
TSLOS		> 15m

The MSO wireframes provided the boundaries to which each mining method is applied. These boundaries along with the artificial contact zones were used in Studio 5D Planner to create the detailed mined design.

The design maximized the recovery of material identified from MSO while keeping to geotechnical guidelines proposed by rock engineering, thus all geotechnical losses were designed for and would not require additional factors.

To obtain initial tonnage and grades, the mine design was evaluated against the block model and the results were exported to EPS for scheduling and reporting.

From the Mineable Shape Optimizer model, ore bodies were delineated by resource characteristics and potential mining methods were selected and derived for each defined mining area through a process of option identification and ranking, and adapted to the rock conditions, including:

- Geometry of orebody;
- Geological complexities;
- Geotechnical properties of the country rock and orebody; and
- Depth below surface of extraction.

The mine is designed to initially develop the high-grade zones to minimize pre-production development capital and maximize early revenues. Further optimization for grade is an opportunity with more detailed mine designs in the Definitive Feasibility stage. Final resource to Reserve reconciliation checks was completed. The QP is satisfied with the Reserve data and has verified the data for the Reserve estimate.

1.14.3 Mine Design Access

The top of mining zones in the current Waterberg mine plan occur at depths ranging from 170m to approximately 350m below surface.

The majority of development is done by mechanized equipment on the ore horizon due to the orebody and various mining methods.

Access to the mine will be via three decline shafts, to service the various zones namely:

- T-Zone : Portal Position - South;
- F Central : Portal Position — Central;
- F Boundary and F North : Portal Position — North.

The design philosophy applied to the Waterberg project followed an approach of proven designs and results of various trade-off studies and was designed to accommodate a mine plan, which ramps up to 7.2 Mtpa.

Practical consideration of the real estate purchases and protection of heritage resources were considered in the selection of surface infrastructure.

The study has concluded that the dual decline option has lower capital cost and lower long-term operating costs and provides a more flexible and easily expandable solution for initial mine access and production ramp-up, as well as an opportunity to achieve higher production rates in the event that resource growth is confirmed.

Other key access design objectives met are:

- To access the workings in a way this minimizes capital development; and
- To facilitate an aggressive production build up, targeting the high-grade areas as quickly as possible.

Various ventilation holes from surface will also be required to provide a ventilation egress point.

1.14.3.1 Portal and Declines

Initial access into the mine would be via portals that service the twin declines.

The dimensions of the main access declines are 6.0 m (W) x 6.0 m (H), while the main conveyor declines have dimensions of 5.5 m (W) x 5.5 m (H). The declines will dip at -9°, generally in an easterly direction. Figure 1-7 shows the position of the portals in relation to the surface infrastructure. The dimensions have been based on the conveyor design, ventilation intake requirements and sizes of equipment.

Positioning the portal as shown, will facilitate quick access to the shallower parts of the ore body, which will reduce the time to ‘first ore’. In addition, the portal position allows quick access to the higher-grade areas of the Waterberg mining area.

Portal designs were created based on professional experience in similar ground environment and geotechnical information gathered from the inspection of four boreholes drilled near the proposed portals location.

Laboratory tests were conducted to confirm the on-site investigation and establish preliminary engineering parameters for the soils and rocks.

The suggested preliminary portals designs will have to be supported and approved with the finite element and limit equilibrium methods during the Definitive Feasibility Study (DFS) to reach an acceptable Factor of Safety (FoS) determined for the project.

The proposed portals designs were conducted in a manner consistent with the level of care and skill ordinarily exercised by members of the geotechnical profession practicing under similar conditions in the locality of the project.

- Portals T-Zone and F Central

The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 37° inclination into soil material. The high wall is 20m high from the footwall position. The overall slope angles are 41° and 50° for the sidewalls and highwall respectively in the preliminary portal design. The top two benches have a height of 4m. The remaining benches are 6m high. The catch berms have a width of 3m across the highwall and sidewalls.

- Portal F North Zone

The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 36° inclination into soil material. The high wall is 35m high from the footwall position. The overall slope angles are 38° and 44° for the sidewalls and highwall respectively in the preliminary portal design. The first bench has a height of 5m. The remaining benches are 6m high. The catch berms have a width of 3m across the highwall and sidewalls.

Each mining method requires a different underground infrastructure, such as access development to sub-levels, loading points, ventilation shafts and silos. Together, they form intricate network of openings, drifts, ramps, shafts and slot raises, each with its designated function.

1.14.3.2 Mining Rates

The PTM Waterberg Project requires significant underground development in order to optimally access the ore body. Access to the high-grade areas of the mine is required as soon as reasonably possible in order to attain a maximized potential project value.

A mining cycle scheduling operation, derived from first principles, for cleaning, supporting, drilling and blasting was completed for various mining systems and face arrangements. This was done to test the theoretical possibility of attaining the required 100m per month system advance, which has been planned, whilst not conservative, is a consistently achievable target from both a theoretical and actual benchmarked operations perspective.

There is significant opportunity to increase the planned system advance rate in areas should it be possible to achieve multi-blast conditions during the course of the mine development. This would entail establishing an independent ventilation district that solely ventilates the development and is removed from stoping operations.

Figure 1-7 gives an overview of the portal positions and extent of strike and dip of the orebody.

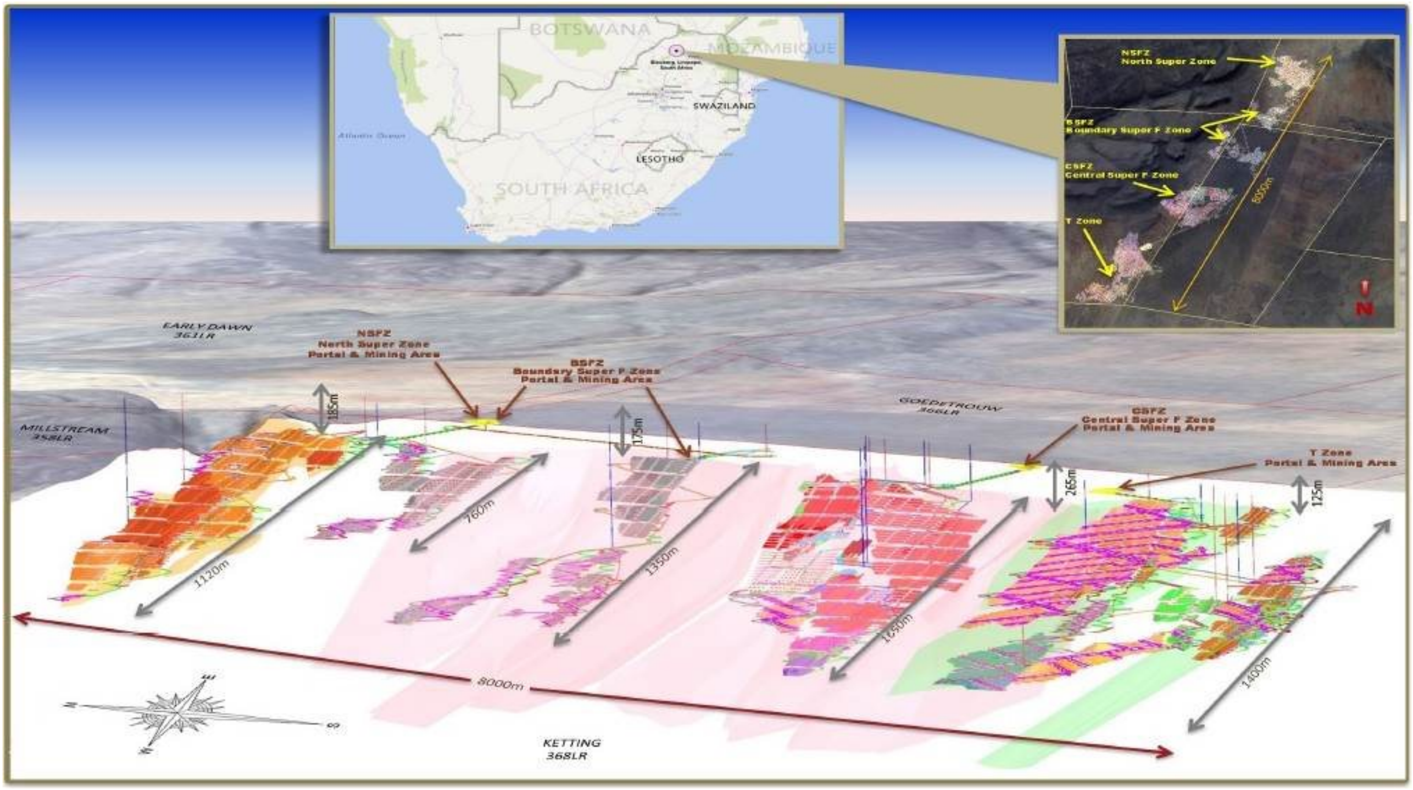


Figure 1-7: Portal and Underground Layouts

1.14.4 Production summary and schedule

The key average annual production results over the 18-year mine life are shown in Table 1-8

Table 1-8: Production Summary

Item	Units	Total
Mined and Processed	Mtpa	7.20
Platinum	g/t	1.11
Palladium	g/t	2.29
Gold	g/t	0.29
Rhodium	g/t	0.04
4E	g/t	3.73
Copper	%	0.08
Nickel	%	0.15
Recoveries		
Platinum	%	82.5%
Palladium	%	83.2%
Gold	%	75.3%
Rhodium	%	59.4%
4E	%	82.1%
Copper	%	87.9%
Nickel	%	48.8%
Concentrate Produced		
Concentrate	ktpa	285
Platinum	g/t	24.2
Palladium	g/t	51.5
Gold	g/t	4.9
Rhodium	g/t	0.6
4E	g/t	81
Copper	%	1.9
Nickel	%	1.8
Recovered Metal in Concentrate		
Platinum	kozpa	222
Palladium	kozpa	472
Gold	kozpa	45
Rhodium	kozpa	6
4E	kozpa	744
Copper	Mlbpa	11
Nickel	Mlbpa	12

Year 4 bases the mine plan on a multiple ramp access underground mining operation ramping up to 600ktpm where it remains for the majority of the LoM until the lower grade end period.

The current status of Life of Mine (LOM) throughput is based on an initial 3g/t 4E cut-off; thereafter, 2.5 g/t 4E will be applied in the final years of the mine life.

The tail of the production schedule for the Waterberg production starts in 2035 and final reef tonnes are scheduled for 2038.

The recommended throughput option for the Waterberg process plant is two modules of 300ktpm each. This configuration is sufficiently flexible to cater for the portal development scenarios and further provides flexibility to cater for both large and small mining operations if selected in future.

Total Mine production with the average grade is shown in Figure 1-8.

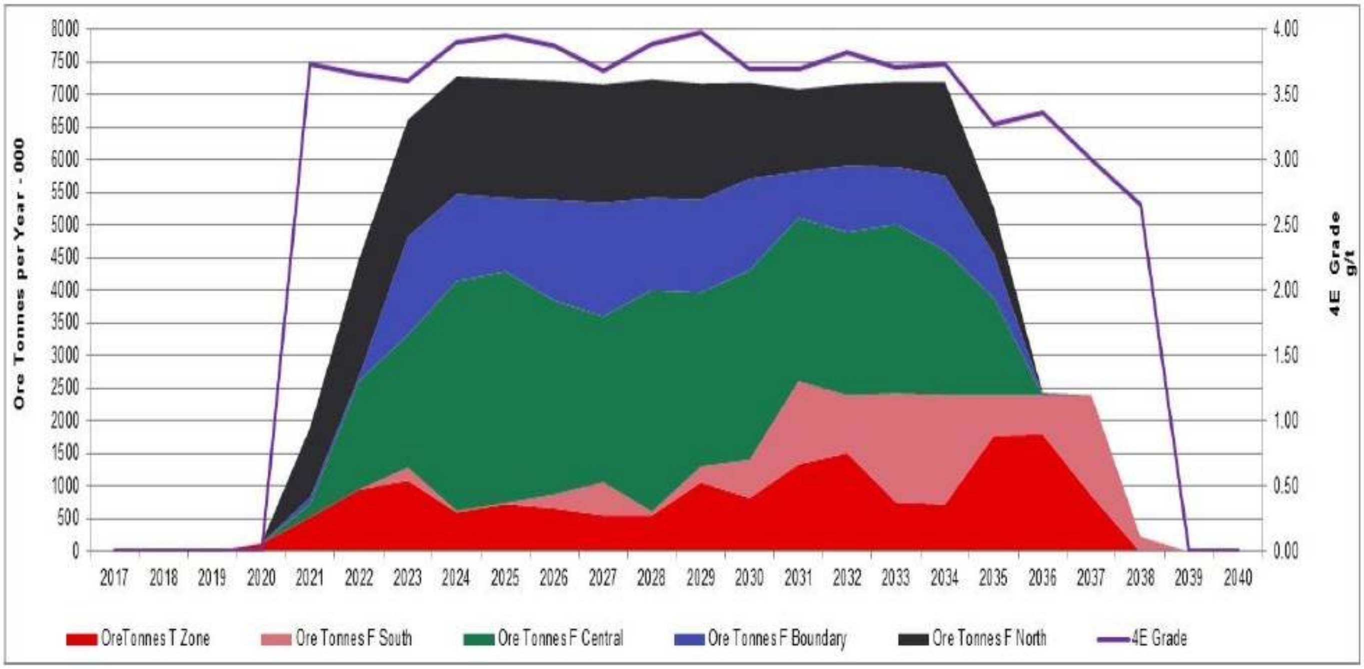


Figure 1-8: Mining Method Total Mine Production

1.14.5 Ventilation

The ventilation and cooling systems consider safety and health requirements in accordance with the Mine Health and Safety Act [MHSA, Act 29 of 1996].

Ventilation and cooling system designs are based on the production and development tonnage profiles and diesel fleet provided by the mine design team. The mining plan is based on steady state production of 600 000 reef tons per month, ventilation and cooling requirements for each mining area is phased-in accordingly over LoM.

Diesel equipment will be a significant heat source accounting for almost 40% of mine heat, in comparison heat flow from rock will account for less than 10% [maximum Virgin Rock Temperature VRT 46.0°C]. The balance will come from auto-compression and other sources including electrical. In mechanized mines, to a depth of approximately 700 mbs this heat can usually be removed by ventilation used to dilute exhaust gasses. However, beyond this depth, heat flowing into the mine from rock and other sources combined with heat from the diesel equipment means that generally, air alone cannot adequately cool the mine and additional mechanical cooling is required. It is confirmed that at depth T-Zone, F1 South, F2 Central, F4 Boundary North and F5 North additional cooling will be required

1.15 Metallurgical Test Work and Recovery

Various metallurgical test work campaigns have been conducted throughout the course of 2013 to 2016 to determine the optimum flowsheet for treatment of the various Waterberg ore lithologies. Metallurgical test work focused on maximizing recovery of PGEs and base metals, mainly copper and nickel, while producing an concentrate product of an acceptable grade for further processing and/or sale to a third party.

In 2013, preliminary metallurgical test work was undertaken at SGS (Booysens, South Africa) using two samples, F-Central and T-zone, taken from the Waterberg deposit as part of the Preliminary Economic Assessment. The results indicated that a potentially saleable concentrate could be produced. The results from the PEA test work program is summarized in the previous PEA technical report, filed in February 2014.

Further investigative test work was performed on an F-Central composite sample, under the management of JOGMEC during the course of 2013 to 2014. The results indicated that a concentrate product in excess of 100 g/t 4E could be produced at acceptable recoveries with the inclusion of Oxalic acid and Thiourea in the reagent suite.

As part of the PFS, extensive metallurgical test work was conducted at MINTEK, which focused on characterizing the various Waterberg lithologies in terms of mineralogical composition, comminution parameters, and flotation response.

Comminution tests have classified the Waterberg ores as hard to very hard and not suitable for Semi-Autogenous Grinding (SAG) milling.

Two flotation flowsheets were tested on each Waterberg lithology, a MF1 circuit utilizing Oxalic acid and Thiourea as part of the reagent suite and a MF2 circuit utilizing typical Southern African PGM reagents, such as SIBX as a collector. Batch open circuit flotation test work as well as locked cycle flotation test work was conducted. Encouraging results were obtained from both flowsheets. Test work results have demonstrated that some of the ore types respond better to a particular configuration. However, superior recoveries were obtained for the mine blend samples using the MF2 configuration, leading to the selection of the MF2 circuit for the process design.

It was noted that extensive scavenging and cleaning was required in the MF2 circuit to maximize recoveries, while lower mass pulls in the high grade and low grade circuits where essential to ensure acceptable concentrate grades were achieved and the product grade specification were met. Flotation work indicated that the optimum final grind for the F-zone ores are 80% passing 75µm; whilst there are evidence that the T-zone material could achieve higher recoveries at finer grinds of 85-90% passing 75µm. Further test work to investigate the optimization of the T-zone final grind is recommended.

The flotation test work indicated that the Waterberg ores are amenable to treatment by conventional flotation without the need for re-grinding. A standard flotation concentrator can be used to produce a saleable concentrate, at a 4E grade of no less than 80 g/t, with no deleterious products. 4E recoveries in excess of 80% are expected at the proposed mill feed grades.

1.16 Process Plant Design

The process design for the Waterberg Concentrator Plant has been developed based on the extensive metallurgical test work results, as well as other desktop level studies completed by the project team. A trade off study was conducted to determine the optimal production ramp up and steady state production. Based on the outcome of the study the plant steady state capacity of 7.2Mtpa will be achieved by the construction of the plant in two phases. Each phase consisting of a 3.6Mtpa concentrator module,

The Phase 1 3.6 Mtpa concentrator module and associated infrastructure, is planned to start production in month 36. Phase 2 includes the construction of the second 3.6 Mtpa module to take the total production to 7.2 Mtpa in month 53. The second concentrator module is designed as a copy of the first module, with minor exceptions with regards to shared infrastructure.

Each of these modules comprises a three-stage crushing circuit, feeding crushed material to the primary milling circuits. Primary milling is achieved in a ball mill with closed-circuit classification followed by a primary rougher flotation bank. The primary rougher concentrate is further upgraded in the primary cleaning/re-cleaning circuit to produce a high grade concentrate product. The primary rougher tailings is further liberated in the secondary milling circuit which consist of a ball mill with closed-circuit classification, before reporting to the secondary rougher and scavenger flotation circuit. The secondary rougher concentrate product reports to the secondary cleaning/re-cleaning stages to produce a medium grade concentrate, whilst the scavenger flotation concentrate is upgraded in the scavenger cleaning circuit to produce a low grade concentrate product. Each of the concentrate products are combined in the concentrate thickener for dewatering, followed by filtration. The flotation tailings products are thickened prior to beings disposed to the residue storage facility.

Refer to Figure 1-9 for an illustration of the above.

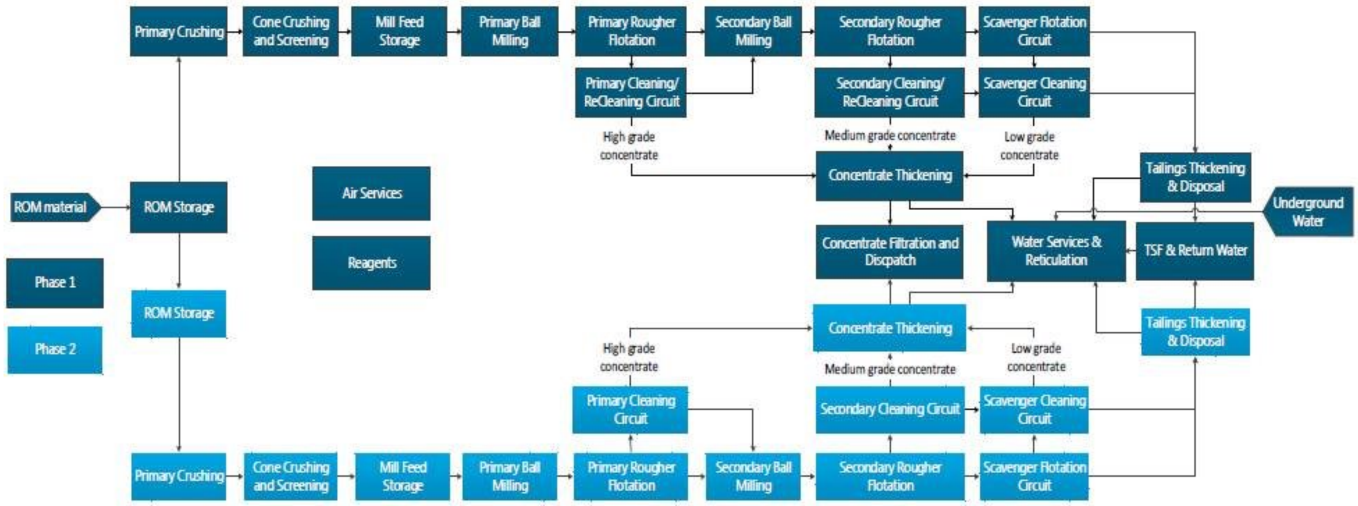


Figure 1-9: Waterberg Concentrator Block Flow Diagram

1.17 Infrastructure

The design philosophy applied to the Waterberg project followed an approach of proven designs and results of various trade-off studies.

The infrastructure was designed to accommodate a mine plan, which ramps up to 7.2Mtpa. Locations and sizing of infrastructures were significantly influenced by the geographical area. Real estate associated with cost, social, and cultural heritage considerations allowed little leeway for selection of locations. A site layout plan covering site facilities is shown in Figure 1-10.

The key infrastructure includes regional infrastructure, local infrastructure, central shared services, portal infrastructure as well as mine ventilation and refrigeration surface infrastructure as described in Section 18.

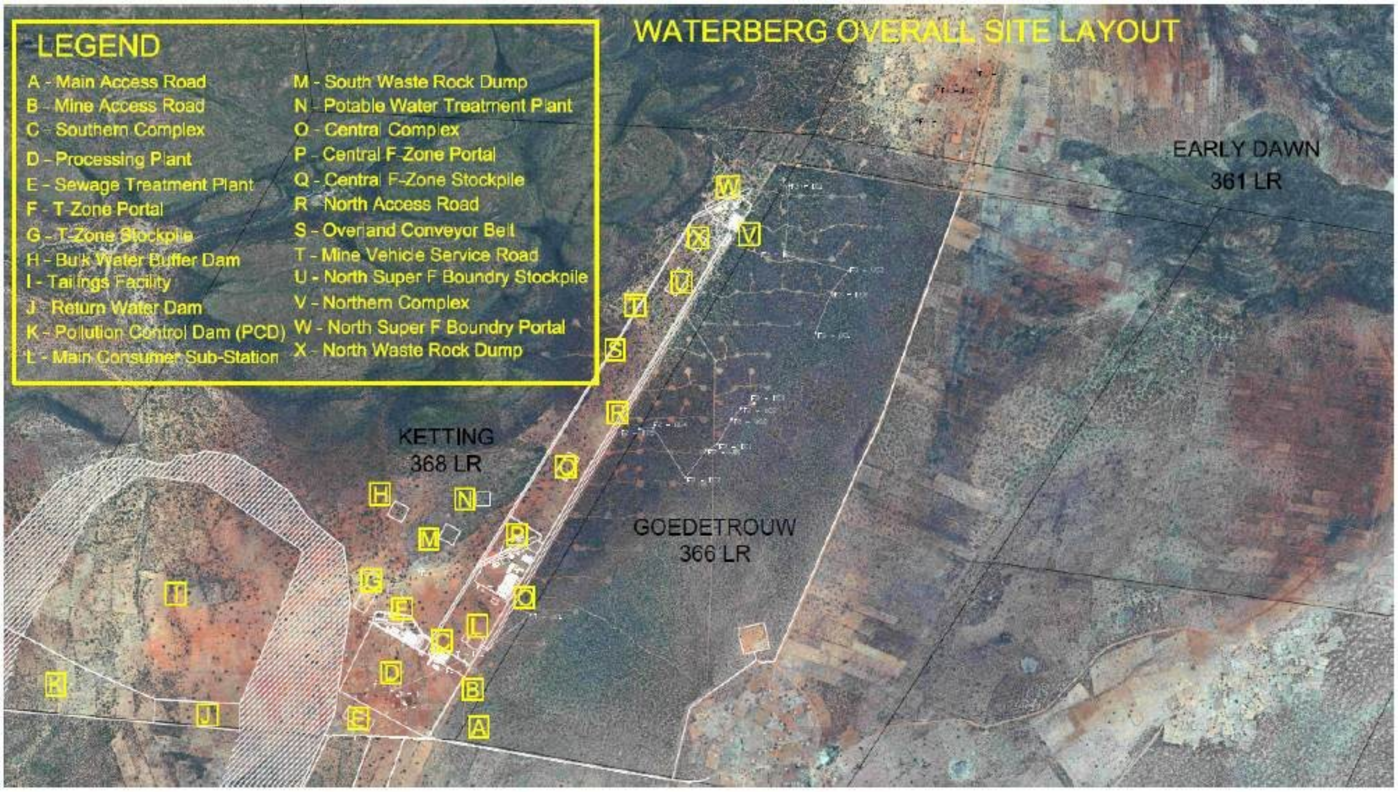


Figure 1-10: General Site Layout

1.17.1 Bulk Water Supply

South Africa is a country of relatively low rainfall and, in particular, the Limpopo province will require significant additional water capacity to meet the growing demand from the mining, agricultural, and domestic sectors. The Government has committed to addressing this shortage in the interest of developing the region. However, there are major planning, infrastructural design, and funding challenges that need to be addressed in order to ensure that sufficient bulk water supply is achieved.

The Olifants River Water Resource Development Project (ORWRDP) has been designed to deliver water to the Eastern Limb and Northern Limb of the Bushveld Igneous Complex (BIC) of South Africa. The ORWRDP consists of the new De Hoop Dam, the raising of the wall of the Flag Boshielo Dam, and related pipeline infrastructure, which will ultimately deliver water via Pruissen to Sekuruwe, located some 30kms to the north of Mokopane and 60kms south of PTM Waterberg Project. From this point, PTM Waterberg will need to develop their own pipeline project to take water to their site.

Implementation of the Flag Boshielo Pruizen pipeline has been put on hold because of funding issues and withdrawal of commitments from some mines due to low commodity prices. The PTM Waterberg project is located on the northern extremity of the ORWRDP area, the delay in implementation will result in PTM Waterberg not meeting their development schedule, and other options would need to be considered.

During the Pre-Feasibility Study, other bulk water supply options were considered. Other options considered were Glen Alpine Dam, transfer of water from Lephalala River, groundwater and effluent from various Waste Water Treatment Works (WWTW) including Louis Trichardt / Makhado and Seshego. The present water balance model simulations showed that the average bulk water supply requirement over the life of the mine would be 10.6 Ml/d.

Of all the water supply options considered a combination of sewage effluent and groundwater is considered the most viable and least risk solution to meet the proposed mining schedule. Wellfields with mainly poor water quality will be targeted so as not to compete with domestic water uses in the area.

From existing borehole information and limited exploration, drilling done to date about 0.5Ml/day of potable water or more could be developed around the mine site. Poor quality groundwater developed within 35kms east of the mine towards Bochum (about 5,5Ml/day) and to the south of the mine, some 4.3Ml/day is thought to be available. Non-potable groundwater resources up to 35kms from the mine could yield up to 9.9Ml/day.

1.17.2 Ground Water

The PTM Waterberg Project site and surrounding area is underlain by the Waterberg Group, Bushveld Igneous Complex and the Archaean Granite/Gneiss rocks. The Waterberg Group overlies the Bushveld Igneous Complex and comprise predominantly of sandstones. The base of the Bushveld Main Zone is characterized by the presence of a transitional zone that constitutes a mixed zone of Bushveld and altered sediments/quartzites before intersecting the Archaean granite basement. The Waterberg Sedimentary package has been intersected by numerous crisscrossing dolerite or granodiorite sills or dykes and act as preferential flow path for groundwater.

Groundwater abstraction in the area is mainly used for domestic consumption at the villages. Water levels in the area vary between artesian and 52m below ground level (mbgl). The groundwater quality does not always comply with the drinking water standards due mainly to the high salt content. Borehole yields vary considerably over the area with yields of up to 10l/s found along major structures in the Waterberg sediments and in the highly weathered and fractured Gneisses. However, due to the low rainfall, recharge to the aquifers is low with the average annual recharge estimated to be only about 12mm per annum.

Inflow into the proposed mine workings has been estimated to be between 3.6Ml/day and 9.4Ml/day depending on hydraulic conductivity of the deeper fault zones and the number of faults intersected. A conservative figure of 3.3Ml/day has been used in the water balance. These inflows will result in an impact zone around the mining lease area of about 6kms. Production boreholes serving communities within this zone could be affected.

From information available at this stage local groundwater around the mine could yield up to 0.5 Ml/day of potable water or more. Non-potable groundwater resources up to 35kms from the mine could yield up to 9.9Ml/day.

1.17.3 Bulk Power supply

The bulk electricity supply for the project is being planned to cater for mining and plant production rates of up to 600ktpm, which correspond to an electrical load of up to 160MVA. A temporary electrical supply is being planned for the construction stage.

Existing 66kV and 132kV networks approach to within 25km from the project site, however, it has been determined that the capacities of these networks are inadequate to supply the project load. The updated electricity supply plan compiled by Eskom therefore provides for the establishment of new 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation, which is located approximately 77km south of the project site. Eskom has confirmed in principle the availability of capacity from this system to supply the mine.

The proposed bulk electricity supply infrastructure comprises the following:

- Two 77km long 132kV overhead lines from Burotho transmission substation;
- Two 132kV line feeder bays for these new lines at Burotho transmission substation; and
- A 132kV switching substation and step-down substation located on the project site.

The development of the abovementioned infrastructure is being done in conjunction with Eskom on a Self-Build basis in terms of which Waterberg JV Resources is responsible for most of the development work.

This work is already in an advanced stage; with line route planning and environmental impact assessment work having progressed well (refer Figure 1-11, which shows some of the 132kV overhead line route options).

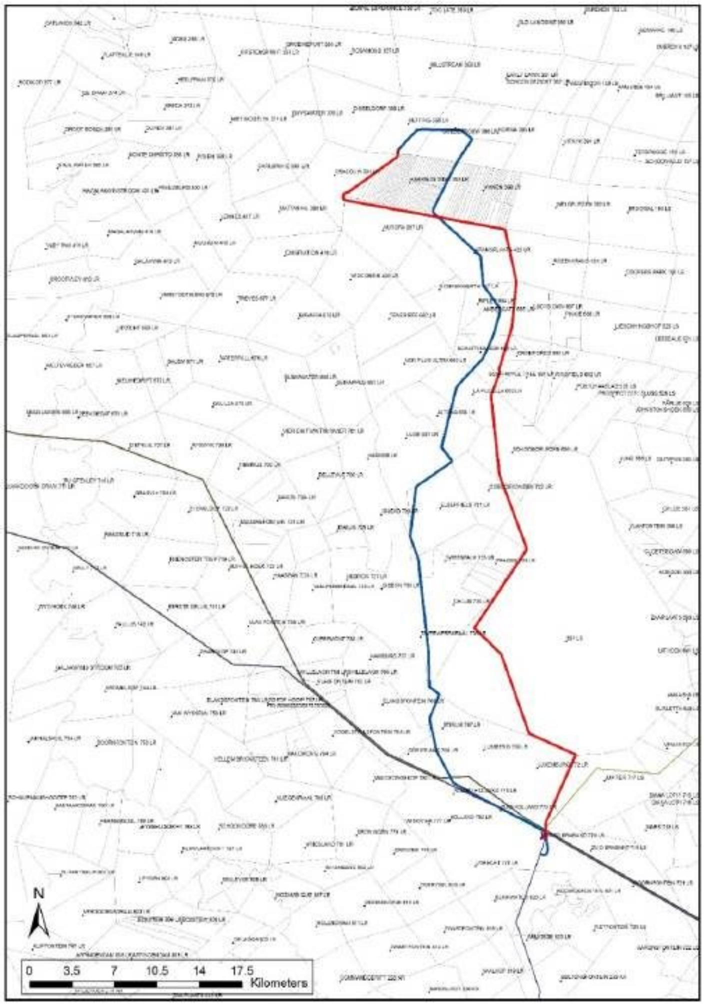


Figure 1-11: Proposed Overhead Line Route

1.17.4 Process Plant

Further to the equipment described in Section 1.16, the following permanent installations are also included to support the processing plant:

- Return water columns from the residue storage facility to the processing plant
- Plant services, i.e. compressed air and raw water
- Plant potable water storage and reticulation
- Plant electrical supply and reticulation, from the plant consumer substation.
- Plant offices
- Plant store
- Plant workshop
- Plant weighbridge

The plant infrastructure includes storm water berms and drains to divert rainwater from the plant and to collection rainwater falling in the plant in a pollution control dam, this water will be captured for use in the process plant and not intended to be discharged to the environment.

1.17.5 Residue Storage facility

A Pre-Feasibility Design (PFD) of the Residue Disposal Facility (RDF) and its associated infrastructure was undertaken. The design of the RDF comprising:

- A Residue Storage Facility (RDF) that accommodates 140 000 000 dry tonnes over a 20 year Life of Mine (LoM);
- A Return Water Dam (RWD) and/or Storm Water Dam (SWD associated with the RDF;
- The associated infrastructure for the RDF (i.e. perimeter slurry deposition pipeline, storm water diversion trenches, perimeter access road, etc.);
- Estimation of the capital costs to an accuracy of ±25%, operating costs associated with these facilities to an accuracy of ±25% and closure costs to an accuracy of ±35%; and
- Estimation of the costs over the life of the facility.

1.17.5.1 Site Selection

A site selection study was undertaken to find the most favorable site. The study found that Ketting farm was the most favorable.

1.17.5.2 Depositional Trade-off Study

A trade-off study was undertaken to determine a suitable depositional methodology as well as to highlight the advantages and disadvantages of each methodology. The following methodologies were investigated:

- Conventional/thickened tailings;
- Cycloned tailings;
- Paste tailings; and
- Dry-filtered tailings.

The following conclusions were drawn from the study:

- Paste disposal is untested in the platinum industry and would pose a significant risk and require an extensive testing regime to consider implementing;
- Dry stacking is a possible option and the potential water recoveries could make this option feasible, however the high capital and operational costs associated with dry stacking could make this potion infeasible compared to a conventional tailings dam;
- Cycloned tailings may provide a cost saving due to the higher rates of rise achievable, however test work is required prior to recommending this option;
- Conventional/thickened tailings are the safest option, well understood in the platinum industry, and have been regarded as the preferred option for Waterberg.

1.17.5.3 Economic Depositional Methodology Trade-off Assessment

Further to this, an Economic Assessment of the various depositional methodologies was undertaken to determine which methodology would provide a cost effective solution given that the scarcity of water at the site. The purpose of this assessment was to determine which option would result in the most cost effective solution in terms of water cost; therefore, the costs were only taken to a conceptual level. The results show that filtered tailings will only be feasible if the water cost exceeds R60/m^3.

Therefore, conventional/thickened tailings were taken forward as the preferred option for Waterberg.

- Key Design Features:

  The key design features of the RDF in Figure 1-12 are as follows:

  - The RDF will be constructed as an upstream, spigotting facility;
  - A compacted earth fill starter wall at elevation 1000m.a.m.s.l.;
  - A penstock system will be used to decant water from the RDF;
  - A RWD with sufficient capacity for the 1 in 50 year storm event (340 000m^3);
  - The RDF has a total footprint area of 297Ha, a maximum height of 55m and a final rate of rise of <3m/year;
  - A concrete lined solution trench to convey seepage water to the RWD;
  - Lined toe paddocks to collect contaminated run-off water from the RDF side slopes; and
  - A slurry spigot pipeline along the crest of the RDF.

Figure 1-12: RDF Layout

1.17.6 Access Roads

The Waterberg Project is located some 85km north of the town of Mokopane (formerly Potgietersrus) in Seshego and Mokerong, districts of the Limpopo Province. Although the bulk of the roads surrounding the site are provincial roads under the jurisdiction of the Roads Agency Limpopo (RAL), some of the minor roads are the responsibility of either the Capricorn District Municipality or the three relevant Local Municipalities.

The Waterberg Project is situated some 56km from the N11 national road that links Mokopane with the Groblers Bridge border post to Botswana. Access to the project area from Mokopane in Figure 1-13 (112km), and Polokwane in Figure 1-14 (94km) includes about 32km of unpaved roads.

It has been assumed in this study that this portion of the access route will remain unsurfaced but provision has been made for re-profiling and adequate drainage run-off along the route and a maintenance contract to maintain the road to an acceptable standard for the life of mine.

The balance of the route will have to be assessed to determine additional costs that may be incurred to upgrade and repair. The transport of the concentrate has been assumed to be done by contract haul and a rate per tonne component has been included in the financial model.

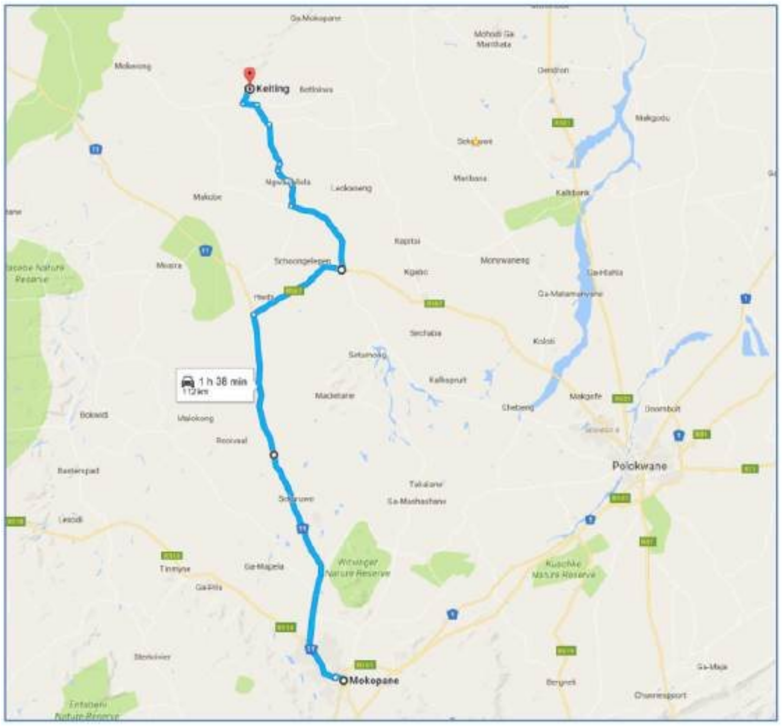


Figure 1-13: Access Route from Mokopane (112km)

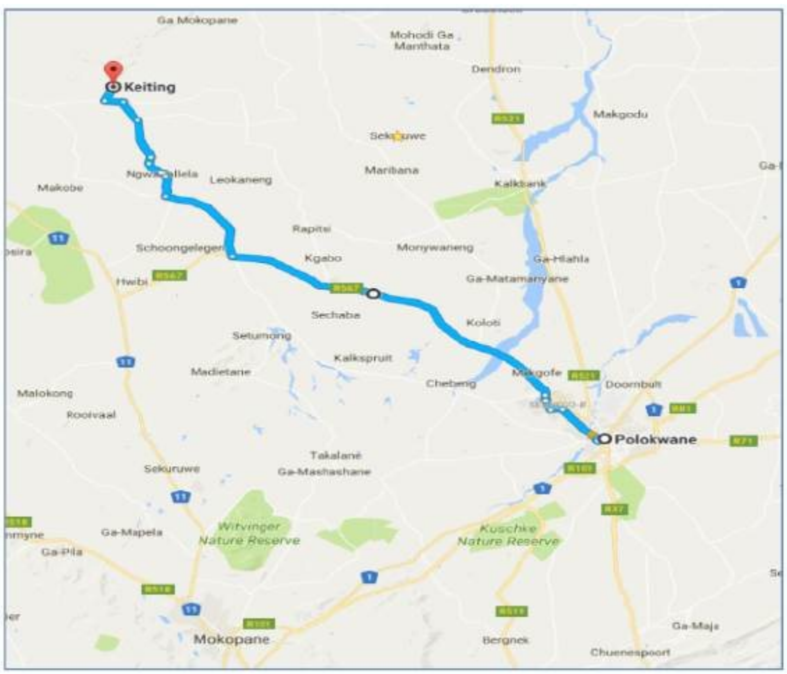


Figure 1-14: Access Route from Polokwane (94km)

1.18 Market Studies and Contracts

Either the Waterberg project will produce a flotation concentrate from the processing plant, which is assumed to be sold, or toll treated into the local South African market.

Production of up to 285 000 tonnes of concentrate per annum will be available at peak production. The concentrate will contain approximately 80g/t 4E's plus copper at between 1% and 9.2% and nickel at between 1.1% and 5%.The concentrate does not contain any penalty elements such as chrome and is rich in Sulphur, thus making it a desirable concentrate to blend with other high chrome concentrates.

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal indications from smelters are that the concentrate is attractive.

Based upon industry data, it is expected that the payability for the concentrate sold to a local smelter operator will be up to 85% for the PGE's, 73% for contained copper and 68% for contained nickel. It is expected that the metal will be available from the refinery after 16 weeks. Opportunity exists to have payment terms with "pipeline' finance facilities and these have been included in the study for the life of the mine.

1.18.1 Metal Prices

The Waterberg Project level financial model begins on 1 July 2016. It is presented in 2016 constant dollars, cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken. The base case real discount factor applied to the analyses is 8%. No allowance for inflation has been made in the analyses.

The following prices, based on a 3-year trailing average in accordance with U.S. Securities and Exchange Commission ("SEC") guidance, was used for the assessment of resources and Reserves.

The exchange rate between the ZAR and the USD is fixed at ZAR15.00:USD1.00 in the financial model throughout the LoM. The pricing and exchange rates above results in the estimated basket prices shown in Table 1-9 below.

Table 1-9: Average Three Year Trailing Metal Prices used in Financial Model

Parameter	Unit	Financial Analysis Assumptions
3 Year Trailing Average Price (Date: 31 July 2016)		
Platinum	US$/oz.	1 212
Palladium	US$/oz.	710
Gold	US$/oz.	1 229
Rhodium	US$/oz.	984
Nickel	US$/lb	6.10
Copper	US$/lb	2.56
Base Metals Refining Charge	% Gross Sales pay	85%
Copper Refining Charge	% Gross Sales pay	73%
Nickel Refining Charge	% Gross Sales pay	68%

Parameter	Unit	Financial Analysis Assumptions
Investment Bank Consensus Price (Date: 16 September 2016)		
Platinum	US$/oz.	1 213
Palladium	US$/oz.	800
Gold	US$/oz.	1 300
Rhodium	US$/oz.	1 000
Nickel	US$/lb	7.50
Copper	US$/lb	2.90

Investment Bank Consensus Sept, 2016 PGMs and base metals.

1.19 Environmental and Impact Assessment Studies

Preliminary environmental baseline studies has been completed for the Waterberg Project and measures have been incorporated in the development of the layouts, designs and operational practices to mitigate potential environmental risks.

The baseline studies included the following:

- Ground Water.
- Air Quality.
- Noise.
- Bio-Diversity.
- Soil.
- Visual Impact.
- Heritage Impact.
- Surface Water.
- Traffic.
- Blasting.

Prior to construction and operation of an underground mine, the following local legislative authorizations would be required:

- In support of a Mining Right Application (MRA), authorization in terms of Section 22 of the Mineral and Petroleum Resources Development Act, 2002 (Act No. 28 of 2002) (MPRD Act) by the Department of Mineral Resources (DMR) is required.
- Environmental Authorization as per the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEMA) and the Environmental Impact Assessment (EIA) Regulations (GNR. 543, 544 and 545 of 18 June 2010) from the Limpopo Department of Economic Development, Environment and Tourism (LEDET).
- A water use license in terms of Section 21 of the National Water Act, 1998 (Act No. 36 of 1998) from the Department of Water and Sanitation (DWS).
- A Waste Management License for categorized waste activities in terms of the National Environmental Management Waste Act, 2008 (Act No. 59 of 2008) (NEMWA) from the National Department of Environmental Affairs (DEA).

There have been discussions with the local communities and stakeholders regarding the environmental protection measures proposed to be undertaken.

The communities that are located within a 5km radius from the proposed project site are:

- Ga-Ngwepe.
- Setlaole.
- Ga-Masekwa.
- Ga-Raweshe.
- Ketting.

Consultations have also been held with the Regulatory Departments on various aspects of the Project and detailed discussions will continue throughout the permitting process and project execution.

A project risk assessment was carried out as part of the Pre-Feasibility Study to identify environmental sensitivities. The key risks potentially affecting the achievement of the project objectives were identified, together with their root causes and potential consequences. Primary mitigating strategies currently in place to address the risks were documented and where the current risk rating was considered unacceptably high, additional action items agreed to reduce it to an acceptable level.

1.20 Community Social Impact Assessment Studies

A social impact assessment is being conducted with the local communities to establish the social understanding within the area of the Waterberg mining operations. The project has maintained a positive open working relationship with the small communities in the area of the project including regular well documented meetings.

The communities that are located within a 5km radius from the proposed prospecting site are Ga-Ngwepe, Setlaole, Ga-Masekwa, Ga-Raweshe, and Ketting.

1.21 Capital and Operating Costs

1.21.1 Capital Costs

Project capital costs total ZAR 27,374M, consisting of the following:

- Initial Capital Costs — includes all costs to develop the property to a sustainable production of 600ktpm. Initial capital costs total ZAR 15,906M and are expended over a 72 month period from January 2017 to Dec 2022 including the pre-production construction and commissioning period; and
- Sustaining Capital Costs — includes all costs over the 16-year mine life related to expansion of production from the initial 300ktpm to 600ktpm and the acquisition, replacement, or major overhaul of assets required to sustain operations. Sustaining capital costs total ZAR 11,468 M and are expended in operating years from Jan 2023 to Jul 2038.
- The peak funding required for the project is estimated at ZAR13,694M (US$914M) in year 2022.

The costs are presented in ZAR 2016 and United States dollars (USD) market terms. It is presented in real money terms and no escalation was added. The base date for the Capital Estimate shall be 31 July 2016 and will be used to qualify the estimate in terms of governing laws, duties, taxes and tariffs.

The exchange rate between the ZAR and the USD will be fixed at ZAR15.00:USD1.00 in the Financial Model throughout the LoM.

The expected order of accuracy of the final estimate is in the range of ±25%

A 12% contingency allowance has been based on an assessment of the risk around the accuracy of the design information, quantities and rates applied using a Monte Carlo statistic process.

The estimate is presented in such a way that it is seamlessly incorporated into the financial model as an input, expressed in monthly cash flows for each WBS Level 1 facility code. Table 1-10 presents the PTM Waterberg capital at Level 1 WBS facility code.

Table 1-10: Total CAPEX

Facility Code	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
2000	Underground Mining	6,092	9,766	406	651
3000	Concentrator	2,850	159	190	11
4000	Shared Services & Infrastructure	1,063	43	71	3
5000	Regional Infrastructure	2,566	—	171	—
6000	Site Support Services	691	67	46	4
7000	Project Delivery Management	1,399	147	93	10
8000	Other Capitalised Costs	246	83	16	6
9000	Contingency	999	1,202	67	80
Total Capital		**15,906**	**11,468**	**1,060**	**765**

The facility level summary of the capital as well as the capital expenditure for LOM is depicted in Figure 1-15.

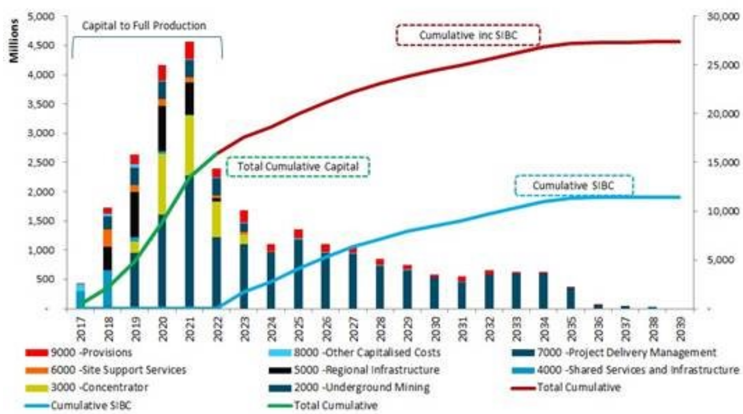


Figure 1-15: Total CAPEX Cashflow

1.21.2 Operating Costs

For the study, OPEX has been defined as:

- All on-reef development as soon as first stoping tonnes are achieved,
- Off-reef development associated with ongoing access and Reserve generation within, when first stoping tonnes are achieved. (These include sub-level off reef, lateral ventilation and other access development),
- All ongoing production related activities after first stoping ore is mined,
- Operating costs associated with the mobile mining equipment and fixed engineering equipment,
- Maintenance of mobile mining equipment and fixed engineering equipment.

Initially the mine will be contractor operated and once first stoping ore is mined for a particular mining zone, it will become owner operated. This excludes some contracted services over LoM such as raise bore, ventilation raises, silo and vertical dams, main access, primary conveyor decline and material decline development. The RDF facility will also be contracted out. The owner-mined operation per zone will coincide with when operating costs starts being incurred. All costs not associated to a particular mining zone will be reported under shared services and will include general, administration, and processing cost.

The operating cost model was developed by following the typical steps and processes prescribed by the Advisian RSA OPEX Estimation standards and methodologies. Methodologies utilized includes first principle costing for the labor, lifecycle costing for all equipment, infrastructure and fleet, zero-based costing for mining consumables and fixed/variable costing for the remainder of operating cost items.

The estimate methodology is aligned to preliminary engineering designs and budgetary quotations for major equipment and consumable cost and conforms to the +-25% accuracy level of a Pre-Feasibility Study. The operating cost estimate is modelled annually in ZAR. Costs reported in USD were converted from ZAR by using and exchange rate of R 15 per USD. A base date of July 2016 was used as costing basis. Costs are reported in real money terms with no escalations or contingency modelled. Quotes and cost rates were sourced from South African suppliers with foreign component cost not having an impact on the operating costs estimate.

The average LoM operating cost for the Waterberg Pre-Feasibility Study project is estimated at R 574.62 per ore tonnes broken (USD 38.31 /t). As indicated in Table 1-11, the total LoM cost amounts to R 58,99 billion (USD 3,93 billion). Average LoM costs are also detailed on a high level per area in ZAR and USD.

Table 1-11: Average LoM Operating Cost Rates and Totals per Area in ZAR and USD

	Average LOM (ZAR/t)		Total LOM (ZAR M)		Average LOM (USD/t)		Total LOM (USD)	
Mining	R	271.90	R	27 915	$	18.13	$	1 861
Engineering & Infrastructure	R	107.49	R	11 036	$	7.17	$	736
General & Admin	R	40.71	R	4 180	$	2.71	$	279
Process	R	154.52	R	15 864	$	10.30	$	1 058
Total OPEX Cost	**R**	**574.62**	**R**	**58 994**	**$**	**38.31**	**$**	**3 933**

The information in the table above is visually represented in Figure 1-16 to provide a better understanding of the breakdown per area of the LoM operating cost.

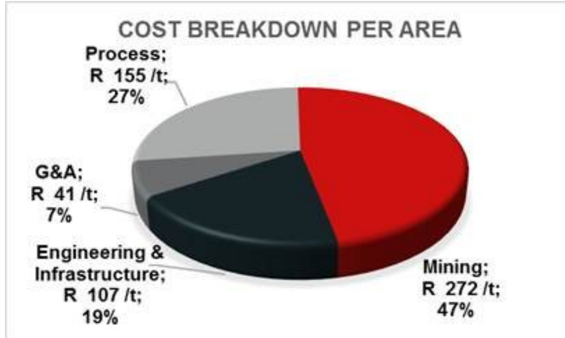


Figure 1-16: LoM Average R/t Operating Cost Breakdown per Area

From the figure, it is evident that mining comprise the bulk of the operating cost at 47%, followed by process at 27% and engineering and infrastructure at 19%. General and administration cost contributes a small portion (7%) of the total operating cost. The mining cost mostly driven by the large materials and supplies cost which is associated to development and production fleet maintenance (R 87/t) and consumables such as fuel (R 30/t). The process cost can be mostly attributed to the high power cost at R 64/t and consumable costs at R 60/t.

Figure 1-17 provides an overview of the operating cost per cost category over LoM. From the graphical representation, it is evident that the majority of costs remain constant. As expected, materials and supplies, cost will vary, as it is the directly related to the production profile.

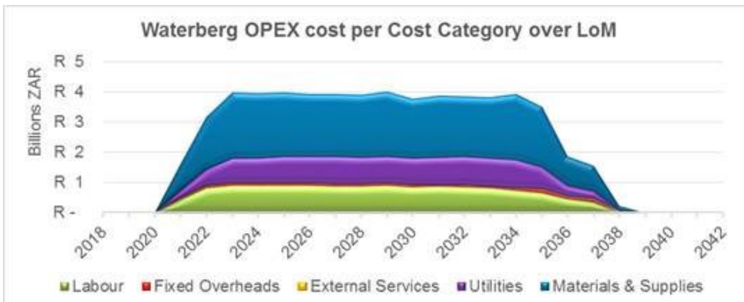


Figure 1-17: Operating Cost broken down per Cost Category over LoM

Figure 1-18 presents the total operating costs over LoM overlaid with the ore tonnage profile. The cost increase observed in 2022 is due to the start of the second process plant in November 2022 (month 53) combined with an increase in tonnage. Steady state is observed around 2024 when the process plant will process 7,2 Mtpa. The process, general, administration, engineering, and infrastructure operating cost remain constant throughout the LoM, whilst the mining operating cost closely resembles the tonnage profile. The two-phased ramp down starting in year 2035 is clearly visible towards the end of LoM.

The dip in operating cost displayed in year 2036 is a result of only one process plant being operational to process 200 ktpm for duration of approximately 17 months, until ore tonnes are depleted.

The operating cost model was developed to enable reporting per zone (e.g. F South), per area (e.g. mining) and per cost category (e.g. labor).

Figure 1-18: Operating Cost per Zone over LoM relative to Ore Tonnes

The operating cost model was developed to enable reporting per zone (e.g. F South), per area (e.g. mining) and per cost category (e.g. labor). For more operating cost detail and results, refer to Section 21.3.

1.22 Summary of Economic Analysis

The results of the financial analysis show an After Tax NPV 8% of ZAR4,805M. The case exhibits an after tax IRR of 13.5% and a payback period of around eleven years. The estimates of cash flows have been prepared on a real basis as at 1 July 2016 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 1-12.

The cumulative cash flow after tax is depicted in Figure 1-19.

Table 1-12: Financial Results Base Case Three Year Trailing Average

Item	Discount Rate	ZAR Millions (Before Taxation)	ZAR Millions (After Taxation)	USD Millions (Before Taxation)	USD Millions (After Taxation)
	Undiscounted	36,096	25,042	2,406	1,669
	4.0%	18,213	11,883	1,214	792
	6.0%	12,666	7,808	844	520
Net Present Value	8.0%	8,565	4,805	571	320
	10.0%	5,519	2,584	368	172
	12.0%	3,249	939	217	62
	14.0%	1,555	-278	104	-19
Internal Rate of Return		16.6%	13.5%	16.6%	13.5%
Project Payback Period (Years)		10	10	10	10

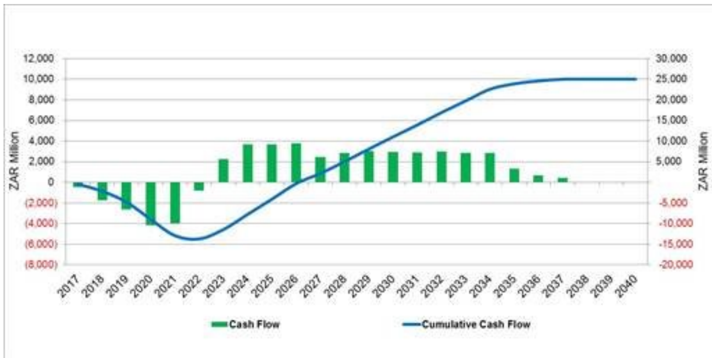


Figure 1-19: Annual Cashflow after Tax

Table 1-13: Investment Bank Consensus Price

Item	Discount Rate	Before Taxation (ZAR)	After Taxation (ZAR)	Before Taxation (USD)	After Taxation (USD)
	Undiscounted	45,781	31,946	3,052	2,130
	4.0%	24,180	16,184	1,612	1,079
	6.0%	17,426	11,263	1,162	750
Net Present Value	8.0%	12,402	7,610	827	507
	10.0%	8,641	4,884	576	325
	12.0%	5,812	2,842	387	189
	14.0%	3,676	1,311	245	87
Internal Rate of Return		19,8%	16.3%	19,8%	16.3%
Project Payback Period (Years)		9	9	9	9

1.23 Mineral Tenure, Surface Rights and Royalties

Currently there are no royalties, back-in rights, payments or other encumbrances that could prevent PTM from carrying out its plans or the trading of its rights to its license holdings at the Waterberg Project. JOGMEC or its nominee has the exclusive right to direct the marketing of the mineral products of the other participants for a 10-year period from first commercial production on an equivalent to commercially competitive arm's length basis and has the first right of refusal to purchase at prevailing market prices any mineral products taken by another participant as its share of joint venture output.

A summary of the mineral exploration and mining rights regime for South Africa is provided in Table 1-12. It should be noted that PTM have a Prospecting Right which allows them should they meet the requirements in the required time, to have the sole mandate to file an application for the conversion of the registered Prospecting Right to a Mining Right.

1.24 Conclusions

Results of this PFS demonstrate that the Waterberg Project warrants development due to its positive, robust economics, large production volume and opportunity relative to the PGM price deck.
.

It is the conclusion of the QPs that the PFS summarized in this technical report contains adequate detail and information to support a Pre-Feasibility level analysis.

Infill drilling over portions of the project area and new estimation methodology has made it possible to estimate a new mineral resource estimate and upgrade portions of the mineral resource to the Indicated category.

A Mineral Resource and Reserves may be declared for the PTM Waterberg project and reported in the tables below:

Table 1-14: T Zone Mineral Resource at 2.5 g/t 4E Cut-off

T-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Pd g/t	Au g/t	Rh g/t	4E g/t	Cu %	Ni %	Metal 4E Kg	Moz
Indicated	2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122 375	3.934
Inferred	2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75 485	2.427

Table 1-15: F Zone Mineral Resource at 2.5 g/t 4E Cut-off

F-Zone 2.5g/t Cut-off

Resource Category	Cut-off 4E g/t	Tonnage Mt	Grade Pt g/t	Pd g/t	Au g/t	Rh g/t	4E g/t	Cu %	Ni %	Metal 4E Kg	Moz
Indicated	2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651 670	20.952
Inferred	2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260 484	8.375

Table 1-16: Probable Reserve at 2.5 g/t 4E Cut-off

Zone	Mt	Moz	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)
T Zone	16.50	2.09	1.14	1.93	0.83	0.04	3.94
F South	10.32	1.26	1.14	2.42	0.19	0.04	3.78
F Central	36.75	4.24	1.08	2.30	0.18	0.04	3.59
F Boundary	16.08	1.94	1.12	2.40	0.19	0.04	3.75
F North	23.02	2.79	1.13	2.42	0.19	0.04	3.78
Total	**102.67**	**12.32**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**

The following prices, based on a 3-year trailing average in accordance with U.S. Securities and Exchange Commission (“SEC”) guidance, was used for the assessment of Resources and Reserves.

The Investment Bank Consensus price and spot price were also used for the Sensitivity analysis.

Table 1-17: Key Economic assumptions

Parameter	Unit	3 Yr Trailing Average 31 Jul 2016	Spot Price 6 Oct 2016	Investment Bank Consensus Price Deck 16 Sep 2016
Platinum	US$/oz.	1.212	964	1.213
Palladium	US$/oz.	710	668	800
Gold	US$/oz.	1.229	1.255	1.300
Rhodium	US$/oz.	984	675	1.000
Basket (4E)	US$/oz.	899	798	960
Nickel	US$/lb	6.10	4.52	7.50
Copper	US$/lb	2.56	2.17	2.90
Base Metals Refining Charge	% Gross Sales	85%		
Copper Refining Charge	% Gross Sales	73%		
Nickel Refinery Charge	% Gross Sales	68%		

The key features of the Waterberg 2016 PFS include:

- Planned steady state total and annual production and recoveries for the Mining zones are depicted in the table below.

Table 1-18: Waterberg 2016 PFS Production results.

Item	Unit	Total LOM	LOM Annual Avg
Ore Production			
Mineral Reserve	Mt	103	7.2
Ore Milled	Mt	103	7.2
T-Zone	g/t	3.94	3.94
F South	g/t	3.78	3.78
F Central	g/t	3.59	3.59
F Boundary	g/t	3.75	3.75
F North	g/t	3.78	3.78
4E	g/t	3.73	3.73
Copper	%	0.08	0.08
Nickel	%	0.15	0.15
Recoveries			
Platinum	%	82.5	82.5
Palladium	%	83.2	83.2
Gold	%	75.3	75.3
Rhodium	%	59.4	59.4
4E	%	82.1	82.1
Copper	%	87.9	87.9
Nickel	%	48.8	48.8
Recovered Metal			
Platinum	koz	3,029	222

Item	Unit	Total LOM	LOM Annual Avg
Palladium	koz	6,297	482
Gold	koz	715	45
Rhodium	koz	73	6
4E	koz	10,114	744
Copper	Mlb	168	11
Nickel	Mlb	163	12

Waterberg Key financial metrics are depicted in the table below:

Table 1-19: Waterberg 2016 PFS Results

Item	Units	Total
Key Financial Results (3 Year Trailing Price Deck — US$/ZAR 15) - 31 July 2016		
Life of Mine	years	19
Capital to Full Production	US$M	1060
Mine Site Cash Cost	US$/oz 4E	389
Total Mine Cash Costs After Credits	US$/oz 4E	248
Total Cash Costs After Credits	US$/oz 4E	481
All in Costs After Credits	US$/oz 4E	661
Site Operating Costs	US$/t Milled	38
After Tax NPV @ 8%	US$M	320
After Tax IRR	%	13.5
Project Payback Period (Start First Capital)	years	10
Investment Bank Consensus Price Deck- 16 September 2016		
After Tax NPV8	US$M	507
After Tax IRR	%	16.3

Standard industry practices, equipment and design methods were used in this PFS Study. The report authors are unaware of any unusual or significant risks, or uncertainties that would affect project reliability or confidence based on the data and information made available. For these reasons, the path going forward must continue to focus on drilling activities and obtaining the necessary permitting approval, while concurrently advancing key activities in the FS that will reduce project execution time.

Risk is present in any mineral development project. Feasibility engineering formulates design and engineering solutions to reduce that risk common to every project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, political risks, schedule and cost overruns, and labor sourcing. Opportunities include further optimization of the mine plan and potential reduction of development sustaining capital. The company indicates they will be focused on these aspects in the DFS phase.

The project provides attractive returns when compared to competitive projects in the Bushveld Complex in the Western or Northern Limb. Based on the competitive returns the project is recommended to proceed to the Definitive Feasibility Stage, ("DFS"). Drilling for measured resources should continue and be designed and budgeted along with the scoping process for the DFS.

1.24.1 Geology and Mineral Estimates

A Mineral Resource may be declared for the PTM Waterberg project. This Resource comprises an Indicated Resource of 31 Million tonnes at 3.88g/t 4E for the T-zone; and 186 Million tonnes at 3.49 g/t 4E for the F-zone. Additional Inferred Resources of 19 Million tonnes at 3.79g/t 4E for the T-zone and 77 Million tonnes at 3.37g/t 4E for the F-zone. These Resources are reported at a 4E grade cut-off of 2.5 g/t. Only Indicated resources are included in the mine plan and financial analysis.

1.24.2 Geotechnical and Rock Engineering

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

- The general geotechnical conditions are suitable for the planned infrastructure and the soil and rock is capable of supporting the planned structures.
- The geotechnical database was adequate for this level of study.
- The mining methods that have been identified as most suited are Blind Longitudinal Retreat (BLR) and Sub-Level Open Stoping (SLOS). These mining methods offer flexibility and with proper sequencing of mining cuts and support strategies, regional stability can be improved.

1.24.3 Mining

The mine design and production schedules presented are deemed as reasonable for a PFS level of confidence. Although, the BLR mining method is not widely utilized, it is the view of the project study team that the layouts and schedule rates are not overly aggressive.

A number of potential optimization opportunities have been identified and can be further quantified and expanded in the DFS.

1.24.4 Metallurgy

Sufficient test work to support the Waterberg Platinum pre-feasibility study has been undertaken.

Extensive metallurgical test work has been conducted on two different flowsheets, namely the MF1 and MF2 flowsheets, with encouraging results obtained from both. Test results have demonstrated that some of the ore types respond better to a particular configuration.

Bench scale test work conducted, on the Waterberg ores types and blends, has demonstrated that a saleable final concentrate containing at least 80 g/t 4E can be produced by applying a MF2 flowsheet and using standard Southern African PGM reagents. No deleterious elements are expected in the final concentrate, whilst 4E recoveries in excess of 80% are expected for the selected process design.

1.24.5 Infrastructure

For the purposes of this PFS, a range of options were considered for the on site and regional infrastructure.

The main infrastructure requirements for the Waterberg Project are access roads, residue disposal, water management, power supply and process plant to service and treat the targeted mine production.

The Waterberg Project is situated in a remote area and will require approximately 32km of existing unpaved roads to be surfaced.

A combination of sewage effluent together with groundwater is considered the most viable solution to meet the bulk water requirements of the proposed mining schedule. Wellfields with poor water quality will be targeted so as not to compete with domestic water uses in the area.

The bulk electricity supply for the project is being planned to cater for mining and plant production rates of up to 600ktpm, which correspond to an electrical load of up to 160MVA. A temporary electrical supply is being planned for the construction stage. Eskom has been engaged in the design process.

The availability of skilled labor resources, for both construction and operational phases, is limited and the training and skills development program will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Waterberg Project. The company plans to use its accredited training center.

1.24.5.1 Residue Storage Facility

The following conclusions were drawn from the study:

- A pre-feasibility design of the Residue Disposal Facility (RDF) for the Waterberg Project has been undertaken, in which:
  - A suitable site for the RDF has been identified;
  - conventional/thickened tailings is the safest option and well understood in the platinum industry and has been regarded as the preferred option for Waterberg;
  - a conventional/thickened RDF has been shown to be the most cost effective option for Waterberg in terms of water costs; and
  - The total LoM cost associated with the Waterberg RDF over the duration of the project life (Feasibility Study to Post Closure) is estimated at R1,057 million.

1.24.5.2 Bulk Water Supply

Of all the options considered, a combination of sewage effluent together with groundwater is considered the most viable solution to meet the proposed mining schedule.

Consider the bulk water source options as described is Section 19.3. The option of wellfields in combination with an effluent water pipeline from Bochum (Senwabarwama Ponds) is the most favorable with the least risk and is considered the base case. This infrastructure would allow the collection of water from various sources along the way, thereby ensuring a more sustainable bulk water supply to the Waterberg site.

The wellfields in combination with Waste Water Treatment Works (WWTW) pipeline from Bochum also creates the following opportunities:

- Access to groundwater from various wellfield areas along the route to supplement supply. This water is considered unsuitable for human consumption and would therefore have little impact on community water requirements;
- collection of water from smaller WWTW at Mogwadi;
- possible future expansion of the pipeline to collect effluent from Makhado WWTW

1.24.5.3 Bulk Power Supply

The updated electricity supply plan compiled by Eskom provides for the establishment of new 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation,

The development of the abovementioned infrastructure will be done in conjunction with Eskom on a Self-Build basis and this work is already in an advanced stage.

1.24.6 Market Studies and Contracts

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal contact by the Company is reported to indicate capacity and interest by two smelters. This will need to be confirmed in the DFS stage. Based on a comparison with the Merensky style of concentrate the Waterberg concentrate is considered attractive.

Based upon industry data, it is expected that the payability for the concentrate sold to a local smelter operator will be up to 85% for the PGE's, 73% for contained copper and 68% for contained nickel. It is expected that the payment terms will be full payment after 16 weeks for all metals, but with financing arrangements, these terms can be improved, but with significant interest charges for the up-front payment.

1.24.7 Environmental Impact Assessment Studies

The environmental permit, not yet approved, is of paramount importance, and delays from the company plan will increase project execution time. Without the permit advancement to a mining right with approval, the Project cannot proceed and failure to secure the necessary permits could stop or delay the Project. The project design considers the environment and local communities.

1.24.8 Community Social Impact Assessment Studies

The Community Social Impact Assessment Study is underway. It is focusing on all the three farms affected by the mining operations. This study is important because it assists in the compilations of the Social and Labor Plan (SLP). The SLP forms part of the Mining Right application process. Detailed consultation has been on going and is well documented.

The process for completing a Mining Right Application is underway. Discussions have been positive and business like. Both the community and the company have arranged experienced mining lawyers to facilitate the negotiations. The small community of approximately 100 homes will have to be relocated to the farm next to Ketting, which is also owned by the same community. This will require relocations costs. The MRPDA provides for a right of access and fair compensation will be required.

Allowance for land purchase and relocation costs was provided for the SLP in the Financial Model.

1.25 Recommendations

The QPs recommend that the Waterberg project advance to the DFS stage. The project financial model, including low capital cost per annual ounce of production and low operating costs provides the basis for further investment and refinement of the project design. The QPs recommend that based on the large scale PGM production profile of the project at 744,000 4E ounces per year that the project owners initiate discussions with smelters an investigate a standalone smelting option. The QPs also recommend that the owners initiate work towards an application for a Mining Right including the development of a Social and Labor Plan and environmental permits.

1.25.1 Geology and Mineral Estimates

It is recommended that exploration drilling continue in order to advance the geological confidence in the deposit through infill drilling. This will provide more data for detailed logging and refined modelling. This is expected to confirm the geological continuity and allow the declaration of further Indicated Mineral Resources.

Given the results of the diamond drilling on the northern area and the extent of target areas generated by geophysical surveys, the completion of the planned exploration drilling is recommended north of the location of the current exploration programme. The objective of the exploration drilling would be to find the limit of the current deposit, confirm the understanding of the F Zone and improve the confidence for a selected part of the deposit to the measured category for the DFS. Geotechnical and Rock Engineering

1.25.2 Geotechnical and Rock Engineering

The following is a list of work that will be required for a feasibility level of study. Although the list is comprehensive is by no means exhaustive.

- Additional trial pits should be excavated at the exact positions of the proposed structures during the Definitive Feasibility Study at the next stage. A diamond drilled triple tubes borehole should be undertaken at each surface crusher up to a depth of 45m or 10m into medium hard rock sandstone or stronger (>25MPa). Appropriate soil and rock laboratory testing should be part of the geotechnical investigation at this stage, including falling head permeability test of the in situ material for the clay/geosynthetic liner of the tailing dam.

- The T-Reef should be explored geotechnically in more detail.

- Sufficient data should be collected to allow for rigorous analyses of joints. This will include oriented core.

- A representative number of boreholes should be logged at selected locations to derive a more complete rock mass model that will inform designs of excavations away from the orebody as well as the main on-reef declines.

- With improved understanding of the model input parameters and the mining configuration, the assessment of the stability of the BLR designs, SLOS stopes and SLOS pillars can be conducted with greater confidence.

1.25.3 Mining

It is recommended that the opportunities mentioned in Section 16.12.2 be investigated further. This could be done prior to the next phase of the study or at least during the next DFS study phase.

- The mine design of underground access infrastructure, other underground excavations and production areas should be prepared to higher level of confidence for the DFS.
- Scheduling rates for development and production should be revisited to ensure that the rates planned remain realistic and achievable.
- Compile a detailed Bill of Quantities of the mine design and involve relevant mining contracting companies so that accurate cost estimates can be prepared.
- Conduct a simulation exercise that considers all underground logistics. It is recommended that this be done using an appropriate software package.
- Review the risks mentioned so that where possible adequate mitigating factors can be incorporated into the mine design and schedule.
- Complete a value engineering exercise on development and mining designs to reduce dilution and increase head grades.
- Waste development in sustaining capital should be studied for reduction with investigation and further detailing of the ventilation plan

1.25.4 Metallurgy

It is recommended that the opportunities mentioned in Section 17 be investigated further. This could be done prior to the next phase of the study or at least during the next study phase.

The following is also recommended for the next study phase:

- Flotation test work using water from the envisaged raw water sources to ensure the flotation performance is not negatively affected.
- Testing of the MF2 circuit using an Oxalic acid and Thiourea reagent scheme
- Comminution variability test work on the individual ore types
- Comminution variability test work on various possible mine blends
- Flotation open circuit batch variability test work on the individual ore types
- Flotation open circuit batch variability test work on various possible mine blends
- Concentrate thickening and filtration test work

Geotechnical investigation of the plant site to accurately determine founding conditions in the plant area and inform the design of the civil engineering works is also recommended.

The Definitive Feasibility Study would be completed using the test work results to optimize the process and infrastructure design and allow a more accurate assessment of the capital cost, operating cost and risks.

1.25.5 Infrastructure

Progress in-depth further infrastructure studies associated with access roads, supply and logistics, RDF design methodologies and any other areas of the Project where studies and confidence levels are lacking and for which information is required to support permitting and feasibility studies.

The Infrastructure component outlines a series of recommendations for the Project including progression to the Feasibility Study phase in order to assess the Waterberg development further including:

1.25.5.1 Residue Storage facility

For the Residue Disposal Facility Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- A geotechnical investigation of the RDF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials.
- A seepage analysis and slope stability study be undertaken to confirm the seepage regimes through the RDF as well as to confirm the RDF stability. The results of these analyses could affect greatly on the geometry of the RDF walls and ultimate height of the facility.
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample.
- Possible further optimization of the RDF preparatory works in terms of layout, footprint extent, etc. including any changes to the mine plan.
- Review the construction rates with a contractor to price the facility with representative rates.
- Compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy; and a hydrological study of potential flood lines near the RDF.

1.25.5.2 Bulk Water Supply

Due to the scarcity of water in the area, it will be critical to conduct more detailed hydrogeological investigations in order to identify in detail the potential groundwater resources that can be developed for mine supply and to predict the mine inflows and impact zone accurately. This will also be important to determine external bulk water requirements and the timing thereof. These hydrogeological investigations should include a numerical model, which will also assist the mine with monitoring and water management during the life of mine.

1.25.5.3 Bulk Power Supply

The electrical supply for the construction phase will involve the strengthening of an existing 22kV rural overhead line until the permanent supply infrastructure is in place. The 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation and the associated infrastructure would form part of the permanent supply infrastructure

1.25.6 Market Studies and Contracts

It is recommended that the local smelter operators be formally approached to better understand the appetite to consume the significant concentrate production once the mine is at steady state. A competitive process could be developed with the Japanese partner JOGMEC.

In addition, during the Definitive Feasibility Study, it is recommended that a Scoping Study be completed into the potential for the inclusion of a Waterberg Project Smelter on site. The product from this smelter could be a furnace matte or a convertor matte, which could be treated locally or exported for refining.

1.25.7 Environmental Impact Assessment Studies

The future development and delivery of the Waterberg Project will be underpinned by a programme of work for the mitigation of social and environmental impacts; creating value through good governance practices.

PTM has a programme of work in place to comply with the necessary environmental, social and community requirements, which include:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA;
- Public Participation Process (PPP) in accordance with the NEMA Guidelines;
- Specialist investigations in support of the ESIA;
- Integrated Water Use License Application (IWULA) in compliance with the National Water Act (NWA); and
- Integrated Waste Management License in compliance with the National Environmental Management Waste Act (NEMWA).

1.25.8 Community Social Impact Assessment Studies

The community impact assessment studies are being conducted and Platinum Group Metals and detailed documentation of the process is recommended to continue with appropriate specialists and counsel.

2. Introduction

2.1 Issuer

This report was compiled for Platinum Group Metals (RSA) (Pty) Ltd (PTM), a wholly owned subsidiary of Platinum Group Metals (Pty) Ltd (Canada) (PTML). Platinum Group Metals is listed on the Toronto stock exchange under the symbol "PTM" and on the NYSE MKT under the symbol "PLG.MKT"

2.2 Terms of Reference and Purpose of this Report

Platinum Group Metals (RSA) (Pty) Ltd (PTM) requested WorleyParsons RSA trading as Advisian, to complete an Independent Technical report on the Resources and Pre-Feasibility Study, updating the estimation of the inferred and indicated Mineral Resources for the Waterberg Project. The project targets an extension to the Northern Limb of the Bushveld Complex, which may have the potential for Platinum Group Metals (PGMs), gold and base metals (Cu, Ni).

This report was prepared in accordance with the requirements set forth in the National Instrument 43-101 Standards of Disclosure for Mineral Project (NI 43-101), Companion Policy 43-101CP to NI 43-101, and Form 43-101F1 of NI 43-101.

The Resource and Reserve Estimates completed in this report were completed within the guidelines of the South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC) published in 2007. For this report, these codes have no material difference to the CIM 2014 Guidelines.

The intentions of the report are as follows:-

- To inform investors and shareholders of the progress of the project;
- To provide sufficient confidence that further studies should be undertaken to further improve the confidence level of the Waterberg Project as a viable business case; and
- To make public and detail the Mineral Resource Estimation and Reserves for the project.

2.3 Sources of Information

Reports and documents listed in Section 3 of the Waterberg Project PFS were used to support preparation of the Report. Additional information was provided by PTM as supporting information for the QPs.

The Independent Author/Qualified Person ("QP") of this report has used the data provided by the representative and internal experts of PTM. This data has been derived from historical records for the area as well as information currently compiled by the operating company, PTM.

2.4 Involvement of the Qualified Person and Personal Inspections

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

The independent QPs (CJ Muller, GI Cunningham and RL Goosen) have visited the Waterberg property during 2016. Mr. RL Goosen visited the site on 13 October 2016, Mr. GI Cunningham on 27 and 28 March 2013 and 13 October 2016. Mr. CJ Muller on numerous occasions since 2012 and early in 2016. They have undertaken due diligence with respect to the PTM data. They have verified the data sufficiently for the reporting of resources, Reserves and this Pre-feasibility Study.

None of the QPs or any associates employed in the preparation of this report have any beneficial interest in PTM and neither is they insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between PTM and the QPs.

2.5 Frequently Used Acronyms, Abbreviations, Definitions and Units of Measure

Table 2-1 comprises of abbreviations used in the Waterberg Project PFS report and similarly with the metric units of measurements in Table 2-2.

The currency is expressed in US dollars unless stated otherwise.

Table 2-1: Abbreviations and Definitions

Abbreviation	Definition
2D	Two Dimensional
3D	Three Dimensional
4E	Platinum, palladium, rhodium and gold
AHP	Analytic Hierarchy Process
Ai	Abrasion Index
AMSL	Above Mean Sea Level
Au	Gold
BBE	Bluhm Burton Engineering
BBWi	Bond Ball Work Index
BC	Bushveld Complex
BCEA	Basic Conditions of Employment Act
BEE	Black Economic Empowerment
BLR	Blind Longitudinal Retreat mining method
BMS	Base Metal Sulphides
BRWi	Bond Rod Work Index
CapEx	Capital Expenditure
CCL	Compacted Clay Layer

Abbreviation	Definition
CIM	Canadian Institute of Mining
CLO	Community Liaison Officer
CoV	Coefficient of Variation
CSI	Corporate Social Investment
CW	Channel Width
CWi	Bond Crushability Work Index
DEA	National Department of Environmental Affairs
DEDECT	Department of Economic Development, Environment, Conservation and Tourism
DFS	Definitive Feasibility Study
DME	South African Department of Minerals and Energy
DMR	Department of Mineral Resources
DRA	DRA Minerals (Pty) Ltd
DTM	Digital Terrain Model
DWS	Department of Water and Sanitation
EBIT	Earnings Before Interest and Tax
ECD	Equivalent Circle Diameter
ED	Extraction Drive
EIA	Environmental Impact Assessment
EMP	Environmental Management Programme
EP	Equator Principles
EPS	Earthworks Production Scheduler
FPP	Pegmatoidal Feldspathic Pyroxenite
FS	Feasibility Study
FULCO	Full Calendar Operations
FW	Footwall
GCL	Geosynthetic Clay Liner
GCS	GCS Environment Engineering (Pty) Ltd
HDPE	High Density Poly-Ethylene
HDSA	Historically Disadvantaged South African
HLS	Heavy Liquid Separation
HW	Hanging Wall
IRUP	Iron Replacement Ultramafic Pegmatoid
IWULA	Integrated Water Use License Application National Water Act (NWA)
JOGMEC	Japanese Oil and Metals National Corporation
LCT	Leachate Concentration Test
LoM	Life of Mine
MF1	Mill-Flotation Circuit, Single Stage Milling followed by Flotation

Abbreviation	Definition
MF2	Mill-Flotation-Mill Flotation Circuit, Two Stage Milling followed by a Single Flotation Circuit
MHSA	Mine Health and Safety Act 29 of 1996
MMF	Mill-Mill-Flotation Circuit, Two Stage Milling followed by a Single Flotation Circuit
MPRDA	Mineral and Petroleum Resources Development Act, No 28 of 2002
MRA	Mining Right Application
MSO	Mineable Shape Optimizer
NEMA	National Environmental Management Act
NEMWA	National Environmental Management Waste Act
NFPA	National Fire Protection Association — America
NI 43-101	Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects
NPV	Net Present Value
NWA	National Water Act
OK	Ordinary Kriging
OpEx	Operational Expenditure
PBA	Public Benefit Activities
Pd	Palladium
PFC	Power Factor Correction
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
PGE	Platinum Group Element
PR	Prospecting Right
Pt	Platinum
PTM	Platinum Group Metals (RSA) (Pty) Ltd
PTML	Platinum Group Metals (Pty) Ltd (Canada)
Ptn	Portion
PXNT	Pyroxenite
QA/QC	Quality Assurance and Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Elecron Microscopy
QP	Qualified Person
RBH	Raisebore hole
RDF	Residue Disposal Facility (includes infrastructure, i.e. RDF, RWD, etc.)
Re	Remaining Extent
Rh	Rhodium
RoM	Run of Mine
RWD	Return Water Dam

Abbreviation	Definition
SACNASP	South African Council for Natural Scientific Professionals
SAMREC Code	South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (2007)
SCADA	Supervisory Control and Data Acquisition
SD/SDV	Standard Deviation
SEM	Scanning Electron Microscopy
SG	Specific Gravity
SIBX	Sodium Isobutyl Xanthate
SIPX	Sodium Isopropyl Xanthate
SK	Simple Kriging
SLOS — L	Sub-level Open Stoping — Longitudinal mining method
SLOS - T	Sub-level Open Stoping — Transverse mining method
SLP	Social and Labour Plan
SMC	SAG Mill Comminution
SMU	Selective Mining Unit
SRP	Stepped Room and Pillar
SWD	Storm Water Dam
TCT	Total Concentration Test
UCS	Uniaxial Compressive Strength
USD	United States Dollar
WHIMS	Wet High Intensity Magnetic Separation
WML	Waste Management License 12/9/11/L628/7
WUL	Water Use License 03/A22F/ABCGIJ/2596
WULA	Water Use Licence Application
XRD	X-ray Diffraction
ZAR	South African Rand

Table 2-2: Units of Measure

Unit	Definition
°	Degrees
°C	Degrees Celsius
°F	Degrees Fahrenheit
cm	Centimetre
cm.g/t	Centimetre grams per tonne
g/t	Grams per tonne
h	Height
ha	Hectare
kg/s	Kilograms per second

Unit	Definition
km	Kilometre
km^2	Square kilometres
ktpa	Kilo tonnes per annum
ktpm	Kilo tonnes per month
m	Metre
m/mth	Metres per month
m^3/s	Cubic metres per second
Moz	Million ounces
MVA	Mega Volt Amperes
MW	Mega Watts
Ø	Diameter
ppb	Parts per billion
t	Tonnes
t/m^3	Tonnes per cubic metre
tpa	Tonnes per annum
tpm	Tonnes per month
w	Width

2.6 Specific Areas of Responsibility

The QPs accept overall responsibility for the entire report. The QPs were reliant, with due diligence, on the information provided by PTM's internal and non-independent experts. The QPs have also relied upon the inputs of the PTM personnel in compiling this filing.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions:

- RL Goosen is a Principal Consultant - Mining Engineering with the firm WorleyParsons RSA (Pty) Ltd trading as Advisian, 39 Melrose Blvd, Melrose Arch, Johannesburg 2176, South Africa. He is a co-author of this report and is responsible for sections in Table 2-3 below.

- CJ Muller is a Principal Geological Consultant with CJM and has been involved with the Waterberg Project since 2012. He is a co-author of this report and is responsible for Sections in Table 2-3.

- GI Cunningham is an independent Metallurgist who consults for a number of companies. He has 22 years' experience in metallurgical production and more than 5 years as corporate metallurgist. He has been consulting as a private metallurgical consultant for 14 years. He has completed a number of independent PFS and BFS studies. He is a Practicing Metallurgical Consultant with PTM and other companies.

The QPs are responsible for specific sections in Table 2-3 below:

Table 2-3: Areas of Responsibility

	QP name		
Report section(s)	**CJ Muller**	**RL Goosen**	**GI Cunningham**
1	X	X	X
2	X	X	X
3	X	X	X
4		X	
5		X	
6	X	X	
7	X		
8	X		
9	X		
10	X		
11	X		
12	X		
13			X
14	X		
15		X	
16		X	
17			X
18		X	X
19			X
20			X
21			X
22			X
23		X	
24	X	X	X
25	X	X	X
26	X	X	X
27	X	X	X

2.7 Effective Dates

There are a number of effective dates for the information included in the Report, as follows:

- Mineral Resource Update on Waterberg Project : 17 October 2016
- Base date for Capital and Operation Cost Estimates : 1 July 2016
- Base date for Metal Prices : 31 July 2016
- Base date for Marketing Information : 31 July 2016

3. Reliance on Other Experts

- Mine Ventilation and Cooling Design for the Waterberg Project was compiled by Bluhm Burton Engineering for information derived through the following documents: BBE report no. 16020-TR-001-(R0);
- Marketing and Contracts for the Waterberg Project was compiled by Turnberry Projects (Pty) Ltd;
- Ownership and Permitting status supplied by Platinum Group Metals RSA (Pty) Ltd. Legal tenure specialists;
- Platinum Group Metals RSA (Pty) Ltd has provided legal tenure specialists royalty and taxes assumptions for royalties and taxes;
- Process plant design and Mineralogical test work was compiled by DRA Mineral Projects ;
- Mintek for all metallurgical testing and associated analyses, under the direction of DRA Minerals and Turnberry Projects for PTM;
- Mineral Resource Estimation was compiled by Mineral Resources CJM Consulting (Pty) Ltd; for information derived and supplied by PTM;
- Geological and assay information supplied by PTM;
- Test work analytical and survey data compiled by PTM;
- Residue Disposal Facility and Associated Infrastructure for the Waterberg Project compiled by Epoch Resources (Pty) Ltd.; for information derived through the following documents: Pre-Feasibility Study of the Residue Disposal Facility and Associated Infrastructure for the Waterberg Project;
- Independent Environmental studies which are filed with the DMR for the Waterberg Project" have been compiled by Bateleur Environmental & Monitoring Services for information derived through the following documents:
  - Two yearly Environmental Monitoring and Reporting in terms of the MPRDA; and
  - Annual Financial Provision Determination reports in respect of the financial guarantees in terms of the MPRDA.
- Community and Social Assessment supplied by Platinum Group Metals RSA (Pty) Ltd.; and
- All other applicable information, and data supplied by other persons and organizations as referenced.

The sources of information were subjected to a reasonable level of inquiry and review. The QPs have been granted access to all information. The QPs conclusion, based on diligence and investigation, is that the information is representative and accurate.

This report was prepared in the format of the Canadian NI 43-101 Technical Report by the QPs;

- CJ Muller
- GI Cunningham
- RL Goosen

These individuals are considered Qualified Persons under NI 43-101 definitions. The QPs have reported and made conclusions within this report with the sole purpose of providing information for PTM's and PTML's use subject to the terms and conditions of the contract between the QPs and PTM.

The contract permits PTML to file this report, or excerpts thereof, as a Technical Report with the Canadian Securities Regulatory Authorities or other regulators pursuant to provincial securities legislation, or other legislation, with the prior approval of the QPs. Except for the purposes legislated for under provincial securities laws or any other securities laws, other use of this report by any third party is at that party's sole risk and the QPs bear no responsibility.

The QPs are not qualified to offer legal opinion on title and offer no opinion as to the validity of the titles claimed. The description of the properties and ownership is provided for general purposes only and was supplied by PTM. The QPs were satisfied with the title to the extent required for the statement of Resources and Reserves and this report.

4. Property Description and Location

4.1 Property and Title

The Waterberg Project is some 85km north of the town of Mokopane (formerly Potgietersrus). The project consists of a prospecting license to the following properties:

- Kirstenspruit 351LR.
- Niet Mogelyk 371LR.
- Carlsruhe 380LR.
- Bayswater 370LR.
- Disseldorp 369LR.
- Ketting 368LR.
- Goedetrouw 366LR.
- Various other adjacent farms beyond the resources and Reserves as listed below.

PTM RSA applied for the original 137km prospecting right for the Waterberg JV area and in September 2009, the DMR granted the prospecting right until 1 September 2012. This prospecting right was later increased in size to 153km by way of Section 102 application to the DMR. Renewal of this prospecting right for a further three years ending 29 September 2018 was granted by the DMR in September 2015. Two further prospecting rights totaling 102km were granted to PTM RSA on 2 October 2013. These two prospecting rights are valid until 1 October 2018 and may each be renewed for a further period of three years thereafter.

The Waterberg Extension property includes contiguous granted and applied for prospecting rights with a combined area of approximately 864km. All of the Waterberg properties were included in one agreement 26 May 2015 for property interests and funding with JOGMEC and Mnombo. Section 11 transfers to the Joint Venture Company are in progress.

4.2 Property Ownership

Land ownership in the area of the proposed development is shown in Table 4-1 below:

Table 4-1: Land Ownership

Farm Name	Owner
Niet Mogelyk 371 LR	Government of Lebowa/Republic of South Africa
Kirstenspruit 351 LR	Republic of South Africa
Bayswater 370 LR	Republic of South Africa
Disseldorp 369 LR	Various individuals and Trusts.
Ketting 368 LR	Various individuals and Trusts.
Carlshue 390 LR	Government of Lebowa/Republic of South Africa
Goedetrouw 366 LR	Various individuals and Trusts.

4.3 Type of Mineral Tenure

A summary of the mineral exploration and mining rights regime for South Africa is provided in Table 4-2 below. It should be noted that PTM have a Prospecting Right which allows them should they meet the requirements in the required time, to have the sole mandate to file an application for the conversion of the registered Prospecting Right to a Mining Right.

Table 4-2: Mineral Tenure

Summary of Mineral Exploration and Mining Rights in South Africa	
Mining Act	Mineral and Petroleum Resources Development Act, No. 28 of 2002.
State Ownership of Minerals	State custodianship.
Negotiated agreement	In part, related to work programme and expenditure commitments.
Mining Title/License Types	
Reconnaissance Permission	Yes
Prospecting Right	Yes
Mining Right	Yes
Retention Permit	Yes
Special Purpose Permit/Right	Yes
Small Scale Mining Rights	Yes
Reconnaissance Permission	
Name	Reconnaissance Permission
Purpose	Geological, geophysical, photo geological, remote sensing surveys. Does not include “prospecting”, i.e. does not allow disturbance of the surface of the earth.
Maximum area	Not limited.
Duration	Maximum 2 years.
Renewals	No and no exclusive right to apply for prospecting right.
Area reduction	No.
Procedure	Apply to Regional Department of Mineral Resources.
Granted by	Minister.
Prospecting Right	
Name	Prospecting Right.
Purpose	All exploration activities including bulk sampling.
Maximum area	Not limited.
Duration	Up to 5 years.
Renewals	Once for 3 years.
Area reduction	No.

Summary of Mineral Exploration and Mining Rights in South Africa

Procedure	Apply to Regional Department of Mineral Resources.
Granted by	Minister.
Mining Right	
Name	Mining Right.
Purpose	Mining and processing of minerals.
Maximum area	Not limited.
Duration	Up to 30 years.
Renewals	Yes, with justification.
Procedure	Apply to Regional Department of Mineral Resources.
Granted by	Minister.

4.4 Surface Rights

Under a common-law position previously in force in South Africa, a landowner was the owner of the whole of the land, including the air space above the surface and everything below it. The MPRDA replaced this common law position.

Although the MPRDA does not specifically indicate the Republic of South Africa as the owner of unmined minerals, the ability of a landowner to exercise absolute rights over minerals found on or under their land has been nullified. A landowner retains the ultimate surface rights ownership, but not the minerals ownership.

In terms of Section 5 of the MPRD Act, the holder of a prospecting right is entitled, among other things, to enter the land to which the right relates together with its employees, to bring machinery and equipment onto the land to lay down and erect infrastructure, to prospect and carry out activities incidental to prospecting.

Prior to the Mineral and Petroleum Resources Development Amendment Act 29 of 2008 coming into effect on 7 June 2013, it was required, before the holder may commence with prospecting, to notify and consult with the owner or lawful occupier of the land. The owner or lawful occupier of the land is entitled to compensation for losses and damages suffered or likely to be suffered because of the proposed prospecting operation.

In the absence of an agreement between the holder and the owner or lawful occupier, compensation for losses and damages must be determined by arbitration or a competent court.

PTM has undertaken extensive consultation with the communities who are the lawful occupiers of the prospecting area, and surface use and cooperation agreements regulating among other things the compensation for losses and damages had been entered into.

A summary of the prospecting rights for the Waterberg and Extension Exploration Project held in the Limpopo Province and their status is summarized in Table 4-3 and their location on Figure 4-1 and Figure 4-2.

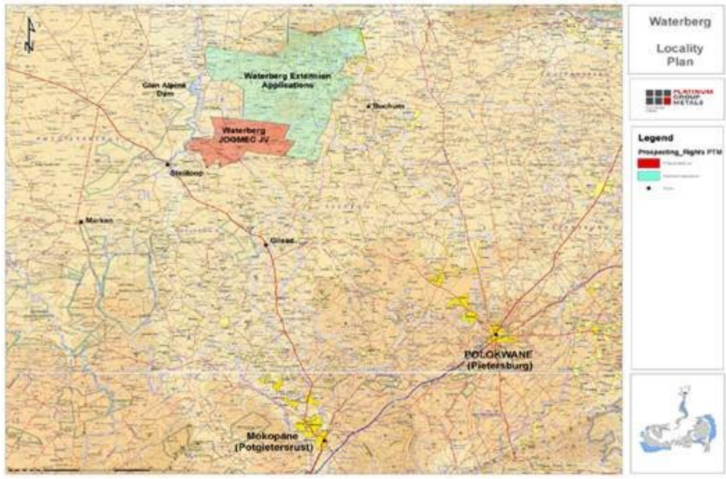


Figure 4-1: Location of the Waterberg Project Properties

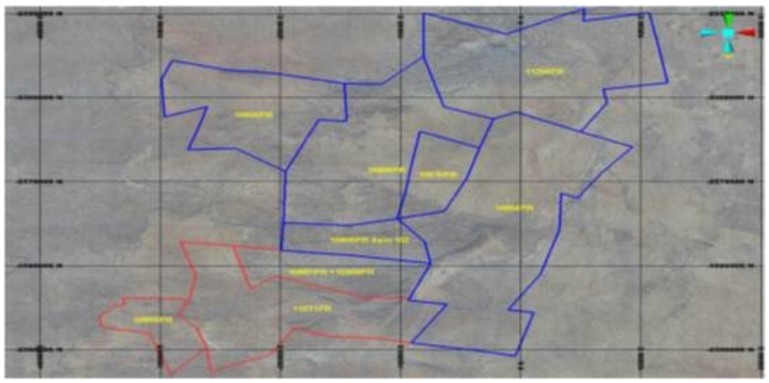


Figure 4-2: Location of the Waterberg Project Prospecting Rights

Table 4-3: Integrated Schedule of Prospecting Rights held by Platinum Group Metals (RSA) (PTY) LTD

Holder	DMR PR Reference	Description of Property	Hectares	Period of Prospecting Right	Minerals	Environmental Management Programme	Status
				SCHEDULE "A" — WATERBERG PROJECT			
PTM	11031 PR (New SAMRAD No) Original JV	The Farms: Niet Mogelyk 371 LR Kirstenspruit 351 LR Bayswater 370 LR Disseldorp 369 LR Ketting 368 LR Carlsruhe 390 LR Goede-trouw 366 LR	15257.00 HA	Renewal of Prospecting Right granted for 3 years with effect from 7 Sep 2015 to 6 Sep 2018	PGM's, Gold, Chrome, Nickel, Copper, Molyb-denum, Zinc, Cobalt, Lead, Silver and Rare Earths.	Environmental Management Programme authorised 7/09/2051	Renewal granted to 6 Sep 2018 Filed for registration in the Mineral Titles Office-Awaiting Registration
PTM	11031 PR (New SAMRAD No) New DMR Ref. 11013 PR 528 for this section 102 application Original JV	The Farms: Niet Mogelyk 371 LR Kirstenspruit 351 LR Bayswater 370 LR Disseldorp 369 LR Ketting 368 LR Carlsruhe 390 LR Goede-trouw 366 LR	15257.00 HA	Section 102 application was filed with the DMR for the inclusion of additional minerals. It will have the same benefits as 11013 PR when granted	Vanadium & Iron		Being adjudicated to in the DMR's office
PTM	10667 PR Original JV	The Farms: Groenepunt 354 LR Rosamond 357 LR Millstream 358 LR	6254.80 HA	Prospecting Right granted for 5 years with effect from 2 Oct 2013 to 1 Oct 2018	PGM'S, Gold, Chrome, Nickel, Copper, Cobalt, Lead, Molyb-denum, Rare Earths, Silver & Zinc	Environmental Management Programme authorised 2/10/2013	Notarially executed on 2/10/2013 under Protocol No. 1117/2013 Registered in Mineral & Petroleum Titles Office on 21/11/2013 under MPTO No 153/2013

Holder	DMR PR Reference	Description of Property	Hectares	Period of Prospecting Right	Minerals	Environmental Management Programme	Status
PTM	10668 PR Original JV	The Farms: Breda 373 LR Duren 387 LR Polen 389 LR	3953.05 HA	Prospecting Right granted for 5 years with effect from 2 Oct 2013 to 1 Oct 2018	PGM'S, Gold, Chrome, Nickel, Copper, Cobalt, Lead, Molyb-denum, Rare Earths, Silver & Zinc	Environmental Management Programme authorised 2/10/2013	Notarially executed on 2/10/2013 under Protocol No. 1118/2013 Registered in the Mineral & Petroleum Titles Office on 28/11/2013 under MPTO No 161/2013
PTM	10668 PR Original JV New DMR Ref. 10668 PR 523 for this section 102 application	The Farms: Breda 373 LR Duren 387 LR Polen 389 LR	3953.05 HA	Section 102 application was filed with the DMR for the inclusion of additional minerals. It will have the same benefits as 10668 PR when granted	Vanadium & Iron		Being adjudicated to in the DMR's office
SCHEDULE "B" — WATERBERG JOINT VENTURE EXTENSION							
PTM	10804 PR Extension	The Farms: R/E Norma 365 LR Portion 1 Norma LR PTN 2 & R/E Uitkyk 394 LR Portion 1 of Goede-trouw 366 LR Schoongezicht 362 LR Early Dawn 361 LR Old Langsine 360 LR Barenen 152 LS Landbryde 324 LR Lomondside 323 LR Rittershouse 151 LS	26961.59 HA	Prospecting Right granted for 5 years with effect from 2 Oct 2013 to 1 Oct 2018	PGMS , Chrome, Copper, Gold, Nickel, Vanadium & Iron	Environmental Management Plan authorised 2/10/2013	Notarially executed on 2/10/2013 under Protocol No. 1119/2013 Registered in the Mineral & Petroleum Titles Office on 10/09/2015 under MPTO No 106/2015

Holder	DMR PR Reference	Description of Property	Hectares	Period of Prospecting Right	Minerals	Environmental Management Programme	Status
		Miltonduff 322 LR Terwieschen 77 LS Brodie Hill 76 LS Willhandshohe 78 LS					
PTM	10805 PR Extension	The Farms: R/E & Portion 1 of Liepsig 264 LR Blackhill 317 LR Gallashiels 316 LR Bognafuran 318 LR Nieuwe Jerusalem 327 LR Sweethome 315 LR Mont Blanc 328 LR The Park 266 LR	17734.80 HA	Prospecting Right granted for 5 years with effect from 2 Oct 2013 to 1 Oct 2018	PGMS , Chrome, Copper, Gold, Nickel, Vanadium and Iron	Environmental Management Plan authorised 2/10/2013	Notarially executed on 2/10/2013 under Protocol No. 1120/2013 Registered in the Mineral & Petroleum Titles Office on 24 /04/2015 under MPTO No 49/2015
PTM	10806 PR Extension	The Farms: Normandy 312 LR La Rochelle 310 LR Les Fontaines 271 LR Springfields 268 LR Langlaagte 279 LR Berg-en-Dal 276 LR Windhoek 307 LR R/E Silvermyn 311 LR	13143.53 HA	Prospecting right granted for 5 years with effect from 30 Sep 2015 to 29 Sep 2020	PGMS		Notarially executed on 30/09/2015 under Protocol No. 801/20135 Filed for registration in the Mineral Titles Office - Awaiting Registration

Holder	DMR PR Reference	Description of Property	Hectares	Period of Prospecting Right	Minerals	Environmental Management Programme	Status
PTM	10809 PR Underlays 10667 PR Original JV	The Farms: Groenepunt 354 LR Rosamond 357 LR Millstream 358 LR	6254.80 HA		Iron & Vanadium		Written acceptance by DMR on 28 05/2013 All additional documentation has been filed with DMR awaiting advices for Grant of PR
PTM	10810 PR Extension	The Farms: Millbank 325 LR Udney 321 LR	4189.86 HA	Prospecting Right granted for 5 years with effect from 23 Oct 2013 to 22 Oct 2018	PGMS , Chrome, Copper, Gold, Nickel, Vanadium and Iron	Environmental Management Plan authorised 23/10/2013	Notarially executed on 23/10/2013 under Protocol No. 1130/2013 Registered in Mineral & Petroleum Titles Office o 3/12/2013 under MPTO No 163/2013
PTM	11286 PR Extension	The Farms: R/E Buffelshoek 261 LR Portion 1 Buffelshoek 261 LR The Bul-Bul 5 LS Portion1 & Extent Inveraan 262 LR Beauley 260 LR Dantzig 3 LS In-Der-Mark 7 LS The Glade 2 LS The Grange 257 LR R/E Innes 6 LS & Portion 1 Nairn 74 LS	19912.44 HA	Prospecting Right granted for 5 years with effect from 6 May 2016 to 5 May 2021	PGMS, Zinc, Gold, Nickel, Cobalt, Lead, Chrome, Molybdenum, Silver, Rare Earths, Copper, Vanadium & Iron		Written acceptance by DMR on 05/ 06/2013 DMR advised on 06/05/ 2016 that the application for the prospecting right have been granted. All additional documentation required for notarial execution has been prepared for filing with Regional Manager.

Holder	DMR PR Reference	Description of Property	Hectares	Period of Prospecting Right	Minerals	Environmental Management Programme	Status
PTM	10805 PR (Section 102) Extension	The Farms: Too Late 359 LR Bonne Esperance 356LR	4475.13 HA	Section 102 application, which includes the two farms. Too Late 359 LR & Bonne Esperance 356 LR it will have the same benefits as 10805 PR when granted	PGMS , Chrome, Copper, Gold, Nickel, Vanadium and Iron	Addendum to approved Environmental Management Plan was approved on 30 Mar 2016.	Written acceptance by DMR and accepted on 09/12/2013 Awaiting advices for Grant of Amendment to PR by DMR
PTM	12526	The Farms: Normandy 312 LR La Rochelle 310 LR Les Fontaines 271 LR Springfields 268 LR Langlaagte 279 LR Berg-en-Dal 276 LR	12296.65 HA	Applied for prospecting period of 5 years, which is the maximum in terms of Section 16 MPRDA.	Gold, Chrome, Nickel, Copper, Cobalt, Rare Earths, Iron, Molybdenum, Zinc, Silver, Lead & Vanadium	Environmental Authorisation filed with application.	Was applied for on the 17 Mar 2015. Awaiting Acceptance from DMR.

4.5 Holdings Structure

Until 2015, the Waterberg Project has been managed and explored under the direction of two separate technical committees for the Joint Venture (JV) and the Extension Projects respectively to cater for the needs, requirements and objectives of the ownership groups. A second agreement described in Section 4.5.3 has resulted in the consolidation of all holdings and the combined exploration and management of both areas. This second agreement gives the Company an effective 58.62% interest in the Waterberg Project, (45% directly and 13.62% indirectly through its 49.9% holding of Mnombo who holds an 26% interest) while JOGMEC has 28.35% and Mnombo's non-Platinum Group Metals shareholders with an effective 13.03% interest. All current expenditures are funded under commitment from JOGMEC.

4.5.1 History of the Waterberg JV Project

PTM applied for the original 137km ^{2} prospecting right for the Waterberg Joint Venture (JV) Project area in 2009 and in September 2009 the DMR granted PTM RSA a prospecting right until 1 September 2012 for the requested area. Application for the renewal of this prospecting right for a further three years has been made. Under the MPRDA, the prospecting right remains valid pending the grant of the renewal.

PTM initially held a 74% share in the Waterberg JV Project with Mnombo Wethu (Pty) Ltd (Mnombo), a BEE (Black Economic Empowerment) partner, holding the remaining 26% share (Figure 4-3).

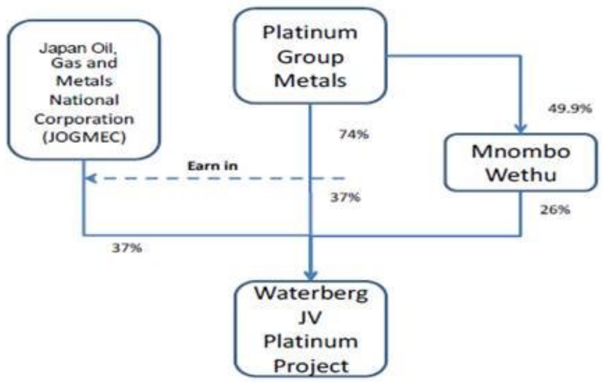


Figure 4-3: Schematic Diagram of the Initial Holdings of the Waterberg JV Project

In October 2009, PTM RSA, JOGMEC and Mnombo entered into a joint venture agreement (the "JOGMEC Agreement") whereby the Japan Oil, Gas and Metals National Corporation (JOGMEC) would earn up to a 37% participating interest in the Waterberg JV Project for an optional work commitment of US$3.2 million over four years, while at the same time Mnombo would earn a 26% participating interest in exchange for matching JOGMEC's expenditures on a 26/74 basis (US$1.12 million).

On November 7, 2011, PTM entered into an agreement with Mnombo whereby PTM acquired 49.9% of the issued and outstanding shares of Mnombo in exchange for cash payments totaling R1.2 million and an agreement that PTM would pay for Mnombo's 26% share of costs on the Waterberg Joint Venture until the completion of a feasibility study.

In April 2012, JOGMEC completed its US$3.2 million earn in requirement to earn a 37% interest in the Waterberg JV Project. Following JOGMEC's earn-in PTM funded Mnombo's 26% share of costs for US$1.12 million and the earn-in phase of the joint venture ended in May 2012. Pursuant to the JOGMEC Agreement, and prior to the closing of the 2nd Amendment, the Company, 37% by JOGMEC and 26% by Mnombo, holds interests in the Waterberg JV Project 37%. Because of the Company's 49.9% ownership interest in Mnombo, the Company currently has an effective interest in the Waterberg JV Project of approximately 50%. This ownership percentage will change if the 2nd Amendment receives Section 11 approval.

To the Company's knowledge, Mnombo remains over 50% held for the benefit of historically disadvantaged persons or historically disadvantaged South Africans ("HDSAs"), as defined respectively by the MPRDA and the Amendment of the Broad-Based Socio-Economic Empowerment Charter for the South African Mining and Minerals Industry, 2010 ("Mining Charter") and is a qualified BEE corporation under the Broad-Based Black Economic Empowerment Act, 2003 (the "BEE Act").

During 2012, PTM made application to the DMR to acquire three additional prospecting rights adjacent to the west (one property of 3,938 ha), north (one property of 6,272 ha) and east (one property of 1,608 ha) of the existing Waterberg JV Project. Upon granting by the DMR, these three new prospecting rights covering a total of 118km^2 became part of the existing joint venture with JOGMEC and Mnombo, bringing the total area in the joint venture to 255km^2.

4.5.2 History of the Waterberg Extension Project

The former Waterberg Extension Project includes contiguous granted and applied-for prospecting rights with a combined area of approximately 864km^2 adjacent and to the north of the Waterberg JV Project. Two of the prospecting rights were executed on 2 October 2013 and each is valid for a period of five years, expiring on 1 October 2018.

The third prospecting right was executed on 23 October 2013 and is valid for a period of five years, expiring on 22 October 2018. The Company has made an application under Section 102 of the MPRDA to the DMR to increase the size of one of the granted prospecting rights by 44km^2.

4.5.3 Waterberg Project Consolidation

On May 26, 2015, the Company announced a second amendment (the "2nd Amendment") to the JOGMEC Agreement. Under the terms of the 2nd Amendment, the Waterberg JV and Waterberg Extension projects (collectively the "Waterberg Project") were consolidated and contributed into a newly created operating company named Waterberg JV Resources (Pty) Ltd. ("Waterberg JV Co."). The Company holds 45.65% of Waterberg JV Co. while JOGMEC owns 28.35% and Mnombo holds 26%.

Through its 49.9% share of Mnombo, the Company holds an effective 58.62% of Waterberg JV Co. Based on the June 2014 Waterberg resource estimate the number of ounces owned by each entity did not change with the revised ownership percentages. Under the 2nd Amendment JOGMEC has committed to fund US$20 million in expenditures over a three year period ending March 31, 2018. Approximately US$8 million remains to be completed and will fund 100% of the costs for the balance of 2016 and into 2017.

Project expenditures in excess of US$6 million in either of years 2 or 3 are to be funded by the JV partners' pro-rata to their interests in Waterberg JV Co. Closing of this transaction is subject to MPRDA Section 11 approval by the DMR to transfer title of the prospecting rights. If Section 11 transfer approval is not obtained the parties will default to the pre-amendment JV arrangement, with any advances received from JOGMEC to be used to offset its spending commitments on the Waterberg JV property.

PTM is the operator of the Waterberg Project, with joint venture partners being JOGMEC and Mnombo. The 2nd Amendment Agreement allows all of the Waterberg area to be considered from a resource and engineering perspective allows for optimization of the 13km+ of target strike length and allows for exploration and engineering to be aggressively advanced.

4.6 Royalties and Encumbrances

4.6.1 Royalties

4.6.1.1 The Mineral and Petroleum Resources Royalty Act, 2008 "The Royalty Act"

The Royalty Act came into effect on 1 March 2010. The Royalty Act gives effect to the MPRDA, which requires that compensation be given to the State (as custodian) of the country's Mineral and Petroleum Resources to the country's "permanent loss of non-renewable resource". The Act distinguishes between refined and unrefined Mineral Resources, where refined minerals have been refined beyond a condition specified by the Act, and unrefined minerals have undergone limited beneficiation as specified by the Act.

The royalty is determined by multiplying the Gross Sales Value of the extractor in respect of that Mineral Resource in a specified year by the percentage determined in accordance with the royalty formula. Both operating and capital expenditure incurred is deductible for the determination of Earnings before Interest and Tax ("EBIT").

The royalty is determined by multiplying the gross sales value of the extractor in respect of that Mineral Resource in a specified year by the percentage determined in accordance with the royalty formula. Both operating and capital expenditure incurred is deductible for the determination of EBIT.

For Refined Mineral Resources is a follows:-

$$\text{Royalty Rate} = 0.5 + \frac{\text{EBIT}}{\text{Gross Sales (refined)} \times 12.5} \quad \text{X } 100$$

The maximum percentage for refined Mineral Resources is 5%.

For Unrefined Mineral Resources:-

$$\text{Royalty Rate} = 0.5 + \frac{\text{EBIT}}{\text{Gross Sales (refined)} \times 9} \quad \text{X } 100$$

The maximum percentage for unrefined Mineral Resources is 7%.

4.6.2 Encumbrances

There are no liens, pledges, mortgage bonds or any encumbrances of whatsoever nature that has been registered against the Waterberg prospecting right.

4.7 Property Agreements

There are no surface property purchase agreements in place. PTM are currently in negotiations with respect to the acquisition of surface property. PTM has surface access agreements as required for exploration.

4.8 Permits

Permits to support mine development activities are more fully set out in Section 20.

4.9 Environmental Liabilities

All environmental requirements on the properties are subject to the terms of a current Environmental Management Plan (EMP) approved by the Department of Minerals Resources (DMR) prior to commencement of work on the properties. All rehabilitation of drill hole sites and access roads required in terms of this EMP has been completed or are ongoing. In addition, the required deposits into the approved environmental rehabilitation trust in respect of related potential liabilities are up to date. There are no other environmental liabilities on the properties.

All the necessary permissions and permits in terms of the environmental liabilities have been obtained. There are no known encumbrances of an environmental nature that may restrict the exploration of the properties.

4.10 Social License

The Social License, which deals with community empowerment, will be signed on granting of the Mining Right. In terms of the MPRDA Social License forms the integral the granting of a Mining Right application hence without it DMR will not grant the Mining Right License. The Social License in this case is represented by the Social and Labor Plan (SLP).

This plan deals with issues of Human Resource Development and Local Economic Development. This intervention is enforced by the government in terms of the MPRDA.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography

The project area to the west and east is relatively flat but the area in the central part of the project area is more mountainous with some steep near vertical cliffs and an elevation difference of 160 - 200m. The lowest point in the project area is at 880m alms and the highest point at 1,365m alms. The drilling has been undertaken on the eastern flat area with an elevation of approximately 1,000m alms. The area is farmed by the local people who grow crops on a limited scale and farm livestock. The vegetation is typically Bushveld vegetation. The Seepabana River cuts across the southwestern side of the Waterberg Joint Venture Project area from east to west joining the Molagakwena River, which flows north into the Glen Alpine dam. The remainder of the area has non-perennial rivers.

5.2 Elevation

High points on the Makgabeng boast elevations of some 1,330m while the project area ranges in elevation of between 970 - 1050m above mean sea level.

5.3 Vegetation

From a regional perspective, the center of the prospecting area is covered with a vulnerable vegetation type, namely Soutpansberg Mountain Bushveld; while the remaining area is covered mainly with least threatened Roodeberg Bushveld. The vulnerable Soutpansberg Mountain Bushveld extends into the northwestern corner of the core study area. Most of the eastern and southern properties of the core study area are covered with least threatened Roodeberg Bushveld.

A small portion of the vulnerable Makhado Sweet Bushveld extents into the eastern boundary of the core study area. The three vegetation types mentioned belongs to the Savannah Biome of South Africa and is summarized in Table 5-1 below.

Table 5-1: Summary of Vegetation

Overview of the Three Vegetation Types present within the Core Study Area

Vegetation Unit	Roodeberg Bushveld SVcb 18	Soutpansberg Mountain Bushveld SVcb 21	Makhado Sweet Bushveld SVcb 20
Vegetation & Landscape Features	Plains and slightly undulating plains, including some low hills, with short closed woodland to tall open woodland and poorly developed grass layer. Kirkia acuminate trees not limited to hills	Low to high mountains, highest in the west, splitting into increasing number of lower mountain ridges towards the east. Dense tree layer and poorly developed grassy layer. The topography of the east-west orientated ridges of the mountain changes drastically over short distances, resulting in orographic rain on the southern ridges, and a	Slightly too moderately undulating plains sloping generally down to the north, with some hills in the southwest. Short and shrubby bushveld with a poorly developed grass layer

		rain shadow effect on the northern ridges. Because of this topographic diversity, the Soutpansberg mountain Bushveld comprises a complex mosaic of sharply contrasting kinds of vegetation within limited areas	
Biogeographically important Taxa	No	Yes - 4 Species	No
Endemic Taxa	No	Yes - 12 Species	Yes - 1 Specie
Conservation Status	Least threatened. Target 19%. Almost 6% statutorily conserved, mainly in the Wonderkop and Blouberg (Malebocho) Nature Reserves. An additional 3% conserved in other Reserves, mainly in areas adjacent to the Wonderkop Nature Reserves. About 18% transformed	Vulnerable. Target 24%. Just over 2% statutorily conserved in the Blouberg, Happy Rest and Nwanedi Nature Reserves. A smaller area is conserved in other Reserves.	Vulnerable. Target 19%. About 1% statutorily conserved, mainly in the Bellevue Nature Reserve
Transformation level	About 18% transformed, mainly by cultivation, with very little urban and built-up areas. Erosion is low to high	21% transformed, with about 14% cultivated and 6% plantations. High rural human population densities in some of the lower lying parts of the eastern section of the unit. Erosion is very low to moderate	Some 27% transformed, mainly by cultivation, with some urban and built up areas. The southwestern half of the unit has densely population rural communities. Erosion is low to high
Utilisation	The area is mostly used for game ranching	None specified	None specified

5.4 The Means of Access to the Property

Paved roads provide access to within 32km of the project, with unpaved typically rural roads providing direct access to site. Currently seasonal conditions may limit the access to four-wheel drive vehicles during the rainy season.

5.5 Local Resources

Minimal service related infrastructure exists, as the area is largely undeveloped rural farmland. Roads are "basic", electricity is three-phase 22kVA rural farmland supply and water is borehole supplied with minimal reticulation. Local population is mostly pastoral based or weekly migrant worker based. Local industries are limited to small-scale mechanical workshops and general dealers. A local governmental hospital falls within the reach of the project. Much of the more serious medical cases are dealt with on a referral basis to the main hub medical facility in Polokwane.

5.6 Regional Infrastructure

Maps at a scale of 1:50,000 are available. The main sheets that cover the project area are 2328BD and to the north 2328BB. Roads are indicated as secondary and tertiary from the N11 (~30km straight line distance) and approximately 50km from Dendron via Bochum to site.

No rail facilities service the area, the nearest serviceable point being Mokopane some 110km distant along the N11 and the secondary and tertiary access roads.

No reticulated water system is noted to exist within 25km of site.

Surface rights, access and construction of regional infrastructure delays may delay the project. Negotiation of surface agreements is provided for in the MRPDA and regional infrastructure construction has been provided for in the project plan.

5.6.1 Power

Local 22kVA low tension farm reticulation is sporadically installed servicing many of the local villages and general dealers.

5.6.2 Water

Borehole based water supply is relied upon for local village, dwelling and farmland cattle trough supply. Limited irrigable land farming is conducted; mostly domestic subsistence dryland cultivation is relied upon for local community needs. Regionally there are significant wells used for agriculture at 4ML or more.

5.6.3 Roads

Secondary and tertiary unpaved roads service the local villages, schools and communities. The paved N11 from Mokopane to Grobblaarsdrift border post passes approximately 30km straight line distance from site but the road access from the N11 is about 32km on unpaved surfaces. The R521 from Polokwane to Alldays passes the farming community of Dendron from where a paved road to Bochum (now known as Senwabarwana), secondary and tertiary roads service to site and local schools and villages.

5.7 Physiography

Cliffs of Waterberg sandstones rise abruptly forming the polygonal-parallelogram shaped Makgabeng plateau from the flat to gently sloping surrounding foothills. These surrounds underlain by Waterberg sandstones and shales of the Makgabeng formation. Sheetlike sub-horizontal sills of doleritic to diabase composition cut and protrude the sandstones leaving slight elevated hillocks. Subvertical doleritic dykes cut the Makgabeng plateau in an orthogonal pattern creating deep gullies several tens of metres wide. Land surface is generally covered by thick sandy soils with sparse tufty grasslands and acacia woodland

5.7.1 Fauna

Based on the known geographic distributions of the sensitive faunal species of the Limpopo Province, the nine Q-grids relevant to the prospecting area were ranked in terms of relative faunal sensitivity. The core study area (the four farms Early Dawn 361, Goedetrouw 366, Ketting 368 and Millstream 358) falls within the grid 2328BD. This specific Q-grid was only ranked 7/9 in terms of relative local faunal sensitivity with only Q-grids 2328BA and 2328DB having lower faunal sensitivities. In other words, the core study area, which is part of the prospecting area, has relatively low faunal sensitivity.

The distribution and extent of national biodiversity areas within the core study show the high sensitivity for most of Millstream 358, and parts of Ketting 368 and Early Dawn 361. These sensitivities are further emphasized by the distribution and extent of the Limpopo Province Conservation Priority Areas within the core study area. The total ecological sensitivity model compiled for the prospecting area revealed a similar sensitivity pattern with most of Millstream 358, the northern part of Ketting 368 and southeastern parts of Early Dawn 361 considered to have very high relative ecological sensitivities.

During a site visit, the presence of five red data birds was confirmed. Red-billed Oxpecker, Cape vulture, Lappet-faced vulture, Pallid harrier and Martial eagle were found to occur in the core study area. It is likely that a detailed faunal investigation will reveal the presence of more sensitive faunal species of the Limpopo Province. Therefore, it is important to keep habitat transformation and degradation associated with the proposed mining activities within the core study area to faunal habitats of low sensitivity.

Based on the national, provincial and regional sensitivity analyses results, it is considered that Millstream 358 has the highest faunal sensitivity of the four original farms within the core study area.

5.7.2 Birds

Woodland birdlife typified by Wood Doves, Kwevoels grey Go-away-bird Corythaixoides commonly known as the “grey go-away-bird” (Corythaixoides concolor dominate the Acacia scrubland.)

Detailed studies will be conducted during the EIA phase.

5.7.3 Herpetofauna

Venomous herpetofauna are not recorded, but numerous brightly colored rock and tree trunk climbing lizards are recorded. The Transvaal Grass Lizard, also known as the Transvaal Snake Lizard is a species of lizard in the genus Chamaesaura. It is found in southern African grasslands and on slopes. The Transvaal Grass Lizard is ovoviparous.   The scientific name refers to its copper color (source Wikipedia).

Several species of snakes are recorded in the area amongst which are the puffadder, grass snake, python (monty-pythanigus pantagonus), green mamba, and the highly dangerous and respected spitting cobra and black mamba.

Detailed studies will be conducted during the EIA phase.

5.7.4 Mammals

Two species of monkeys are recorded in the foothills and on the highlands of the Makgabeng. Chacma baboons (Papio ursinus) are often heard barking with alarm when approached or disturbed while foraging.

Smaller vervet monkeys (Cercopithecus pygeryhrus) live in the foothills surrounding villages and are known to raid for food as is their want.

5.7.5 Local Rock Art

While local legend records the presence of Bushmen rock art in the region in general, none has been located within or adjacent to the project area despite several scouting exercises. Local "experts' have also been unsuccessful in pointing out local rock art within or adjacent to the current project area. These sites will be protected if properly identified. No such sites have been located in the PFS project development area.

5.7.6 Sites of Sensitivity in the Area

The pastoral village farming based community basis of habitation in the area has naturally allowed local gravesites to be developed in proximity to the homesteads and village groupings of dwellings. These need to be located, mapped and demarcated for site preservation. Initial environmental assessments have located and mapped these sites in the area of the exploration work.

5.8 Climate and the Length of the Operating Season

Temperate to savannah summer rainfall conditions prevail with summer highs reaching the low 40ºC values, but typically, mid 30ºC. Winter temperatures drop to low teens and may rarely reach single ºC figures.

The majority of the 350-400mm of average annual rainfall occurs in the period November to March. Climatic conditions have virtually no impact on potential mining operations in the project area. The dry season persists from April to mid to late September typically.

6. History

The Waterberg Project is a part of a group of exploration projects that came from a regional target initiative of the Company over the past six years. PTM targeted this area based on its own detailed geophysical, geochemical and geological work along trend, off the north end of the mapped Northern Limb of the Bushveld Complex.

The permits for the properties to the northwest were applied for based on the initial findings on the Project combined with an analysis of publicly available regional government geophysical data that showed an arching NNE tend to the signature of the interpreted edge of the Bushveld Complex.

6.1 Prior Ownership

PTM developed the exploration concept for the Waterberg Project and filed for a Prospecting Right application, which was granted in 2009. In October 2009, PTM entered an agreement with JOGMEC and Mnombo whereby JOGMEC would earn up to a 37% interest in the project for an optional work commitment of US$3.2 million over 4 years on the Waterberg JV Project only. A condition was required to match JOGMEC's expenditures on a 26/74 basis. PTM agreed to loan Mnombo their first $87,838 in project funding. JOGMEC has completed their earn-in expenditure in April 2012.

On 7 November 2011, the Company entered into an agreement with Mnombo whereby the Company would acquire 49.9% of the issued and outstanding shares of Mnombo in exchange for cash payments totaling R 1.2 million and paying for Mnombo's 26% share of project costs to feasibility. When combined with the Company's 37% direct interest in the Waterberg Project (after JOGMEC earn-in), the 12.974% indirect interest to be acquired through Mnombo brings the Company's effective project interest to 49.974%.

The Waterberg Extension Project Licenses were applied for separately and later with PTM having a direct 74% interest and Mnombo retaining the remaining 26%. Subsequently, an amendment to the Waterberg JV structure consolidated the properties under one structure, subject to the Ministers consent as outlined in Section 4.4.

6.2 Exploration

Previous work that was conducted over the property was the regional mapping by the Council for Geoscience as presented on the 1:250,000 scale – Map No 2328 – Pietersburg. This geological map of the area, along with the regional aeromagnetic and gravity surveys is the basis for the initial exploration concept.

There is no publically available detailed exploration history available for the area. As a result, of the cover on the Bushveld Complex, there is no record of specific exploration for platinum group metals and the extensive exploration for platinum group metals on the Platreef targets to the south did not extend this far north. There are undocumented reports of a drill hole through the Waterberg Group into the Bushveld Complex on a farm immediately north of the Waterberg JV area and immediately west of the Waterberg Extension area.

The original exploration models for the property involved a potential for paleo placer at the base of the Waterberg Group sediments or an embayment to the west. Both of these models were discarded with the current discovery and drilling data showing a strike to the north northeast.

6.3 Historical Mineral Resources Estimates

6.3.1 September 2012

The initial mineral resource was declared for the T- and F-Zone mineralization and is confined to only the property Ketting 368LR of the Waterberg JV Project. Data from the drilling completed by PTM to September 2012 was used to undertake a mineral resource estimate from over 58 intersections representing 27 drill holes. The data and the geological understanding and interpretation were considered of sufficient quality for the declaration of an inferred mineral resource classification.

This estimate was presented in a NI 43-101 in September 2012 by Mr. KG Lomberg, entitled "Exploration Results and Mineral Resource Estimate for the Waterberg Platinum Project, South Africa. (Latitude 23° 21' 53"S, Longitude 28° 48' 23"E)". Table 6-1 shows the Mineral Resource Statement for September 2012.

The drill hole intersections were composited for Pt, Pd, Au, Cu and Ni. A common seam block model was developed into which the estimate was undertaken. An inverse distance weighted (power 2) was undertaken using the 3D software package CAE Mining Studio™. Geological loss of 25% was estimated based on the knowledge of the deposit. The geological losses are made up of areas of where the layers are absent due to faults, dykes, potholes and mafic/ultramafic pegmatites.

Table 6-1: Summary of Waterberg Project, Mineral Resource Estimate 1 September 2012, SAMREC Code, Inferred Resource at 2 g/t (2PGE+Au) cut-off 100% Project Basis

	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	2PGE +AU (g/t)	Pt:Pd:Au	2PGE +AU (koz)	Cu (%)	Ni (%)
T1	2.85	10.49	0.77	1.27	0.51	2.55	30:50:20	863	0.17	0.10
T2	3.46	16.25	1.10	1.82	0.92	3.84	29:47:24	2,001	0.18	0.09
T Total	3.19	26.74				3.33	29:48:23	2,864		
FH	4.63	18.10	0.80	1.48	0.09	2.37	34:62:4	1,379	0.03	0.12
FP	5.91	23.20	1.01	2.00	0.13	3.14	32:64:4	2,345	0.04	0.11
F Total	5.27	41.30				2.80	31:57:12	3,724		
Total	4.19	68.04	0.94	1.71	0.37	3.01		6,588		
Content (koz)			2,049	3,733	806	—				

6.3.2 February 2013

An updated mineral resource was declared for the T- and F-Zone mineralization and confined to only the properties Ketting 368LR and Goedetrouw 366LR of the Waterberg JV Project.

Data from the drilling completed by PTM to February 2013 was used to undertake a mineral resource estimate from 207 intersections representing 40 drill holes. The data and the geological understanding and interpretation were considered of sufficient quality for the declaration of an inferred mineral resource classification. Table 6-2 shows the Mineral Resource Statement for February 2013. This estimate was presented in a NI 43-101 in February 2013 by Mr. KG Lomberg, entitled "Revised and Updated Mineral Resource Estimate for the Waterberg Platinum, South Africa (Latitude 23° 21' 53"S, Longitude 28° 48' 23"E)".

The drill hole intersections were composited for Pt, Pd, Au, Cu and Ni. A common seam block model was developed into which the estimate was undertaken. An inverse distance weighted (power 2) was undertaken using the 3D software package CAE Mining Studio™.

Geological loss of 25% was estimated based on the knowledge of the deposit. The geological losses are made up of areas of where the layers are absent due to faults, dykes, potholes and mafic/ultramafic pegmatites.

Table 6-2: Summary of Waterberg Project, Mineral Resource Estimate 1 February 2013, SAMREC Code, Inferred Resource at 2 g/t (2PGE+Au) cut-off 100% Project Basis

	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	2PGE +AU (g/t)	Pt:Pd: Au	2PGE +AU (koz)	Cu (%)	Ni (%)
T1	2.58	4.33	0.91	1.37	0.52	2.80	32:49:19	390	0.21	0.11
T2	4.08	25.46	1.07	1.87	0.78	3.72	29:50:21	3,045	0.17	0.09
T Total	3.76	29.78	1.05	1.79	0.75	3.59	29:50:21	3,435	0.18	0.09
FH	4.02	7.19	1.09	2.37	0.20	3.66	30:65:6	847	0.10	0.22
FP	5.46	55.95	1.01	2.10	0.14	3.25	31:65:4	5,838	0.06	0.16
F Total	5.24	63.15	1.02	2.13	0.15	3.29	31:65:4	6,685	0.06	0.17
Total	4.63	92.93	1.03	2.02	0.34	3.39	30:60:10	10,120		
Content (koz)			3,071	6,040	1,009	—				

6.3.3 September 2013

6.3.3.1 Waterberg JV Project

A mineral resource was declared for the T- and F-Zone mineralization and confined to only the properties Ketting 368LR and Goedetrouw 366LR of the Waterberg JV Project. Data from the drilling completed by PTM to 1 August 2013 was used to undertake a mineral resource estimate from 337 intersections representing 112 drill holes. Table 6-3 shows the Mineral Resource Statement for September 2013. The data and the geological understanding and interpretation were considered of sufficient quality for the declaration of an inferred mineral resource classification. This estimate was presented in a NI 43-101 in September 2013 by Mr. KG Lomberg and Mr. AB Goldschmidt; entitled “Revised and Updated Mineral Resource Estimate for the Waterberg Platinum Project, South Africa”.

The drill hole intersections were composited for Pt, Pd, Au, Cu and Ni. A common seam block model was developed into which the estimate was undertaken. An inverse distance weighted (power 2) was undertaken using the 3D software package CAE Mining Studio™.

Geological loss of 12.5% was estimated based on the knowledge of the deposit. The geological losses are made up of areas of where the layers are absent due to faults, dykes, potholes and mafic/ultramafic pegmatites.

Table 6-3: Summary of Waterberg Project, Mineral Resource Estimate 2 September 2013, SAMREC Code, Inferred Resource at 2 g/t (2PGE+Au) cut-off 100% Project Basis

	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	2PGE +AU (g/t)	Pt:Pd: Au	2PGE +AU (koz)	Cu (%)	Ni (%)
T1 (Cut- off=2g/t)	2.30	8.5	1.04	1.55	0.47	3.06	34:51:15	842	0.17	0.10
T2	3.77	39.2	1.16	2.04	0.84	4.04	29:51:21	5.107	0.18	0.10
T Total	3.38	47.7	1.14	1.95	0.77	3.86	30:51:20	5,948	0.18	0.10
F (Cut- off=2g/t)		119.0	0.91	1.98	0.13	3.02	30:65:4	11,575	0.07	0.17
Total		166.7	0.98	1.97	0.32	3.26	30:60:10	17,523	0.10	0.15
Content (koz)			5,252	10,558	1,715	—				

6.3.3.2 Waterberg Extension Project

A mineral resource estimate had not been declared for the Waterberg Extension Project as there was insufficient drilling completed to support a resource estimate in September 2013.

6.3.4 June 2014

In June 2014, Waterberg JV Project was more advanced in exploration status and included an Inferred Mineral Resource estimate. The majority of the Waterberg Extension Project was then still at an early exploration stage, however recent drilling on the property Early Dawn 361LR, just north of the Waterberg JV Project had sufficient surface drilling to confirm continuity of mineralization, hence the areas could be classified as Inferred Mineral Resource.

The data was used to define the characteristics of the various layers based on their geochemical signatures. Validation was undertaken on the core with the intention of finding diagnostic features to identify the layers directly from the core. This was successfully achieved for the T-Zone. Due to the pervasive alteration, it proved difficult in the F-Zone.

All the intersections were checked on the core to ensure that the layer designation was true to the core and consistent for all the deflections from a drill hole. Seven different layers (FP and H1-FH6) within the F-Zone were identified. It is the identification of these layers and the classification of historical exploration data to fit a new interpretation that is the primary difference between this and previous mineral resource estimates. These cuts formed the basis of the Mineral Resource Estimate. The cuts were also defined based on the geology, a marginal cut-off grade of 2g/t PGM and a minimum thickness of 2m. Basic statistics were undertaken on the data noting that the data was clustered due to the number of deflections for each drill hole.

Data from the drilling completed by PTM in the estimate consisted of intersections from 138 drill holes. Each drill hole was examined for completeness in respect of data (geology, sampling, and collar) and sample recovery prior to inclusion in the estimate.

Geological models (wireframes) of the seven F-Zone units were modelled by CAE Mining (South Africa) on behalf of PTM, using the Strat 3D module of CAE Mining Studio™.

The coded drill hole database supplied by PTM was composited for Pt, Pd, Au, Cu, Ni and density. For each unit a three-dimensional block model was modelled and an inverse distance weighted (power 2) estimate was undertaken. Two areas were defined where geological loss of 25% and 12.5% respectively were applied.

The resource estimate tabulation is set out in Table 6-4.

Table 6-4: Summary of Waterberg Project, Mineral Resource Estimate 12 June 2014, SAMREC Code, Inferred Resource at 2 g/t (2PGE+Au) cut-off 100% Project Basis

	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	2PGE+AU (koz)	Pt:Pd:Rh: Au	2PGE+Au (koz)	Cu (%)	Ni (%)	Cu (Mlbs)	Ni (Mlbs)
T1 (Cut-off=2g/t)	2.44	10.49	1.02	1.52		0.47	3.01	34:50:0:15	1,015	0.17	0.10	40	23
T2	3.87	43.57	1.14	1.99		0.82	3.95	29:50:0:21	5,540	0.17	0.09	167	90
T Total	3.60	54.06	1.12	1.90		0.75	3.77	30:50:0:20	6,555	0.17	0.10	207	114
F		232.82	0.90	1.93	0.05	0.14	3.01	30:64:2:5	22,529	0.08	0.19	409	994
Total		286.88	0.94	1.92	0.04	0.25	3.15	30:61:1:8	29,084	0.10	0.18	617	1,107
Content (koz)			8,652	17,741	341	2,350			kt	280	502		

6.3.4.1 Waterberg Project

On 22 July 2015, the Company declared an updated mineral resource for the Waterberg Project.

6.3.5 July 2015

Data used in this estimate comprised 220 original drill holes with 270 deflections. The total dataset comprised 231 original drill holes and 374 deflections. Of these 89 intersections occurred in the T-Zone ranging from approximately 140m to 1380m in depth below surface. Three hundred and sixty five intersections in the F-Zone were used ranging from approximately 200m to 1250m in depth. The resource estimate tabulation is set out in Table 6-5.

Table 6-5: Summary of Waterberg Project, Mineral Resource Estimate effective 20 July 2015 on 100% Project Basis

Resource Category	Cut-off 3E g/t	Tonnage Mt	Grade Pt g/t	Grade Pd g/t	Grade Au g/t	Grade 3E g/t	Grade Cu %	Grade Ni %	Metal 3E kg	Metal 3E Moz
T Zone 2.5 g/t Cut-off										
Indicated	2.5	16.53	1.28	2.12	0.85	4.25	0.16	0.09	70253	2.26
Inferred	2.5	33.56	1.25	2.09	0.83	4.17	0.13	0.08	139945	4.50
F Zone 2.5 g/t Cut-off										
Indicated	2.5	104.47	0.93	2.00	0.15	3.08	0.06	0.16	321768	10.35
Inferred	2.5	212.75	0.93	2.01	0.15	3.09	0.07	0.17	657398	21.14
T Zone 2.0 g/t Cut-off										
Indicated	2.0	18.97	1.20	2.00	0.79	3.99	0.16	0.09	75 690	2.43
Inferred	2.0	34.99	1.23	2.04	0.82	4.09	0.13	0.08	143 109	4.60
F Zone 2.0 g/t Cut-off										
Indicated	2.0	192.94	0.81	1.76	0.13	2.70	0.06	0.16	520 938	16.75
Inferred	2.0	440.13	0.79	1.72	0.13	2.64	0.07	0.17	1 161 943	37.36

6.3.6 April 2016

Data used in this estimate comprised 301 original drill holes with 479 deflections. Intersections occurred in the T-Zone ranging from approximately 140m to 1380m in depth below surface, and in the F-Zone ranging from approximately 200m to 1250m in depth. The resource estimate tabulation is set out in Table 6-6.

Table 6-6: Summary of Waterberg Project, Mineral Resource Estimate effective 19 April 2016 on 100% Project Basis

T-Zone 19/04/2016

Cut-off	Tonnage	Grade					Metal	
2PGE+Au		Pt	Pd	Au	Rh	2PGE+ Au	2PGE +Au	
g/t	Mt	g/t	g/t	g/t	g/t	g/t	Kg	Moz
Indicated								
2	36.308	1.1	1.8	0.7	—	3.61	131,162	4.217
2.5	30.234	1.2	1.9	0.8	—	3.88	117,363	3.773
3	22.33	1.3	2.1	0.9	—	4.28	95,640	3.075
Inferred								
2	23.314	1.1	1.8	0.7	—	3.66	85,240	2.741
2.5	21.196	1.1	1.9	0.8	—	3.79	80,394	2.585
3	14.497	1.3	2.1	0.9	—	4.28	62082	1.996

F-Zone 19/04/2016

Cut-off	Tonnage	Grade					Metal	
4E		Pt	Pd	Au	Rh	2PGE +Au	2PGE +Au	
g/t	Mt	g/t	g/t	g/t	g/t	g/t	Kg	Moz
Indicated								
2	281.184	0.9	1.9	0.2	0	3.03	851,988	27.39
2.5	179.325	1.1	2.2	0.2	0	3.49	625,844	20.12
3	110.863	1.2	2.5	0.2	0	3.95	437,909	14.08
Inferred								
2	177.961	0.8	1.8	0.1	0	2.76	491,183	15.79
2.5	84.722	1	2.1	0.2	0	3.35	283,819	9.125
3	43.153	1.2	2.5	0.2	0	3.96	170,886	5.494

6.4 Production from the Property

There is no historic production from the Waterberg Project.

7. Geological Setting and Mineralisation

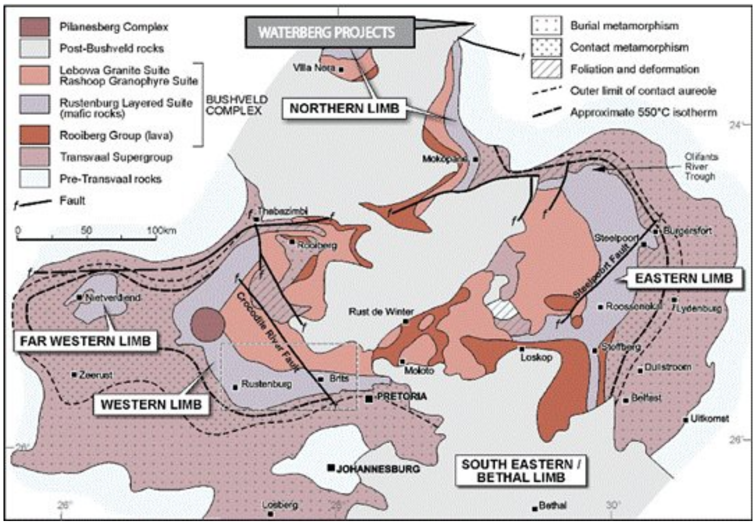
7.1 Regional and Local Setting

The Bushveld and Molopo Complexes in the Kaapvaal Craton are two of the most well-known mafic/ultramafic layered intrusions in the world. The Bushveld Complex was intruded about 2,060 million years ago into rocks of the Transvaal Supergroup, largely along an unconformity between the Magaliesberg quartzite of the Pretoria Group and the overlying Rooiberg felsites. The Bush Complex is estimated to exceed 66,000km ^{2} in extent, of which about 55% is covered by younger formations. The Bushveld Complex hosts several layers rich in Platinum Group Metals (PGM), chromium and vanadium, and constitutes the world's largest known resource of these metals.

The Waterberg Project is situated off the northern end of the previously known Northern Limb, where the mafic rocks have a different sequence to those of the Eastern and Western Limbs.

The Bushveld Complex in the Waterberg Project area has intruded across a pre-existing craton scale lithological and structural boundary between two geological zones. The known Northern Limb has a north – south orientation to the edge contact that makes an abrupt strike change to the northeast coincident with projection of the east-west trending Hout River Shear system, a major shear zone that marks the southern boundary of the South Marginal Zone (SMZ). The SMZ is a 2500Ma aged compressional terrain formed within the Kaapvaal Craton during the collision with the Zimbabwe Craton. It is comprised of granulite facies granitic gneiss, amphibolitic gneiss and minor quartzite. Within the SMZ, several major shears trend parallel to the Hout River Shear (van Reenen, 1992) and trend through the Waterberg Extension Project area. The footwall to the Bushveld Complex on Waterberg Project is interpreted to be comprised of facies of the SMZ.

The Platreef characterizes the geology of the Northern Limb of the Bushveld Complex. It was first described by van der Merwe (1976). The Platreef is typically a wide pyroxenite-hosted zone (up to 100's of metres), of elevated Cu and Ni mineralization with associated anomalous PGM concentrations. The sulphide mineralization is typically pyrrhotite, chalcopyrite and pentlandite. It was postulated that the interaction with the basement rocks, and in particular the dolomites, was instrumental in the formation of the mineralization (Vermaak and van der Merwe, 2000).



Source: After Cawthorn et al 2006

Figure 7-1: Geological Map of the Bushveld Complex showing the Location of the Waterberg and Waterberg Extension Projects

7.1.1 Bushveld Complex Stratigraphy

The mafic rocks of the Bushveld Complex are stratigraphically referred to as the Rustenburg Layered Suite and can be divided into five zones known as the Marginal, Lower, Critical, Main and Upper Zones from the base upwards (Figure 7-1 and Figure 7-2).

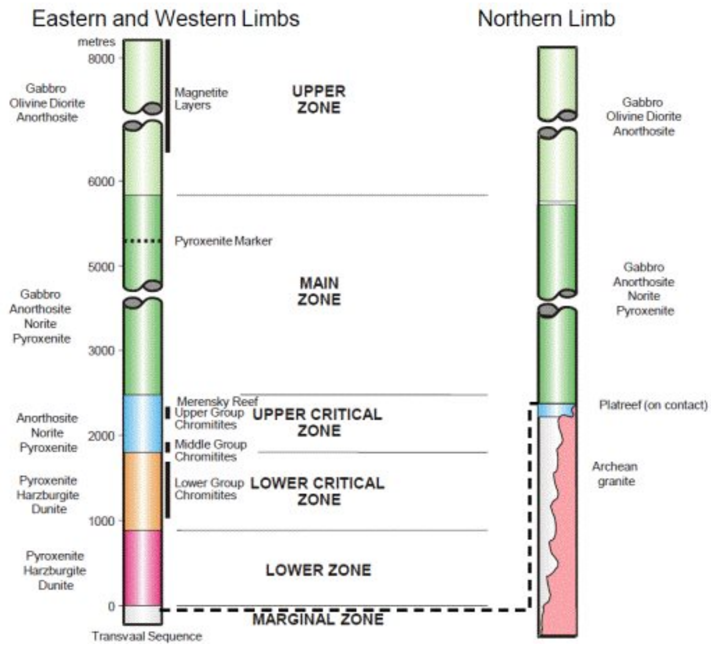


Figure 7-2: Waterberg Project Generalised Stratigraphic Columns of the Eastern and Western Limbs compared to the Stratigraphy of the Northern Limb of the Bushveld Complex

7.1.2 Mineralisation

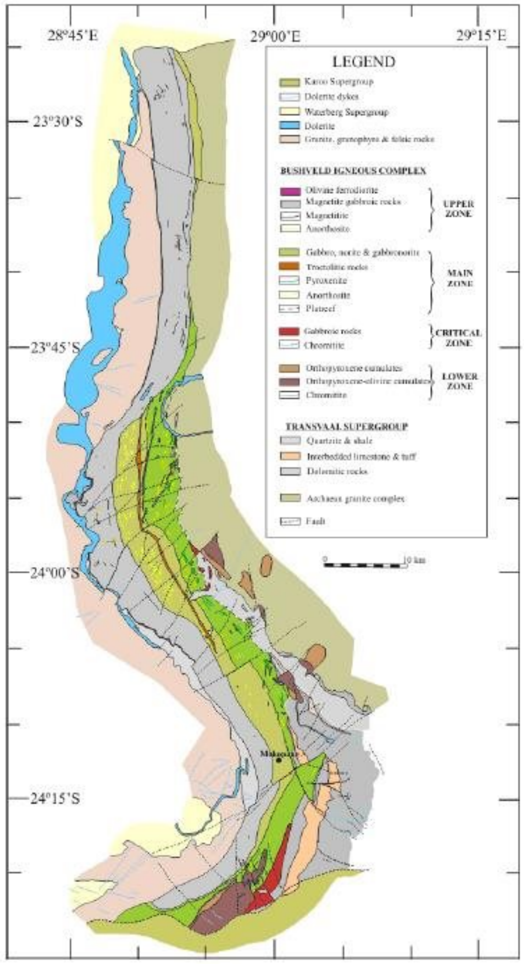
The Critical Zone hosts the majority of the PGE mineralization and is characterized by regular and often fine-scale rhythmic, or cyclic, layering of well-defined layers of cumulus chromite within pyroxenites, olivine-rich rocks and plagioclase-rich rocks (norites, anorthosites etc.). The pyroxenitic Platreef mineralization, north of Mokopane (formerly Potgietersrus), contains a wide zone of more disseminated style PGE mineralization, along with higher grades of nickel and copper, than occur in the rest of the Bushveld Complex.

7.1.3 The Northern Limb

The Northern Limb is a north-south striking sequence of igneous rocks of the Bushveld Complex with a length of 110km and a maximum width of 15km (Figure 7-3 and Figure 7-4). It is generally divided up into three different sectors namely the Southern, Central and Northern sectors that have characteristic footwall units: -

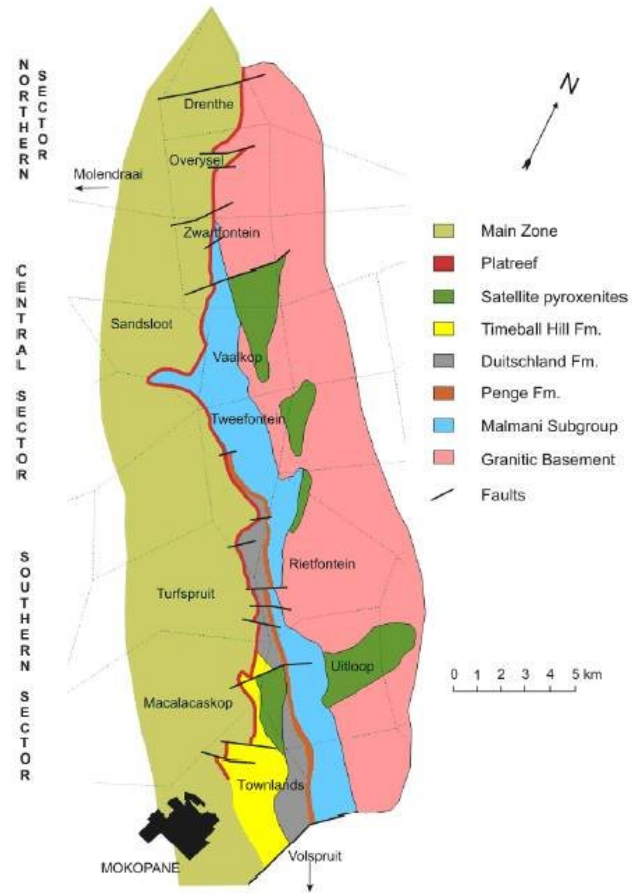
- The Southern Sector is characterized by a footwall of the Timeball Hill, Duitschland and Penge Formations of the Transvaal Supergroup;

- The Central Sector generally has a footwall of Malmani Subgroup; and
- The Northern Sector has a footwall consisting of Archaean granite.



Source Ashwal et al, 2005

Figure 7-3: General Geology of the Northern Limb of the Bushveld Complex



Source: Kinnaird et al, 2005

Figure 7-4: Geology of the Northern Limb of the Bushveld Complex showing the Various Footwall Lithologies

7.2 Waterberg Group/Bushveld Complex Age Relationship

The contact between the Waterberg Group and the weathered Bushveld Complex was observed in the drillcore to generally be sharp. The unusual contact zone between the two rock units was examined by Prof. McCarthy from the University of the Witwatersrand and is interpreted as a palaeosol (fossilized soil) developed on the Bushveld gabbros. The weathering palaeosol is interpreted to be a result of typical spheroidal weathering seen in modern weathering of Bushveld rocks. The product of the weathering is a very fine-grained turf layer (vertisol), typically logged as “shale” in the drill intersections.

The nature of the relationship between the Waterberg Group and the Bushveld Complex is confirmed as having no bearing on the presence of mineralization in the gabbros (T- or F-Zones) (McCarthy, 2012).

In addition, Prof McCarthy observed that the northern extremity of the Northern Limb of the Bushveld Complex contains a well-developed Platreef horizon, but also has mineralization developed in the Upper Zone. The T-Zone has a high Cu/Ni ratio and is Pd and Au dominated. Sulphides similar to this were described previously from the Upper Zone, but occur in very small quantities, suggesting that atypical conditions pertain in the project area (McCarthy, 2012)..

7.3 Project Geology

The Waterberg Project is located along the strike extension of the Northern Limb of the Bushveld Complex. The surface geology is depicted in Figure 7-5. The geology of the Bushveld intrusion consists predominantly of the Main Zone gabbros, gabbronorites, norites, pyroxenites and anorthositic rock types with more mafic rock material such as harzburgite and troctolites that partially grade into dunites towards the base of the package. In the southern part of the project area, Bushveld Upper Zone lithologies such as magnetite gabbros and gabbronorites do occur as intersected in drill holes WB001 and WB002. On the south of the project property (Disseldorp), where Hole WB001 was drilled, a 2.5m thick magnetite was intersected.

Generally, the Bushveld package strikes south-west to northeast with a general dip of 34º - 38º towards the west is observed from drill hole cores for the layered units intersected on Waterberg property within the Bushveld package. However, some blocks may be tilted at different angles depending on structural and/or tectonic controls.

The Bushveld Upper Zone is overlain by a 120m to 760m thick Waterberg Group, which is a sedimentary package predominantly, made up of sandstones, and within the project area, the two sedimentary formations, known as the Setlaole and Makgabeng Formations, constitute the Waterberg Group. The Waterberg package is flat lying with dip angles ranging from to 2º to 5º.

The base of the Bushveld package is marked by the presence of a transitional agmatite zone that constitutes a mixed zone of Bushveld magmatic rocks and granitic veining from a melted gneissic floor. The cores typically end in Archaean granite gneiss basement.

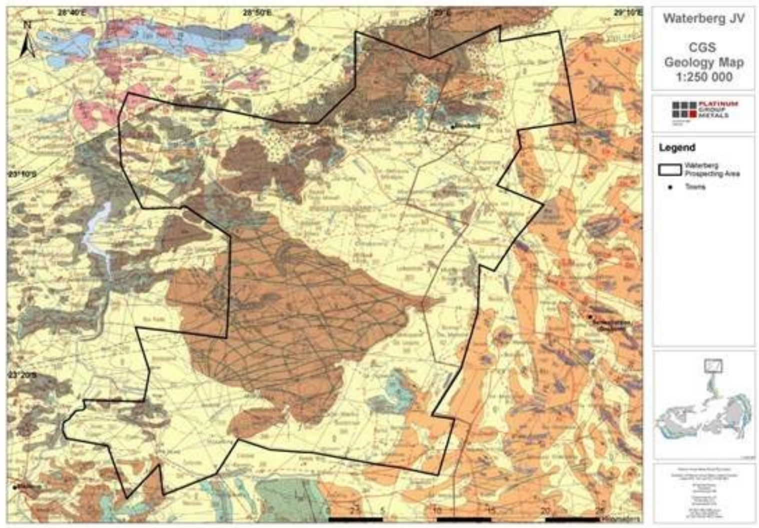


Figure 7-5: Surface Geology of the Waterberg Project

Structurally, the area has abundant intrusives in the form of thick dolerite, diorite and granodiorite sills or dykes predominantly in the Waterberg package. A few thin sills or dykes were intersected within the Bushveld package. Faults were interpolated from the aerial photographs, geophysics and sectional interpretation, and drilling. The faults generally trend east-west across the property and some are north-west and south-west trending (Figure 7-6)

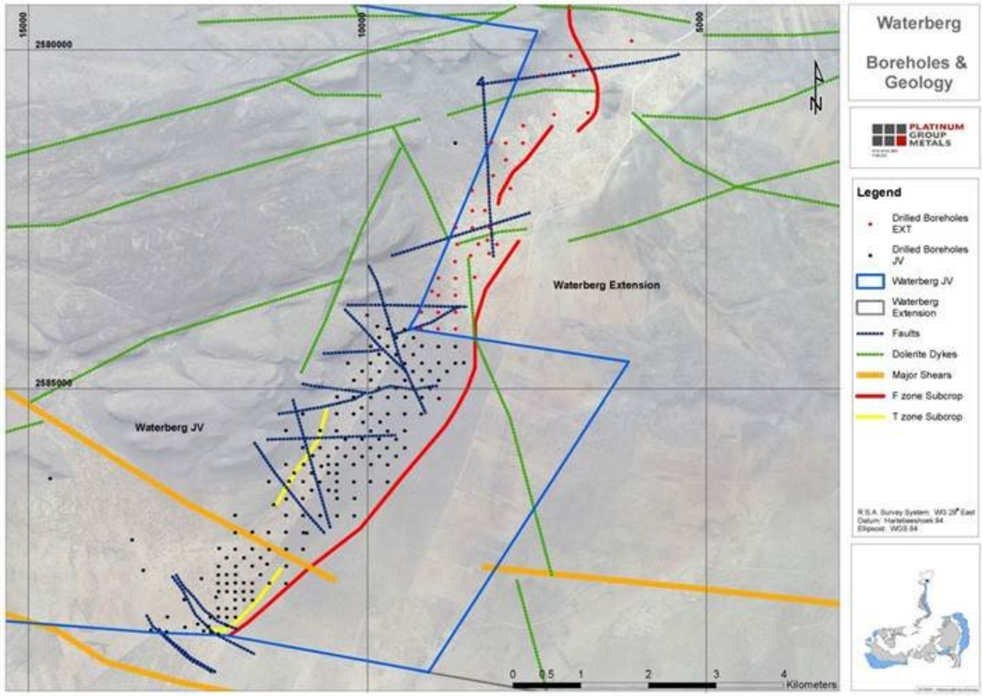


Figure 7-6: Project Geology of the Waterberg Project

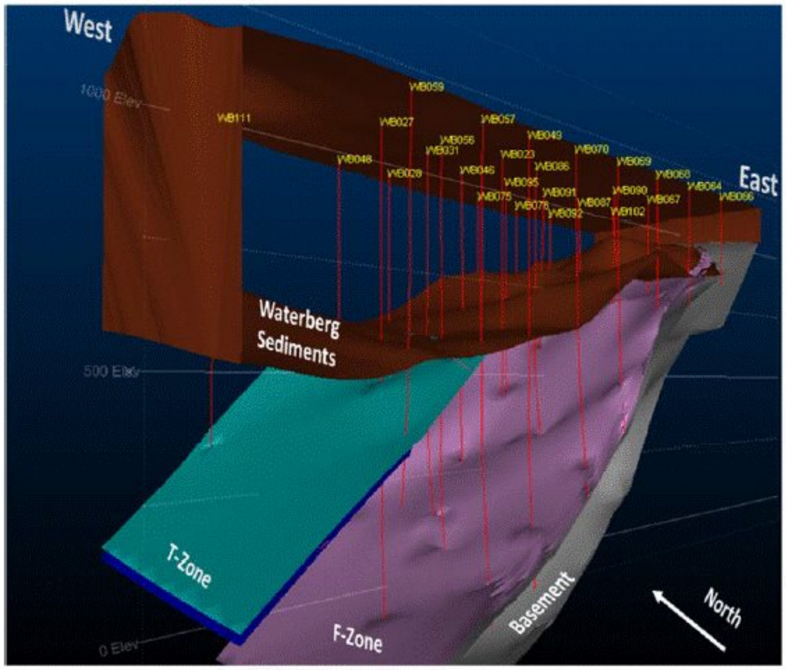


Figure 7-7: Section showing Geology at the Waterberg Project

The project geology in the northeastern portion of the Waterberg Project appears to be similar to the geology in the southeast. However, due to the widely spaced drilling further north, the project geology is less well understood.

There is a general increase in the frequency of late intrusive rocks in the form of dolerite, diorite and granodiorite dykes predominantly in the Waterberg package. A few thin sills or dykes were intersected within the Bushveld package. The dolerite dykes have a variable positive magnetic response and were modelled in 3D from the detailed airborne magnetic data as being vertical to a minimum depth of 300m. Field mapping confirms the vertical nature of the dykes and recessive weathering nature on surface. The sills and dykes are of similar composition however, the interrelation of the two is currently not known. Many of the east-west dykes appear to have exploited pre-existing structures such as major shears and faults.

A flat lying dolerite sill with an average thickness of 80m appears to be exploiting the contact between the Bushveld Complex igneous rocks and the overlying Waterberg sedimentary rocks (Figure 7-7). This sill, as seen in drill holes, displays both an upper and lower chill margin indicating post Waterberg emplacement. The sill outcrops to the east of the projected edge of the Bushveld and forms low, flat top hills. Using the depth of the sill intersections in drilling and the surface outcrop pattern to the east there appears to be a kink in the dip of sill at or near the projected Bushveld edge that explains the vertical difference in the position of the sill between surface and downhole.

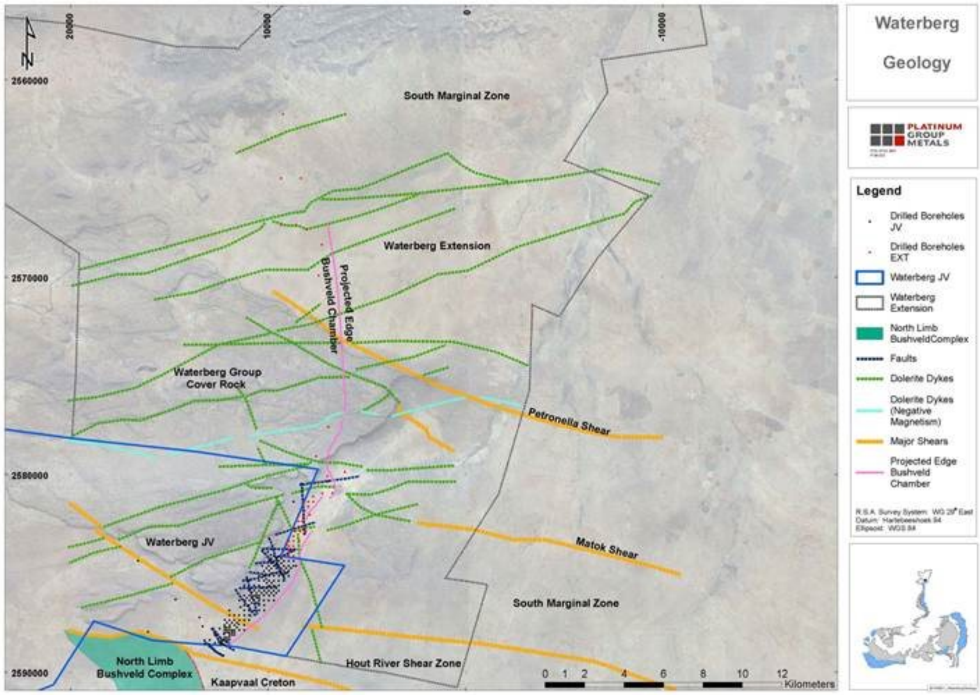


Figure 7-8: Geology of the Northern Waterberg Extension Area

7.4 Stratigraphy

The initial phase of diamond exploration drilling (WB001 and WB002) on the Waterberg JV Project intersected Waterberg Group Sediments (sandstones) and Bushveld Upper Zone and Main Zone lithologies in the western portion of Disseldorp property. The follow-up drilling campaign revealed a generalized schematic stratigraphic section that was adopted for use in this property as presented in Figure 7-8.

The initial phase of diamond exploration drilling on the Waterberg Extension Project has intercepted similar stratigraphy to the adjacent and contiguous Waterberg JV Project to the south. Generally, the layers correlate well between the projects and at the Waterberg Extension Project on the farm Early Dawn 361LR, drilling has intersected Waterberg Group sediments (sandstones) and Bushveld Complex Main Zone lithologies.

The floor rocks underlying the transitional zone are predominantly granite gneiss hosting remnants of magnetite quartzite, metaquartzite, metapelites, serpentinites and metasediments. Some drill holes within the project area have shown dolerite intrusions within the floor rocks, such as in drill hole WB028. Pink pegmatoidal granite was noted in the basement of one drill hole on the Waterberg Extension Project.

7.5 Bushveld Complex

Igneous Bushveld Complex lithologies underlie the Waterberg Group starting with the Upper Zone and underlain by the Main Zone.

7.5.1 The Main Zone

The Main Zone which hosts the PGM mineralised layers in its cyclic sequences of mafic and felsic rocks, is 150m to 900m thick. It is predominantly composed of gabbronorite, norite, pyroxenite, harzburgite, troctolite with occasional anorthositic phases.

Abundant alteration occurs in these lithologies including chloritisation, epidotisation and serpentinisation. Parts of the F-Zone are magnetic due to the formation of secondary magnetite during serpentinisation of the olivines. The F-Zone forms the base of the Main Zone, and it is usually underlain by a transitional zone of intermixed lithologies such as metasediments, metaquartzite/quartzite, and Bushveld lithologies.

7.5.2 The Upper Zone

The southwestern part of the project area (west of the farm Ketting 368LR towards the farm Disseldorp) has a thick package of Upper Zone lithologies. The package consists of magnetite gabbro, mela-gabbronorite and magnetite seams and may be as thick as 350m. Drill hole WB001 on farm Disseldorp collared in Upper Zone and drilled to the depth of 322m and while still in the Upper Zone intersected a 2.5m thick magnetite seam.

The appearance of the first non-magnetic mafic lithologies indicates the start of the underlying Main Zone.

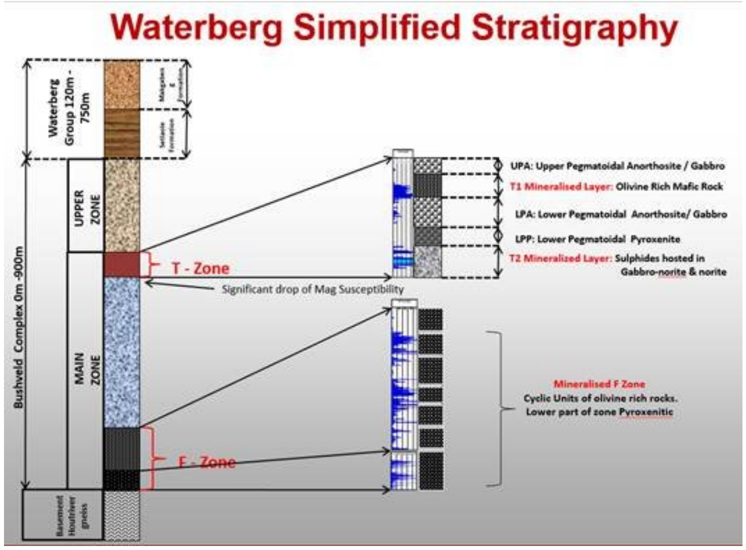


Figure 7-9: General Stratigraphy of the Waterberg Project

7.6 Waterberg Group

The Waterberg sedimentary package occurs with mostly two formations within the project area i.e. the Makgabeng and Setlaole Formations. The whole package may have a thickness ranging from 120m to just over 760m. Generally, the Waterberg sedimentary package thickens in the southwest and thins towards the center of the project area before thickening again to the north. The east-west trending feature through the southern part of the Project is considered an erosional channel.

7.6.1 Setlaole Formation

This is the sedimentary formation underlying the Makgabeng Formation at the base of the Waterberg Group sedimentary succession. A formation that overlies the Bushveld Complex igneous rocks, and was intersected in more than 90% of the drill holes within the project area.

Lithologically, the Setlaole Formation consists of medium- to coarse-grained sandstones and several mudstones and shales that have a general purple color and usually the package displays a coarsening-down sequence. Towards the base of the formation, pebbles may be seen that will eventually appear to be forming a conglomerate. The rocks are frequently intruded by dolerite and granodiorite sills. A red shale band of variable thickness is generally present below the basal conglomerate of the Setlaole Formation. This represents the eroded roof of the Bushveld Complex.

7.6.2 Makgabeng Formation

This sedimentary formation overlies the Setlaole Formation and is mostly exposed in the mountain cliffs in the northern part of the project area. The formation is composed of light- red colored banded sandstone rocks and is generally flat lying.

7.7 Structure

Numerous crisscrossing dolerite or granodiorite sills or dykes intersected the Waterberg sedimentary package. These usually range from as thin as 5cm to as thick as 90m.

A major northwest-southeast trending fault was inferred based on drill holes towards the southern part of the Ketting 368LR property. The fault throw is estimated to be approximately 300m. A further fault splay has also been interpreted on the southeastern part of Ketting 368LR.

7.8 Mineralised Zones

PGM mineralization within the Bushveld package underlying the Waterberg sediments is hosted in two main layers: the T-Zone and the F-Zone.

The T-Zone occurs within the Main Zone just beneath the contact of the overlying Upper Zone. Although the T-Zone consists of numerous mineralised layers, three potential economical layers were identified, T1, T2HW and T2 - Layers. They are composed mainly of anorthosite, pegmatoidal gabbros, pyroxenite, troctolite, harzburgite, gabbronorite and norite.

The F-Zone is hosted in an olivine-rich package towards the base of the Bushveld Complex. This zone consists of alternating units of harzburgite, troctolite and pyroxenites. The F-Zone has been divided into the FH and FP layers. The FH layer has significantly higher volumes of olivine in contrast with the lower lying FP layer, which is predominately pyroxenite. The FH layer is further subdivided into six cyclic units chemically identified by their geochemical signature, especially chrome. The base of these units can also be identified lithologically by a pyroxenite layer.

The mineralization generally comprises sulphide blebs, net-textured to interstitial sulphides and disseminated sulphides within gabbronorite, norite, pyroxenite, troctolite and harzburgite.

Within the F-Zone, basement topography may have played a role in the formation of higher grade and thicknesses where embankments or large scale changes in magma flow direction facilitated the accumulation of magmatic sulphides. These areas are referred to by PTM as the “Super F” Zones where the sulphide mineralization is over 40m thick and within the defined areas average 3g/t to 4g/t 2PGE+Au. Layered magmatic sulphide mineralization is generally present at the base of the F-Zone. As with the T-Zone, the sub-outcrop of the F-Zone unconformably abuts the base of the Waterberg Group sedimentary rocks and trends northeast from the end of the known Northern Limb and dips moderately to the northwest.

The T-Zone includes a number of lithologically different and separate layers (Figure 7-10) which were initially recognized in the drilling. With subsequent drilling, it has become clear that the most easily identifiable and consistent are the T1, T2HW and T2 Layers.

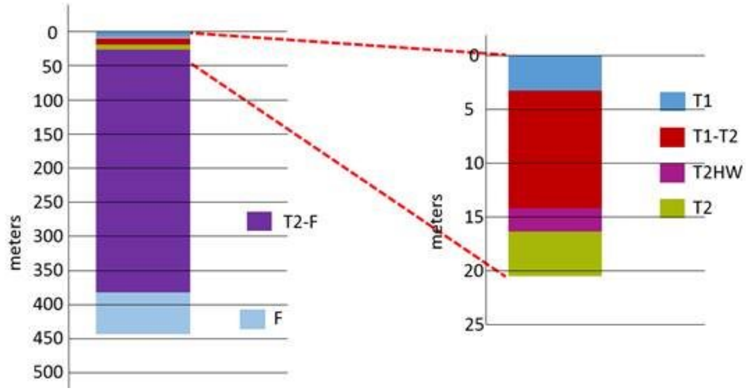


Figure 7-10: Stratigrapy of the Mineralised Layers

7.8.1 Description of T-Zone Layering and Mineralisation

The T-Zone is a linkable unit, which includes five identifiable layers. The three mineralised and economical potential layers are the T1 Layer, the T2HW Layer (Middling between T1 and T2), and the T2 Layer. Figure 7-11 is a geological interpretation of the T-Zone layers.

UPA (Upper Pegmatoidal Anorthosite) — This is the T1 Layer hangingwall, which has a pegmatoidal texture, is mostly anorthositic and in a few cases gabbroic. This unit is generally not mineralised however it was found to have some sulphide mineralization in a few drill holes and the mineralization is hosted within the pyroxenes of the pegmatoid.

This unit has a thickness ranging from 2m up to 100m, and can be correlated in over 80% of the drill holes. It must be noted that this unit is absent in a few drill holes and it also appears more mafic in some instances.

Mineralisation within the T1 Layer is hosted in a troctolite with variations in places where troctolite grades into feldspathic harzburgite. In other localities, olivine bearing feldspathic pyroxenite grades into feldspathic harzburgite. The 4E grade (g/t) is typically 1-7g/t with a Pt:Pd ratio of about 1:1.7. The Cu and Ni grades are typically 0.08% and 0.08% respectively.

The unit is mineralised with blebby to net-textured Cu-Ni sulphides (chalcopyrite/pyrite and pentlandite) with minimal Fe-sulphides (pyrrhotite). The thickness of the layer varies from 2m to 6m.

The direct footwall unit of the T1 Layer can be divided into two identifiable units: the Lower Pegmatoidal Anorthosite (LPA) and the Lower Pegmatoidal Pyroxenite (LPP). These units have an unconformable relationship with one another, as both are not always present.

LPA (Lower Pegmatoidal Anorthosite) — This is the first middling unit underlying the T1 Layer. It has the same composition as that of the UPA but is usually thinner than the UPA. The thickness for this unit ranges from 0 — 3m, and in some drill holes this unit is not developed. This unit is mineralised in some drill holes.

LPP (Lower Pegmatoidal Pyroxenite) — This is the second middling unit, which underlies the LPA, is predominantly composed of pegmatoidal pyroxenite. It also ranges from 0 — 3m as it is not developed in all drill holes. This unit also forms the hangingwall to the T2 Layer. Mineralisation has not been identified in this unit.

Mineralisation within the T2 Layer is hosted in Main Zone norite and gabbronorite that shows a distinctive elongated texture of milky feldspars. In some instances, the T2 gabbronorite/norite tends to grade into pyroxenite and in places into a pegmatoidal feldspathic pyroxenitic phase, with the same style of mineralization as in the gabbronorite/norite. Lithologically, the T2 Layer is generally thicker than the T1 Layer. The high grade zones range from 2m to approximately 10m within these lithologies. Sulphide mineralization in the T2 Layer is net textured to disseminate with higher concentration of sulphides compared to the overlying T1 Layer. The 4E grade (g/t) is typically 1-6g/t with a Pt:Pd ratio of about 1:1.7. The Cu and Ni grades are typically 0.17% and 0.09% respectively.

The mineral resource estimate used the data to define the characteristics of the various layers based on their geological characteristics and geochemical signatures (Figure 7-11).

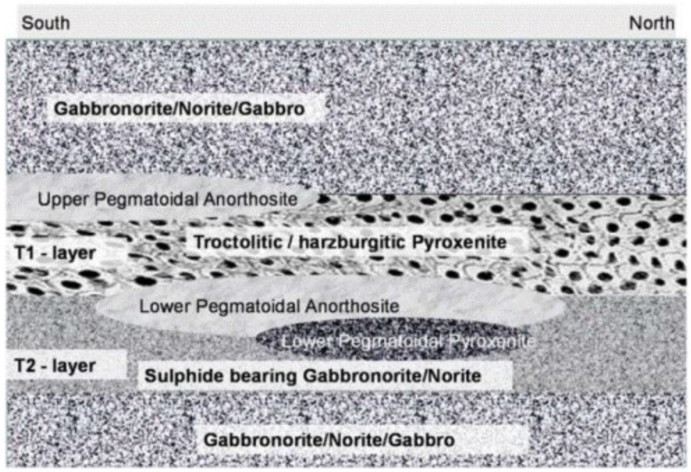


Figure 7-11: Waterberg Project A Geological Interpretation of the T-Layer

7.8.2 Description of F — Zone Layering and Mineralisation

A thick package of norite and gabbronorite ranging from 100m to about 450m underlies the T-Zone and overlies the F-Zone.

F-Zone mineralization is hosted in a thick package of troctolite, which usually has small bands of pyroxenite and/or pegmatoidal pyroxenite and harzburgite. These layers or pulses were identified using their geochemical signatures and various elemental ratios. The initial subdivision was into a harzburgitic layer (FH) which is underlain by a pyroxenitic layer (FP). The harzburgitic layer (FH) was further subdivided into six units of varying thickness based on the noted significant occurrence of chrome in the geochemical signature (Figure 7-12). In each case, the concentration of the chrome falls off steadily going up in the sequence until the next significant occurrence of chrome is noted.

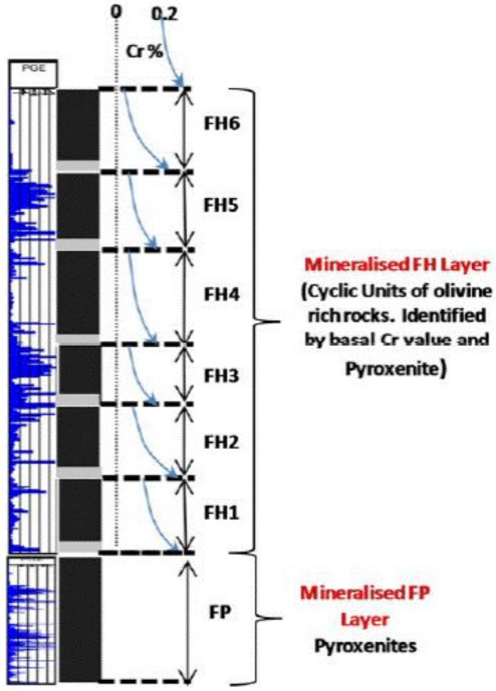


Figure 7-12: Individual Units of the F-Zones

8. Deposit Types

The Waterberg deposit is a magmatic layered Bushveld type of palladium and platinum mineralization with significant copper and nickel. The deposit is confirmed by age-dating by Wits University to be Bushveld in age. The host rocks are magmatic high iron magnesium rocks, classic for magmatic copper nickel deposit types. The deposit has a consistent dip and strike in general terms with a value zone in a rock package with distinctive markers. In detail, the value zone crosses small scale lithologic layers.

The exploration and development plan is based on a layered deposit and a mechanized potential mine plan.

9. Exploration

9.1 Current Exploration

The Waterberg Project is at an advanced exploration stage and includes Inferred and Indicated Mineral Resource estimates. Exploration further north has investigated the interpreted strike extension of the Bushveld Complex. Because of this drilling programme, portions of this area are classified as an Inferred Mineral Resource.

Previous mineral exploration activities were limited due to the extensive sand cover and the understanding that the area was underlain by the Waterberg Group. Initial exploration was driven by detailed gravity and magnetic surveys. Subsequently exploration was driven by drilling and was undertaken by PTM.

Detailed engineering including metallurgy, rock mechanics, infrastructure design work and mine planning has been completed in this PFS and is ongoing.

9.2 Surface Mapping

Topographical and aerial maps for Waterberg at a scale of 1:10,000 were used for surface mapping. A combination of the surface maps and the public aeromagnetic and gravity maps formed the basis for the structural map.

Ground exploration work undertaken includes geological mapping and ground verification of the geology presented in various government and academic papers. The major faults and SMZ geology described was confirmed to exist within the property. Contact relationships with the Bushveld Complex were not seen due to the Waterberg cover rocks and Quaternary sand deposits.

Data for any outcrop observed (or control point) was recorded. Each of such outcrop points had the following recorded in the field book: point's name, description of the outcrop's rock, identified rock name, XY coordinate points, and if well oriented the dip and strike for the outcrop.

It is noted that most of the area surrounding the Waterberg Mountains is covered by recent sands and as such, mapping in these areas has provided minimal information. Access to some parts of the Waterberg Mountains is problematic due to steep slopes close to the mountains.

9.3 Geochemical Soil Sampling

In March 2010, two north-south sampling lines (Figure 9-1) were undertaken. Sampling stations were made at intervals of 25m. Each sample hole was allowed to go to a minimum depth of 50cm to 1.00m at most.

During December 2011 and January 2012, two additional north-south lines on the property Niet Mogelyk 371LR were also sampled (Figure 9-1). These two lines were done to target the east-west trending dykes that are running through this property and the sampling stations were set at 50m apart.

During January 2013 an additional three lines were taken on the farms Bayswater 370LR and Niet Mogelyk 371LR. These samples were taken to investigate soil anomalies discovered by the previous sampling programs (Figure 9-1)

723 samples, of which 367 were soil samples, 277 stream sediment samples and 79 rock chip samples, were collected during this process.

Geochemical sampling of the soils was also partially compromised due to very thin overburden because of sub cropping rock formations. Geochemical sampling showed elevated PGM's and this increased exploration interest in the area in 2011.

9.4 Geophysical Surveys

Initial detailed ground geophysical surveys were confined to the Waterberg JV Project and were funded by the partner JOGMEC. The detailed airborne survey was completed predominantly over the Waterberg Extension Project, with some overlap beyond the defined edge of the Bushveld geology to include part of the advanced stage Waterberg JV Project to obtain response characteristics.

9.4.1 Initial Survey

Approximately 60 lines of geophysical survey for 488 line kilometer using gravity and magnetics were traversed in March 2010 (Figure 9-2). These were east — west trending lines and were traversed on the farms Disseldorp 369LR, Kirstenspruit 351LR, Bayswater 370LR, Niet Mogelyk 371LR and Carlsruhe 390LR. At this time, the Prospecting Right for the farm Ketting 368LR was still pending.

As soon as Ketting 368LR was granted, a second phase of geophysical survey was also conducted on this farm from mid-August 2011 to September 2011 (Figure 9-1).

Two additional north-south ground magnetic lines were surveyed over the farm Ketting 368LR in November 2012. This information was used to interpret and locate east-west striking structures (Figure 9-1).

When considering the Waterberg Extension, due to the presence of Waterberg Group cover rocks, there was no exposure of Bushveld Complex rocks on the property. Geophysical techniques were employed to aid in the modelling of the projected Bushveld Complex. Comparing the projected edge of the Bushveld Complex from the regional geophysics modelling, the FALCON airborne survey interpretation and the ground gravity profiles, there is general correlation, with local variations, of a north-northeast arc where the edge of the denser mafic intrusive rocks may project beneath the Waterberg sediment cover.

9.4.2 Extended Airborne Gravity Gradient and Magnetics

An airborne gravity survey was completed on 100 and 200m line spacing. An interpretation of the results of the survey suggests that there may be continuity to the Bushveld Complex rocks to the north-west and north, which has the potential to host PGM mineralization to the northeast within the Project area.

PTM contracted FUGRO Airborne Surveys (Pty) Ltd. to conduct airborne FALCON® gravity gradiometry and total field magnetic surveys. The target for the survey was the interpreted edge sub-crop of the Bushveld Complex. The survey conducted in April 2013, was comprised of 2306.16 line kilometers of Airborne Gravity Gradiometry (AGG) data and 2469.35 line kilometers of magnetic and radiometric data. The total extent of the survey covered approximately 25km of interpreted Bushveld Complex edge in the northeastern part of the project area (Figure 9-2).

Interpretation was based on creating a starting model using the known geology from drilling and linking it to the airborne response (Figure 9-3 and Figure 9-4). The geological units were modelled in three dimensions in order to facilitate a three-dimensional stochastic inversion of the geometry and density of the units making use of the gravity gradient data. Average rock unit densities were extrapolated from the adjacent Waterberg Project.

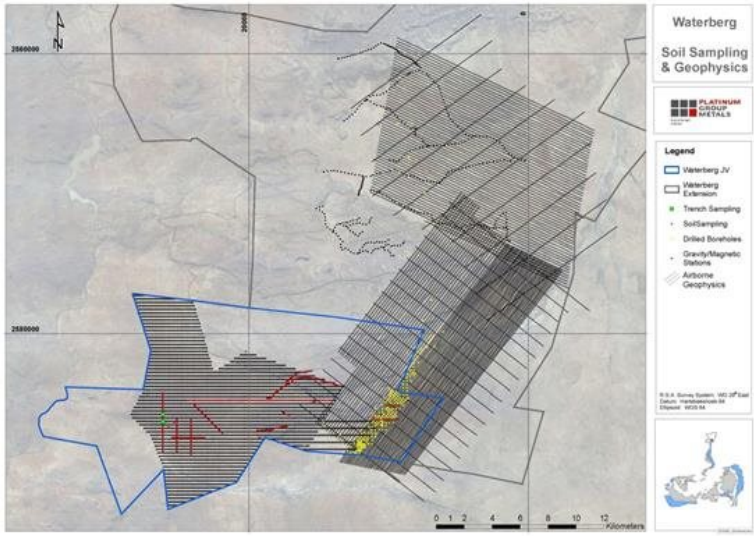


Figure 9-1: Locations of Geochemical Sampling and Geophysical Traverses

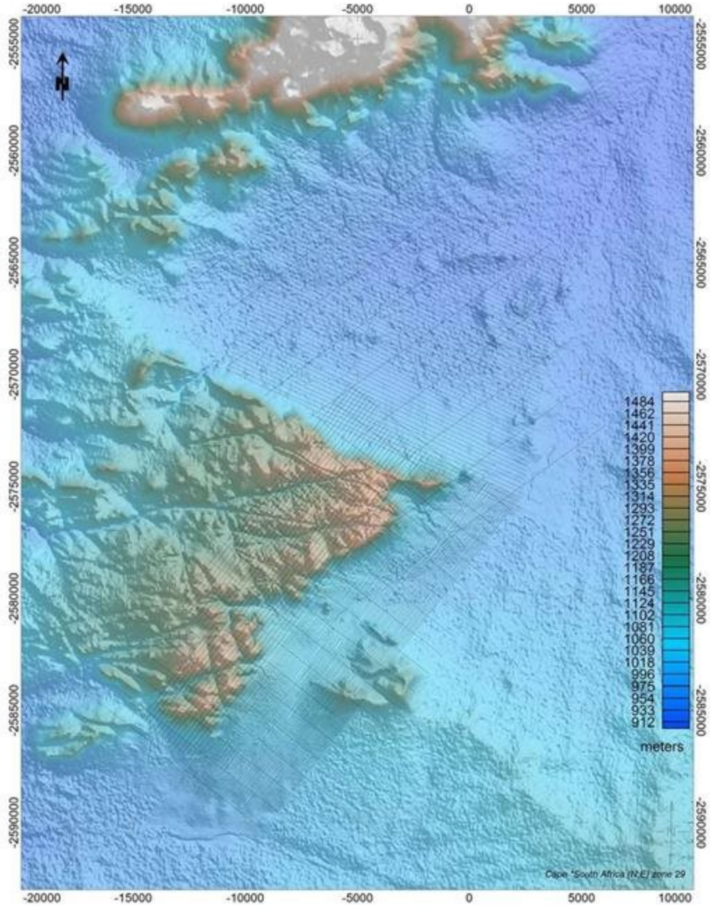


Figure 9-2: Airborne Gradient Gravity and Magnetic Survey Flight Lines

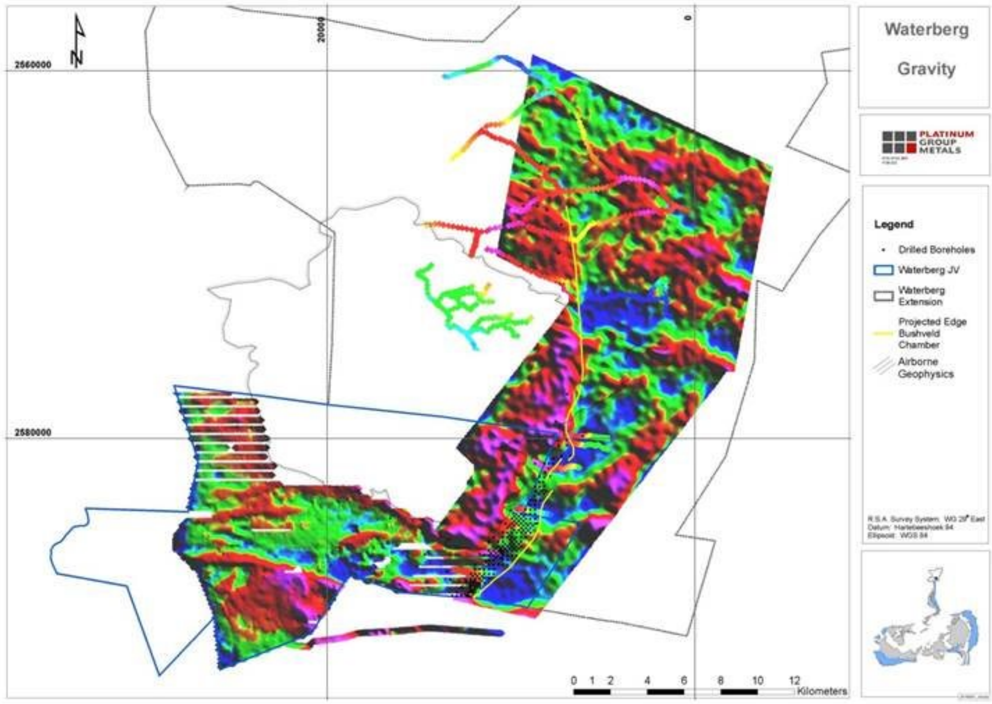


Figure 9-3: Waterberg Project Airborne Gradient Gravity Plot with Interpreted Bushveld Complex Edge

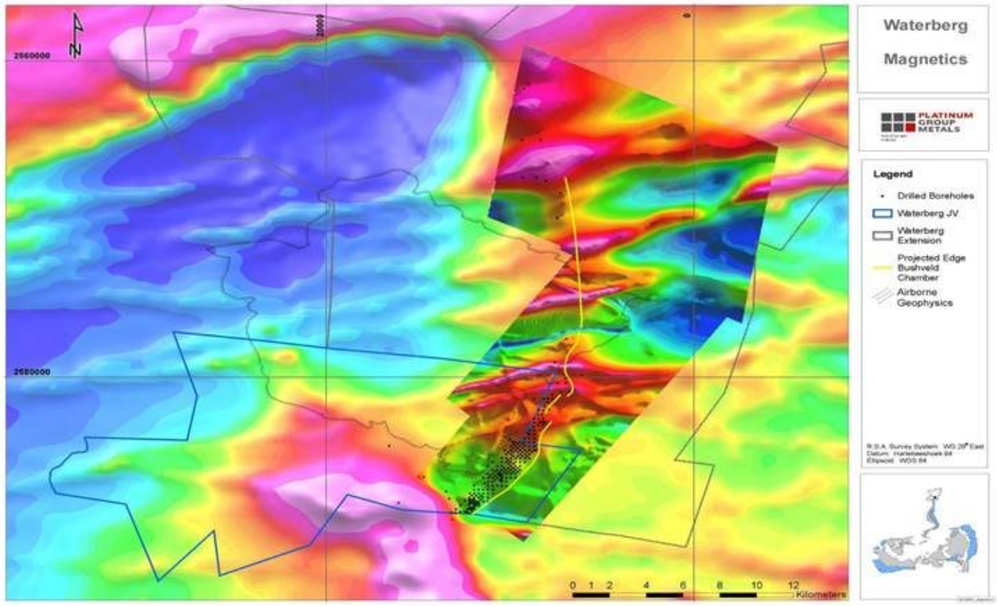


Figure 9-4: Airborne Total Field Magnetics Plot with Interpreted Bushveld Comlex Edge

9.4.3 Ground Gravity

Geospec Instruments (Pty) Ltd along roads and tracks completed nine ground gravity traverses. The survey lines were designed to traverse across the projected edge of the Bushveld Complex in the same area covered by the airborne survey as ground confirmation of the airborne results. The two surveys were compared and a good correlation between gravity data sets was noted. In planning the ground survey, one control line over the known deposit edge at the point where it projected from the southern part of the project was completed in order to acquire a signature profile over a known source to compare the remaining regional lines to. The interpretation of the linked ground gravity profiles suggests that there may be a northwest trending continuity to the Bushveld Complex rocks, which have the potential to host PGM mineralization.

9.5 CJM — Technical Review

Suitable exploration was undertaken with appropriate conclusions and follow-up work completed.

- Ashwal, L. D., Webb, S.J. and Knoper, M.W. (2005). Magmatic stratigraphy in the Bushveld Northern Lobe: continuous geophysical and mineralogical data from the 2950m Bellevue drillcore. South African Journal of Geology 108(2): 199-232

- Kinnaird, J. A., Hutchinson, D., Schurmann, L., Nex, P.A.M., and de Lange, R. (2005). Petrology and mineralization of the southern Platreef: northern limb of the Bushveld Complex, South Africa. Mineralium Deposita 40(5): 576-597.

10. Drilling

Drilling was done by specialized contractors, Discovery Drilling (Pty) Ltd mobilized out of Marken town, South Africa. All drilling is done by diamond drill coring and are near vertical at their collars. Generally speaking, holes are drilled using NQ core (47.6 mm); occasionally necking down to BQ if poor ground conditions are encountered or deep drilling is required. Metallurgical holes were drilled using NQ sized core. Table 10-1 summarizes the drilling by year.

Table 10-1: Summary of Drilling by Year

Year	No of Holes	Deflections	Total Meters	Cumulative Meters	Results	Database End Drilling Date
2010	2	2	1935	1935	T-Zone Discovery	
2011	1	3	1773	3708	F-Zone Discovery	
2012	27	62	32453	36161	Resource Delineation Drilling	January 2012-September 2012
2013	84	168	94528	130689	Resource Estimate 2013-09-03	October 2012-July 2013
2014	35	47	31880	162569	Resource Estimate, 2014-06-12	October 2013-May 2014
2015	82	99	86176	248745	Resource Estimate, 2015-07-20	June 2014-Feb 2015
2016	70	98	49636	298381	Resource Estimate, 2016-04-19	May 2015-April 2016
2016	2	4	2494	300875	Resource Estimate, 2016-09-09	May 2016-
Totals	**303**	**483**	**300875**			

The average drill hole length is 722 m, the minimum drill hole length is 200.20 m (WE074) and the maximum drill hole length is 1643.39 m (WB004).

10.1 Drilling in 2010

Based on the target generation and the results of the geochemical sampling and geochemical surveys, two drill holes WB001 and WB002 were initially drilled between July and October 2010 on the farm Disseldorp 369LR. A total of 1,934.77m was drilled for the first two drill holes in 2010. These holes intersected the "T" layers of mineralization.

10.2 2011 Drilling

Drilling resumed in 2011 with a third drill hole WB003 drilled on the farm Ketting 368LR. This hole cut both "T" and "F" zone mineralization.

10.3 2012 Drilling

These drill holes lead to the 2012 drill campaign, which delineated a portion of the Waterberg mineralization. In 2012 30 268.52 m in 27 holes with 62 deflections were completed. This work delineated the southern portion of the Waterberg Deposit.

10.4 2013 Drilling

A total of 128,505m of core had been drilled by September 2013, the cut-off date of the mineral resource estimate. The results of 112 drill holes were available for the mineral resource estimate. A basic 250m x 250m grid drilled grid was used to place the drill holes where possible.

Drilling in some areas proved difficult due to bad ground formations particularly in the Waterberg sediments and so some drill holes had to be re-drilled a few metres away or totally abandoned or moved.

Diamond drilling commenced towards the northeast in October 2013 upon the official granting of the prospecting right for the Waterberg Extension Project. The initial drill hole locations were chosen to test the interpreted northeast strike continuation of the Bushveld Complex edge and mineralised layers defined on the adjacent Waterberg Project with step outs of 1 to 2km. Six diamond drill machines were mobilized. Eight of the nine initial drill holes intersected Bushveld Complex stratigraphy.

10.5 2014 Drilling

The 12 June 2014 resource estimate dataset consisted of 153 drill holes, 278 deflections and 163,384 m.

10.6 2015 Drilling

The initial database for this mineral resource estimate was received on 22 April 2015. The raw database consisted of 231 drill holes with 373 deflections totaling 248,748 m. The southern JV area contains 182 holes and 303 deflections and the northern Extension area contains 49 drill holes with 70 deflections.

10.7 Exploration Drilling Status as at 1 April 2016

The initial database for this mineral resource estimate was received on April 1, 2016. The raw database consists of 294 drill holes with 459 deflections totaling 298,538 m.

10.8 Exploration Drilling Status as at 7 July 2016

The initial database for this mineral resource estimate was received on July 7, 2016. The raw database consists of 303 drill holes with 483 deflections totaling 300,875 m.

10.9 Collar Surveys

A contracted certified land surveyor used a differential Trimble GPS system to conduct collar surveys on all completed holes. Stations were tied in with survey stations established by the National Survey General Directorate. Test work coordinates were given in the Hartebeesthoek 1994 LO29 national coordinate system.

10.10 Downhole Surveys

Downhole surveys are done on 1m intervals using a gyroscopic (gyro) tool with some older holes using an electronic multi-shot (EMS) tool. Deflections are done using a gyroscopic (gyro) tool. There are five mineralised, vertically drilled motherholes that were not surveyed due to bad ground conditions (WB108 - 427.60m, WB110 - 1276.47m, WE006 - 498.23m, WE016 - 883.80m and WE025 - 736.28m).

10.11 Drilling Quality

CJM has examined core from randomly selected drill hole cores. The core recovery and core quality meet or exceed industry standards. The quality of the work in the drilling programs is excellent.

Drilled core is cleaned, de-greased and packed into metal core boxes by the drilling company. The core is collected from the drilling site on a daily basis by PTM personnel and transported to the exploration office. At no time is the core left unattended at the rig. Before the core is taken off the drilling site, the depths are checked and entered on a daily drilling report, which is then signed off by PTM. The core yard manager is responsible for checking all drilled core pieces and recording the following information:-

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

Each core box is photographed using a digital camera from fixed vertical distance. The photographs are stored on a network server.

10.12 Geological Logging

Standardized geological core logging conventions were used to capture information from the drill core. Detailed geological logging was completed daily by qualified geologists onto a PTM pro-forma capture sheets under supervision of the project geologist.

Geological core logging involved the recording of lithology (rock type, grain size, texture, angle to the core axis, top and bottom contact types, color and optional comments); stratigraphic units; type and degree of alteration (infill, partial, or pervasive); and mineralization (type, style and visible %age of sulphides).

Three magnetic susceptibility readings are taken and averaged together from the beginning of the BC lithologies to the end of hole at 1 m intervals.

Once the geological logging is captured into the SABLE database on site, the logs are printed and a qualified geologist checks the core against the captured logs to verify that the data has been recorded and captured correctly. The printed logs are then signed off and stored in the borehole file.

All data captured into the field SABLE database is extracted and emailed to Johannesburg Head Office on a weekly basis, or more often if required. The data is loaded into the master SABLE database and checked.

All documentation relating to each borehole including geological logs, survey certificates, collar certificates, sampling sheets, assay certificates etc. are collated and filed in a file for each borehole located in Marken. All documentation is scanned and sent electronically to the head office in Johannesburg and saved on the server along with all available digital photographs.

10.13 Diamond Core Sampling

Qualified geologists based on a minimum sample length of approximately 25cm — 50cm, undertook sample selection. Not all drill hole cores were sampled, but all cores with visually identifiable sulphide mineralization was analyzed, and low grade to waste portions straddling these layers have been sampled. A maximum sample length of 1m was applied where appropriate. The true width of the shallow dipping (30° to 35°) mineralized zones that were sampled are approximately 82% to 87% of the reported interval from the vertical drill hole.

The sampled core is split using an electric powered circular diamond blade saw. Samples are cut according to the sampling sheet created by the geologist logging the hole.

10.14 Core Recovery

Core recoveries, RQD (Rock Quality Designation) and a note of core quality, are recorded continuously for each drill hole and for each drill run. The core recovery within the first few metres of boreholes (approximately 5 m) is poor in most cases due to the associated soil horizon classified as overburden. Poor recovery occasionally extended to about 30 m depth due to the weathering of bedrock. However, in the majority of instances, core recovery improved considerably once drilling reached the Main Zone hanging-wall, reef horizons and footwall rocks, and in these units was commonly 100%. The recoveries only show a substantial decrease within faulted/sheared zones.

10.15 Sample Quality

CJM has examined selected drill holes and has assessed the quality of sampling to meet or exceed industry standards.

10.16 Interpretation of Results

The results of the drilling and the general geological interpretation are digitally captured in SABLE and a GIS software package named ARCVIEW. The drill hole locations, together with the geology and assay results, are plotted on plan. Regularly spaced sections are drawn to assist with correlation and understanding of the geology. This information was useful for interpreting the sequence of the stratigraphy intersected as well as for verifying the drill hole information.

10.17 CJM — Technical Review

Suitable drilling was undertaken with appropriate standards in place to ensure that the data is suitable for use in geological modelling and mineral resource estimation on the Waterberg Project. Further exploration drilling is planned.

In the opinion of CJM the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programmes are sufficient to support Mineral Resource estimation as follows:

- Core logging meets industry standards for PGE—Au—Ni—Cu exploration.
- Collar surveys and downhole surveys have been performed using industry-standard instrumentation.
- Recovery from core drill programmes is acceptable to allow reliable sampling to support Mineral Resource estimation.

11. Sample Preparation, Analyses and Security

Waterberg Project staff members were responsible for the following:

- Sample collection
- Core splitting
- Sample dispatch to the analytical laboratory
- Sample storage
- Sample security

11.1 Sample Methods

Once geological logging is complete and validated, the qualified geologist identifies the units to be sampled based on stratigraphic, lithological and visible sulphide mineralization criteria. Continuous sampling from the top of the mineralized zone to well below footwall contacts is undertaken. The geologist varies the thickness of sampling intervals according to changes in stratigraphy, lithology and mineralization so as to ensure that samples do not cross-cut these boundaries. Areas of core loss are recorded and depths of the samples are carefully noted to exclude these intervals. Samples vary from 25 cm to 1.5 m in thickness.

The geologist prepares the sampling instruction sheet for the samples. Sample depths, sample numbers, blanks and standards to be inserted and into what positions are provided. A blank is inserted for one in every 10th sample and a standard (CRM) is also inserted for one in every 10th sample. The result is that there is a quality control sample after every five primary samples.

Before any sampling takes place, the core is orientated and secured together with tape where it is broken in places. A continuous line, marking the estimated plane of symmetry, is drawn on the core by the sampling geologist to ensure that all cores are split correctly.

Drill core is cut using a wet saw. The split core is placed back in the core tray and put in the sun to dry. When the core is dry, samplers mark the sampled intervals and the sample number on the core on both the section of core to be sampled and the core remaining in the tray as instructed from the sample sheet. The section of core to be sampled is placed in a plastic bag with a sample ticket from the ticket book.

For inserted standards (CRM's), the label identifying the standard is removed and stored in a separate bag for reference purposes. The sample number assigned to the standard is written on the standard label itself. All the CRM labels are filed in the Marken office and are checked if there are any queries. The sachet is placed in a sample bag with the sample ticket.

For blanks, material is placed in the sample bag with the corresponding sample ticket.

The sample bags are then sealed and the sample number written on the bag itself. The sample in the bag is then weighed and the weight in grams recorded on the sample sheet.

Samples are placed together into a bigger bag and sealed prior to dispatch.

The sample instruction sheets are loaded into the SABLE database and validated. A copy of the sampling data is sent to Johannesburg Head Office where it is loaded into the Master Sable database and checked again.

11.2 Density Determinations

Routinely samples are subjected to bulk density determinations by the Archimedes immersion method on site at the core yard. Both the dry mass and the wet mass of the sample are recorded. This data is captured into the SABLE database and validated. The density (SG) is then calculated and matched to the assay results for that sample for modelling purposes.

The formula for Specific Gravity is as follows:

- Specific Gravity = Ma / (Ma-Mw)
- Where Ma = Mass in Air and Mw = Mass in Water

33,754 samples have been measure for bulk density. These densities are representative of the stratigraphic and lithological units used within the geological model.

11.3 Sample Preparation and Quality Control prior to Dispatch

The project geologist is responsible for timely delivery of the samples to the relevant laboratory. The supervising and project geologists ensure that samples are transported by PTM contractors.

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

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

11.3.1 Security

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

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

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

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

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

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

The QP of this report is satisfied with the level of security and procedures in place to ensure sample integrity.

11.4 Sample Analysis

The laboratories that have been used to date are Set Point Laboratories (South Africa), Bureau Veritas Testing and Inspections South Africa (Pty) Ltd as the primary laboratories and Genalysis (Perth, Western Australia) for referee samples.

Bureau Veritas Testing and Inspections South Africa (Pty) Ltd (Rustenberg, South Africa) has served both as a primary and as a referee laboratory for a sub-set of the samples (5,299 primary samples from the 2016 drilling program, 2,045 primary samples from previous drilling programs and 702 referee samples).

Set Point Laboratories and Bureau Veritas are both accredited by the South African National Accreditation System (SANAS).

The National Association of Testing Authorities Australia has accredited Genalysis Laboratory Services Pty Ltd, following demonstration of its technical competence, to operate in accordance with ISO/IEC 17025, which includes the management requirements of ISO 9001: 2000.

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

Samples are analyzed for Pt (g/t), Pd (g/t) and Au (g/t) by standard 25 g lead fire-assay using silver as requested by a co-collector to facilitate easier handling of prills as well as to minimize losses during the cupellation process. The resulting prills are dissolved with aqua-regia for ICP analysis.

After pre-concentration by fire assay and microwave dissolution, the resulting solutions are analyzed for Au and PGM's by the technique of ICP-OES (inductively coupled plasma—optical emission spectrometry).

The base metals (copper, nickel, cobalt, chromium and sulphur) are analyzed using ICP-OES (Inductively Coupled Plasma — Optical Emission Spectrometry) after a multi- acid digestion. This technique results in "almost" total digestion.

Samples submitted for Rh analysis are assayed by fire assay using palladium collection followed by ICP-OES. Currently samples containing more than 1 g/t PGE plus gold are routinely analyzed for Rhodium as part of the primary batch.

All pulp rejects and coarse rejects are returned to the Marken core yard for storage.

The assay results are reported to the PTM database manager in the Johannesburg Head Office as Excel spreadsheets via email. The Excel spreadsheets are imported directly into the SABLE database using customized import routines. There is no editing or manipulation of the Excel spreadsheet before import. Once imported, QAQC checks are done using SABLE software and in Excel.

11.5 Quality Assurance and Quality Control

PTM has a well-established and functional quality assurance or quality control (QA/QC) procedure. PTM are the custodians of the QAQC results. Over the history of the Project CJM Consulting has reviewed the findings of QAQC results for the purposes of establishing validity of the data for inclusion into the Mineral Resource estimation, with particular focus on the results since the last Resource Statement. To this end, data from Set Point and Genalysis was examined.

PTM has well established QAQC protocols. The following summarizes the PTM protocols for quality control during sampling:-

- The project geologist oversees the sampling process.
- The core yard manager oversees the core quality control.
- The exploration geologists and the sample technician are responsible for the actual sampling process.
- The project geologist oversees the chain of custody.
- The internal QP verifies both processes and receives the laboratory data.
- The database manager merges the data into the database and produces the SABLE sampling log with assay values.
- Together with the project geologist, the resource geologist determines the initial mining cut.

- An external auditor verifies the sampling process and signs off on the mining cut.
- A second external database auditor verifies the SABLE database and highlights QA/QC failures.
- QAQC graphs (standards, blanks and duplicates) and anomalies and failures are reported to the internal QP,
- Routine QA/QC analysis is done by the database manager on the receipt of each batch.
- If required, the database manager requests re-assay based on failures or anomalies identified.
- Referee samples are sent to Genalysis to verify the validity of data received from Set Point. Referee samples are randomly selected from all samples with PGE's plus Au > 1g/t.

Additional PTM QA/QC procedures include examination of all core trays for correct number sequencing and labelling. Furthermore, the printed SABLE sampling logs (including all reef intersections per drill hole) are compared with the actual remaining drill hole core left in the core boxes. The following checklist forms the standard PTM checklist for verification:

- Sampling procedure sample length between 0.25 to 1.2 m;
- Quality of core (core-loss) recorded;
- Correct packing and orientation of core pieces;
- Correct core sample numbering procedure;
- Corresponding numbering procedure in sampling booklet;
- Corresponding numbering procedure on printed SABLE log sheet;
- Comparing SABLE log sheet with actual core markings;
- Corresponding chain-of-custody forms completed correctly and signed off;
- Corresponding sampling information in hardcopy drill hole files and safe storage;
- Assay certificates filed in drill hole files;
- Electronic data from laboratory checked with signed assay certificate;
- Sign off each reef intersection (reef contacts and mining cut);
- Sign off completed drill hole file; and
- Sign off on inclusion of mining cut into Mineral Resource database.

As part of the sampling protocol, PTM regularly inserts QC samples (i.e. standards and blanks) into the sample stream. It should be noted that PTM do not include field duplicates into the samples stream, and the analytical laboratory was asked to regularly assay split coarse reject and pulp samples as duplicates to monitor analytical precision.

11.5.1 Analytical Quality Assurance and Quality Control Data

11.5.1.1 Standards (Certified Reference Materials)

Analytical standards/Certified Reference Materials (CRM's) were used to assess the accuracy and possible bias of assay values for Pt and Pd. Rh and Au were monitored where data for the standards were available, but standards were not failed on Rh and Au alone. Quality control data for the Waterberg Project is managed by PTM using SABLE Data Works software.

A selection of standards including some made from BIC mineralization is used in all sample submissions. The Standards currently in use are tabled in Table 11-1. These CRMs were purchased from commercial African Mineral Standards (AMIS), Johannesburg. The standards are stored in sealed containers and considerable care is taken to ensure that they are not contaminated in any manner (e.g. through storage in a dusty environment, being placed in a less than pristine sample bag or being in any way contaminated in the core saw process).

Table 11-1: Current Standards in Use

STD_REF_ID	PT_EV	PT_3STD_MIN	PT_3STD_MAX	PD_EV	PD_3STD_MIN	PD_3STD_MAX	AU_EV	AU_3STD_MIN	AU_3STD_MAX	CU_EV	CU_3STD_MIN	CU_3STD_MAX	NI_EV	NI_3STD_MIN	NI_3STD_MAX
AMIS0002	0.82	0.652	0.988	0.89	0.743	1.037	0.155	0.131	0.179	1310	1130	1490	1970	1745	2195
AMIS0110							2.3	2.03	2.57						
AMIS0124	0.84	0.735	0.945	0.87	0.78	0.96	0.16	0.13	0.19	1324	1165	1483	1917	1713	2121
AMIS0148	1.64	1.49	1.79	1.13	1.01	1.25	0.84	0.78	0.9	541	458.5	623.5	900	784.5	1015.5
AMIS0170	0.72	0.63	0.81	0.81	0.75	0.87	0.09	0.075	0.105	709	641.5	776.5	1071	940.5	1201.5
AMIS0208							1.38	1.23	1.53						
AMIS0277	1.34	1.25	1.43	1.47	1.29	1.65	0.2	0.17	0.23	1318	1231	1405	2305	1943.5	2666.5
AMIS0278	1.7	1.55	1.85	2.12	1.91	2.33	0.26	0.23	0.29	1294	1174	1414	2026	1672	2380
AMIS0302							4.47	3.96	4.98						
AMIS0325	2.06	1.79	2.33	2.25	1.98	2.52	0.3	0.24	0.36	2426	2159	2693	4091	3666.5	4515.5
AMIS0326	1.05	0.93	1.17	1.25	1.13	1.37	0.17	0.14	0.2	1403	1269.5	1536.5	2446	2297.5	2594.5
AMIS0395	0.51	0.45	0.57	0.62	0.53	0.71	0.095	0.074	0.116	847	781	913	1606	1364.5	1847.5
AMIS0396	0.75	0.66	0.84	0.93	0.84	1.02	0.105	0.081	0.129	969	888	1050	1840	1604.5	2075.5
AMIS0442	2.11	1.915	2.305	2.66	2.42	2.9	0.33	0.285	0.375	1029	961.5	1096.5	1996	1879	2113

Tolerance limits are set at three standard deviations from the certified mean value of the reference material:

- If two consecutive standards (CRM) in a batch are outside two standard deviations, the batch will be submitted for re-assay from the CRM prior to the failed CRM until the CRM after the second failed CRM.
- If a single CRM is outside of three standard deviations, the samples will be re-assayed including those samples from the CRM prior to the failed CRM until the next CRM thereafter.

A failed standard is considered cause for re-assay if it falls within a determined economic mining cut for either T- or F-Zone. Analysis of any fails is conducted and the appropriate action is undertaken. In general, the compliance with the certified values is good; however, there is some evidence of sample swapping, although this is minimal. This may be reduced or eradicated by a change in laboratory procedure regarding sample labelling and tray labelling.

11.5.1.2 Blanks

The insertion of blanks provides an important check on the laboratory practices, especially potential contamination or sample sequence miss-ordering. Blanks consist of a selection of Transvaal Quartzite and or Pool sand pieces (devoid of platinum, palladium, copper and nickel mineralization) of a mass similar to that of a normal core sample. AMIS0415 (Blank Silica Pulp) has also been introduced as an additional blank for insertion. The blank being used is always noted to track its behavior and trace metal content. The plotted graphs have a warning limit, which is equal to ten times the blank background.

In general, the failure rate is deemed not to have a material effect on the data, with more than 90% of the assays falling within acceptable limits.

11.5.1.3 Sample Duplicates

The purpose of having field duplicates is to provide a check on possible sample over-selection. The field duplicate contains all levels of error — core or reverse-circulation cutting splitting, sample size reduction in the preparation laboratory, sub-sampling at the pulp and analytical error. Field coarse duplicates were, however, not used on this project due to the assemblage of the core. Because of this problem, the laboratory was asked to regularly assay coarse reject samples as a duplicate sample to monitor analytical precision. Coarse reject duplicates were created by the laboratory by routinely making a sample from the coarse reject of every 20th sample, and assigning it the same sample number as its duplicate pair, with the addition of a prefix CRD.

The duplicate results graphs and calculations are available on request in digital format. The original analysis vs. the duplicate analysis showed minimal irregular values. This indicates minimal sample swapping.

In addition to the Coarse Laboratory Reject duplicates (CRD's), field pulp duplicates are selected at random, allocated a new sample number and re-submitted with a new sample number in a new batch to Set point. These show good correlation with the original samples with between 80% and 95% of the data falling within acceptable limits.

11.5.1.4 Laboratory Inserted Standards and blanks

All laboratories used in the Waterberg Project exercise quality control in the form of pulp duplicates, CRMs and blanks. These controls are included in each assay report.

11.5.1.5 Assay Validation

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

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

11.5.1.6 Re-assay

These samples are laboratory coarse duplicates. They are re-assayed on a random basis if there is bad correlation between the original assay and the routine laboratory duplicate. In general, re-assayed samples show good correlation with the original sample with greater than 90% of the data falling within acceptable limits.

11.5.1.7 Laboratory Duplicates

The laboratory (Set Point) regularly assays pulp duplicates with each batch of data. One in every 20 samples is randomly selected for both a coarse reject split and a pulp split. The original analysis vs. the duplicates analysis showed minimal irregular values. This indicates minimal sample miss-ordering or nugget effect.

11.5.1.8 Check Assays

At this time, the external umpire laboratory used to conduct check assays is Genalysis. Generally, batches are sent to Genalysis on a bi-annual basis. As a further check, 702 samples originally analyzed by Set Point were sent to Bureau Veritas (Rustenburg, South Africa) in November 2015. The majority of the samples are selected at random from within samples batches known to cover the economic intersections within drill holes. Umpire results from both Bureau Veritas and Genalysis confirm the satisfactory performance of the primary laboratory reporting results for the primary samples.

11.6 Databases

Databases in use at PTM currently include SABLE ™ , which is a SQL based relational database. This is a centrally managed database containing all aspects of drilling information including logging and assay results. In addition, PTM uses ARCVIEW, a GIS database system that is also SQL based for all spatial information relating to exploration activities. A number of other datasets exist including several Excel spreadsheets of information; however, these are derived from the SQL databases referenced above to ensure that all information is centrally updated and stored.

11.7 Sample Security

The QA/QC practice of PTM is a process beginning with the actual placement of the drill hole position (on the grid) and continuing through to the decision for the 3D economic intersection to be included in (passed into) the database. The values are also confirmed, as well as the correctness of correlation of reef/mining cut so that populations used in the geostatistical modelling are not mixed; this makes for a high degree of reliability in estimates of Mineral Resources/Mineral Reserves. In the opinion of CJM Consulting, the QAQC procedures as well as the sample preparation and security procedures are adequate to allow the data to be used with confidence in the Resource Estimation.

12. Data Verification

12.1 Verification off Data by QP

CJM Consulting as part of the Mineral Resource estimation for the Waterberg Project as detailed below conducted data verification: -

Printed logs for 90 percent of the holes were check with the drilled core. The depths of mineralization, sample numbers and widths and lithologies were confirmed. The full process from core logging to data capturing into the database were reviewed at the two exploration sites.

Collar positions of a few random selected drill holes were checked in the field and found to be correct.

With regard to missing specific gravity (SG) values, the average was generated for each individual lithological type, and the missing SG values inserted according to the lithological unit.

Assay certificates were checked on a test basis. The data was reviewed for statistical anomalies

12.2 Nature of the Limitations of Data Verification Process

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

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

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

The QPs opinion is that the data is adequate for use in Resource Estimation.

12.3 Possible Reasons for Not Having Completed a Data Verification Process

All PTM data was verified before being statistically processed. Copies of the QA/QC analysis can be provided on request.

12.4 Independent Audits and Reviews

Each Resource Estimation and Report to date has involved an independent Audit and Review of the Data and Procedures used by PTM. This includes site visits, verification of drill hole positions, logging verification, assay verification, visits and audits on laboratories used amongst other checks to ensure the accuracy of any Mineral Resource Statement.

13. Mineral Processing and Metallurgical Testing

13.1 Historic Test Work

As part of the pre-feasibility study (PFS), metallurgical characterization test work was conducted on samples from the Waterberg deposit to assess the metallurgical response and generate sufficient process design data to support the PFS study. The PFS test work program was conducted between August 2014 and September 2016 at MINTEK in Johannesburg, South Africa.

13.1.1 SGS 2103

Preliminary metallurgical test work was undertaken using samples taken from the Waterberg deposit as part of the Preliminary Economic Assessment (PEA) concluded in 2013. The results from the PEA test work program is summarized in the previous PEA technical report, filed in February 2014.

The following is a summary of the preliminary test results obtained:

- Metallurgical test work was conducted at SGS, in Booysens South Africa. Test work was conducted during the course of 2013, under the supervision of PTM.
- Test work was undertaken on two (2) composite samples from the Waterberg deposit, namely a T2-zone sample and an F-zone sample.
- The test work program included preliminary mineralogical characterization, in the form of petrographic and quantitative microscope test work, and bench scale flotation test work in the form of a single stage (MF1) cleaner flotation test on each of the two samples.
- The feed characterization showed that the T2-zone sample has a greater Pt, Pd, Au, Ni and Cu content than the F-zone sample. Quantitative mineralogy carried out on sample composites of F-zone and T2-zone, highlighted that T2-zone sample has better beneficiation properties, when compared to the F-zone sample. This due to the fact that there is a greater degree of liberation and particle size.
- Flotation test work on both samples confirmed the mineralogical observations. The T2-zone sample had a better rate of flotation and maximum recovery. However, the T2-zone sample contained clayish minerals and floatable gangue. It was noted that both samples appeared to be soft as they milled easily.
- The single MF1 cleaner flotation test on the F-zone sample achieved a final 2PGE + Au recovery of 76% at a grade of 18 g/t utilizing the 3.6 g/t 2PGE + Au sample; while the T2-zone sample achieved a 2PGE + Au recovery of 85.8% at a grade of 60 g/t utilizing the 6.7 g/t 2PGE + Au sample. It is noted that only a single MF1 cleaner test were performed on each sample, and no test work was done on grind or reagent optimization.

13.1.2 SGS 2013 — 2014

Following the scoping test work performed at SGS Booysens, South Africa, during 2013, further investigative test work was performed on an F-zone composite sample, under the management of JOGMEC during the course of 2013 to 2014.

The following is a summary of the results obtained:

- Various reagent schemes were tested utilizing a MF1 flotation flowsheet, of which the use of Oxalic acid as an activator, and Thiourea as a promotor achieved the best results.

- A 4E recovery of 84% was obtained in producing a 118 g/t 3E + Au product from a 3.52 g/t 3E + Au sample.
- 74% of the Cu was recovered, while 45% of the Ni was recovered.

13.2 Current Metallurgical Test work

The following section summarizes the metallurgical test work outcomes, as conducted by MINTEK under the management of PTM and DRA, between August 2014 and September 2016. The following test work campaigns were conducted:

- **Phase 1a:**

  The aim of Phase 1a campaign, conducted at MINTEK during August 2014 to November 2015, was to produce a typical concentrate product for preliminary discussions relating to third party smelting and precious metal refining, with particular reference to the likely sulphur and iron achieved in the final concentrate. The scope of work included bench scale flotation testing, final flotation concentrate mineralogical characterization, and investigations to reduce the iron content in the final concentrate by means of magnetic separation.

- **Phase 1b:**

  The Phase 1b campaign focused on determining the comminution parameters and the optimum flotation flowsheet for the F-Central, F-Boundary, and T-zone material. The scope of work included comminution test work, bench scale and locked cycle flotation testing, and mineralogical characterizations on various feed samples. Further to this, test work was also conducted to determine the tailings dewatering parameters and to investigate the possibility of including a pre-concentration stage by means of Heavy Liquid Separation (HLS) testing.

- **Phase 2:**

  Phase 2 test work included a number of bench scale collector optimization tests on the MF1 flowsheet.

- **Phase 3:**

  The Phase 3 campaign included comminution and flotation characterization test work of the F-North material from the Early Dawn farm area. Mineralogy was also conducted on flotation feed sample to support the characterization.

- **Phase 4:**

  Phase 4a and Phase 4b involved further grind and reagent optimization test work on the T-zone and F-Boundary material.

13.2.1 Sample Selection, Preparation and Characterization

Master composites were used in the PFS test work program to investigate the metallurgical response of the various ore bodies. The master composites involved the combination of several drill core intervals to provide information relating to spatial distribution, depth and lithology response for the major ore classifications.

It is noted that any crushed sample was stored in an inert refrigerated environment ahead of mineralogical and flotation test work, to avoid oxidation of the material.

Refer to Figure 13-1 for an illustration of the location of each of the drill cores used as part of the metallurgical test work campaigns.

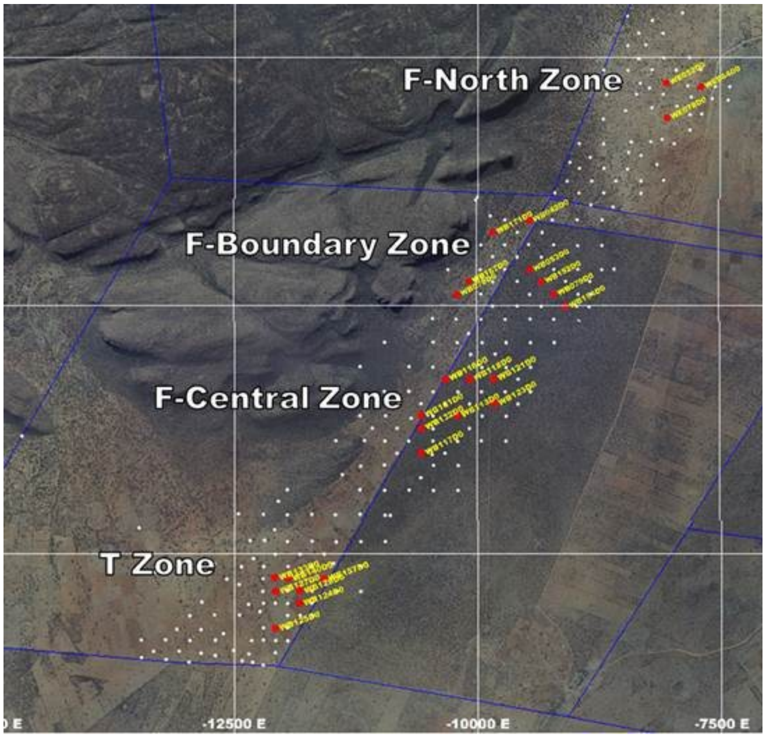


Figure 13-1: Metallurgical Sample Drill Core Location Map

13.2.1.1 F-Central Ore

Two (2) F-Central area metallurgical samples namely, Central FH-Upper (Sample F1) and Central FH-lower (Sample F2) were prepared during the Phase 1 test work campaign. A total of four (4) NQ full core drill core samples and four (4) NQ ¾ core drill core samples were delivered to MINTEK during October 2014, with depth intervals as per Table 13-1.

Table 13-1: F-Central Metallurgical Ore Sample

Sample	Borehole ID	Drill Core Size	Depth From (m)	Depth To (m)
Central FH-upper (Sample F1)	WB113-D2	¾ NQ	501.50	514.00
	WB116-D0	¾ NQ	750.50	767.61
	WB117-D1	¾ NQ	612.87	624.50
	WB118-D1	¾ NQ	567.00	586.62
	WB123-D1	Full NQ	370.00	403.95
	WB132-D1	Full NQ	773.00	796.50
	WB161-D1	Full NQ	771.00	792.00

Sample	Borehole ID	Drill Core Size	Depth From (m)	Depth To (m)
Central FH-lower (Sample F2)	WB113-D2	¾ NQ	561.50	596.00
	WB116-D0	¾ NQ	777.50	798.04
	WB117-D1	¾ NQ	626.50	642.70
	WB121-D1	Full NQ	474.00	482.73
	WB123-D1	Full NQ	418.50	450.00
	WB132-D1	Full NQ	803.00	813.20
	WB161-D1	Full NQ	797.69	818.08

The Central FH-upper and Central FH-lower comminution samples that were tested as part of the Phase 1b comminution test work were selected from the NQ (47.6mm diameter inside core) full drill core material; none of the ¾ NQ core material was included in the comminution samples due to top size requirements of the scoped test work. The remaining full core sample was crushed to less than 1.7mm and composited into individual F1a and F2a composite samples.

The first flotation composite sample for use during Phase 1a, sample F12a, was prepared by blending this F1a and F2a samples in a 50:50 ratio. No information was available at the time about the Mineral Reserve contribution, and thus this blend ratio was assumed.

The ¾ NQ drill cores were also crushed to less than 1.7mm after which it was added to the remaining F1 and F2 samples, described above. The resulting samples were then referred to as the F1 flotation master composite sample and the F2 flotation master composite sample, used during the Phase 1a and Phase 1b flotation test work campaigns. Both master composite samples were individually blended and split into 2kg representative sub-samples.

The second flotation composite sample was prepared for further use in the Phase 1a flotation testing and was referred to as the F12b sample. The F12b sample consisted of a 50:50 blend of the F1 flotation master composite sample (after the ¾ core material was added) and the F2 flotation master composite sample (after the ¾ core material was added).

A third composite sample was prepared for use in the Phase 1b flotation test work and was referred to as the F4 sample. The F4 sample consisted of a 60:40 blend of the F1 flotation master composite (F-Central FH Upper) and the F2 flotation master composite samples (after the ¾ core material was added). The F4 blend ratio was selected based on the expected contributions of F-Central FH upper and F-Central FH lower material within the F-Central area, at the time of sample preparation.

The head grades for each of the F-Central samples used during the different test work phases are presented in Table 13-2 below:

Table 13-2: F-Central Ore Samples Head Grades

Sample ID	Pt g/t	Pd g/t	Au g/t	2E + Au g/t	Cu %	Ni %	Fe %	SiO_2 %	MgO %	S %
F12a composite	0.67	1.87	0.23	2.76	0.09	0.14	8.93	47.28	21.72	0.24
F12b composite	0.96	2.07	0.13	3.15	0.06	0.19	8.12	48.19	23.01	0.21
F4 composite average	0.88	1.91	0.16	2.95	0.08	0.21	8.36	43.98	23.51	0.32

13.2.1.2 F-Boundary Ore

Two separate sets of sample were used for the comminution and flotation testing of the F-Boundary Ore [1]. The comminution sample was delivered to MINTEK during April 2015, in the form of four (4) BQ (36.5mm diameter inside core) half core samples, with depth intervals, as per Table 13-3. The comminution sample was selected from these drill cores based on sample size requirements.

Table 13-3: F-Boundary Ore Comminution Sample

Borehole ID	Depth From (m)	Depth To (m)
WB042-D2	792	829
WB053-D1	797	825
WB078-D1	1026	1038
WB079-D1	604	620

The flotation test work sample was delivered to MINTEK during January 2015 in the form of six (6) crushed BQ drill core samples, with depth intervals, as per Table 13-4. Each of these drill cores have been crushed and assayed by a third party laboratory, prior to being delivered to MINTEK.

The core samples used to assess the impact of varying head grade on the flotation response during the Phase 1b campaign is outlined in Table 13-4below.

Table 13-4: F-Boundary Ore Flotation Sample

Borehole ID	Flotation Sample Reference	Depth From (m)	Depth To (m)
WB152-D1	VT 11	690.00	699.00
WB152-D2	Sample stored for future use	684.00	693.50
WB154-D2	VT 12	377.50	387.50
	Sample stored for future use	387.50	406.50
WB154-D1	Sample stored for future use	377.00	396.50
WB167-D1	Sample stored for future use	1088.00	1117.50
	VT 10	1117.50	1118.00
WB171-D1	Sample stored for future use	986.50	1008.00

A Mineralogical investigation as well as further flotation test work was conducted as part of Phase 4b, which required a master composite sample (F-Boundary composite) to be prepared. This sample was prepared by blending material from the VT10 and VT12 samples in a 50:50 ratio.

The head grades for each of the samples tested are summarized in Table 13-5.

[1] F-Boundary ore is referred to as F-North ore in earlier test work report references.

Table 13-5: F-Boundary Ore Samples Head Assays

Sample ID	Pt g/t	Pd g/t	Au g/t	Rh g/t	4E g/t	Cu %	Ni %	Fe %	SiO_2 %	MgO %	S %
VT 10	0.80	1.44	0.15	0.07	2.45	0.09	0.19	8.34	48.00	20.50	0.46
VT 11	1.04	2.61	0.19	0.07	3.90	0.13	0.29	9.37	43.95	20.75	0.89
VT 12	1.50	3.19	0.22	0.08	4.99	0.14	0.24	6.66	44.70	16.20	0.61
F-Boundary composite	0.98	2.37	0.23	NR	3.58[2]	0.15	0.28	7.74	50.72	19.55	0.55

13.2.1.3 T-Zone Ore

Three (3) T-zone metallurgical samples were delivered to MINTEK for metallurgical testing. The first sample, T2a, was delivered to MINTEK during October 2014, in the form of six (6) NQ full core samples, with depth intervals, as per Table 13-6 below.

Table 13-6: T2a T-zone Metallurgical Sample

Borehole ID	Depth From (m)	Depth To (m)
WB124-D4	210.98	220.00
WB125-D2	197.25	203.00
WB126-D2	236.00	245.00
WB127-D2	381.68	406.50
WB133-D1	444.00	473.10
WB140-D2	365.00	373.50

The comminution sample for the Phase 1b test work was selected from the T2a sample drill cores based on sample mass and size requirements. After the comminution sample selection was completed, the remaining sample from the above NQ full drill cores were crushed to less than 1.7mm, blended and split into representative 2kg sub-samples.

A limited amount of flotation test work has been conducted using the T2a sample due to the low grade of the sample.

The second batch of T-zone sample (T2b) was delivered to MINTEK during January 2015, in the form of two (2) crushed BQ drill cores, as per Table 13-7.

Table 13-7: T2b T-zone Metallurgical Sample

Borehole ID	Depth From (m)	Depth To (m)
WB133-D2	446.50	455.00
WB157-D1	148.00	152.00

The above T2 core samples were used during Phase 1b to assess the impact of varying head grade on the flotation response. The sample details for these tests are summarized in Table 13-8.

[2] 2E + Au

Table 13-8: T-zone ore (T2b) Flotation Sample

Borehole ID	Flotation Sample Reference	Depth From (m)	Depth To (m)
WB133-D2	VT 8	446.50	455.00
WB157-D1	VT 7	148.00	149.50
	VT 9	149.50	152.00

A third T-zone sample (T2c) was delivered to MINTEK during January 2016, in the form of four (4) half core BQ drill cores and three (3) ¾ core NQ drill cores, as per Table 13-9.

Table 13-9: T2c T-zone Metallurgical Sample

Borehole ID	Depth From (m)	Depth To (m)
WB003-D3	644.00	652.00
WB010-D4	228.75	234.75
WB012-D2	323.50	329.50
WB013-D2	143.84	149.50
WB179-D1	632.11	638.50
WB180-D1	646.42	649.50
WB193-D2	656.59	662.50

The T2c composite sample was used for Phase 4a flotation test work and was prepared by compositing the above drill core material while targeting a final sample grade of ~4 g/t 2E + Au, based on the expected T-zone mining head grade at the time of sample preparation. The following material was excluded from the T2c composite sample:

- Material from WB193-D2 was excluded due to high grade of 8.4 g/t 2E + Au.
- 2kg each of WB010-D4 and WB180-D1 was stored for future use.

A fourth composite sample, T2d, was prepared for use in Phase 4b flotation test work, and was prepared by compositing the following stored samples, while targeting a final sample grade of ~4 g/t 2E + Au:

- WB193-D2,
- WB180-D1, and
- T2a composite sample.

The head grades for each of the T-zone samples tested are summarized in Table 13-10.

Table 13-10: T-zone Ore Samples Head Grade

Sample ID	Pt g/t	Pd g/t	Au g/t	2E + Au g/t	Cu %	Ni %	Fe %	SiO_2 %	MgO %	S %
T2a composite	0.53	0.63	0.41	1.58	0.12	0.07	6.31	55.72	9.71	0.24
VT 7	1.34	2.20	0.82	4.36	0.46	0.22	10.90	42.00	17.50	0.97
VT 8	0.86	0.87	1.37	3.10	0.27	0.15	8.47	47.00	10.55	0.68
VT 9	0.83	1.47	0.54	2.84	0.30	0.20	12.40	40.80	17.40	0.60
T2c composite	1.39	2.10	1.09	4.57	0.23	0.13	6.08	48.86	7.99	0.53
T2d composite	1.12	2.12	0.78	4.02	0.15	0.11	6.34	47.50	9.08	0.41

13.2.1.4 F-North Ore

The F-North ore sample for the Phase 3 test work campaign was delivered to Mintek during January 2016, in the form of three (3) NQ ¾ core samples, with depth intervals, as per Table 13-11.

Table 13-11: F-North Metallurgical Sample

Borehole ID	Depth From (m)	Depth To (m)
WE052-D2	465.94	557.23
WE064-D1	283.77	308.20
WE078-D1	283.39	300.66

The comminution sample was prepared by selecting material from the above drill cores based on sample requirements, after which the remaining sample were crushed to less than 1.7mm. A 100kg F-North master composite sample was prepared for flotation test work by applying the same weight ratio as the delivered core masses. The material was thoroughly blended and split into representative 2kg sub-samples prior to storage. The head grade detail for this sample is shown in Table 13-12.

Table 13-12: F-North Ore Samples Head Grade

Sample ID	Pt g/t	Pd g/t	Au g/t	2E + Au g/t	Cu %	Ni %	Fe %	SiO_2 %	MgO %	S %
F-North composite average	0.93	2.41	0.18	3.51	0.11	0.24	8.52	47.05	20.25	0.50

13.2.1.5 Mine Blend Sample

Three different mine blend samples were prepared during the course of the various campaigns.

The first mine blend sample (Mine Blend a) was prepared for flotation test work as part of the Phase 1b campaign. The sample composition was based on the expected life-of-mine contributions from each ore type, at the time of the sample preparation. The expected life-of-mine blend consisted of 25% T-zone, 25% F-Boundary, 25% F-Central FH upper, and 25% F-Central FH lower (by mass). The sample was prepared by compositing equal masses of the following samples: F-Central FH Upper (F1a sample), F-Central FH Lower (F2a sample), T-zone (VT 8), and F-Boundary (VT 11).

The second mine blend sample (Mine Blend b) was prepared for flotation test work as part of the Phase 1b and Phase 2 campaigns. This sample composition was based on the same ore contributions as the "Mine Blend a" sample, i.e. 25% T, 25% F-Boundary, 25% F-Central FH upper, and 25% F-Central FH lower (by mass). The "Mine Blend b" sample was prepared by compositing equal masses of the following samples: F-Central FH Upper (F1a sample), F-Central FH Lower (F2a sample), T-zone (VT 8), and F-Boundary (VT 12).

A third mine blend sample, Mine Blend c, was tested during the course of the Phase 4 flotation test work campaign. This sample consisted of 50% F-Central F4 composite and 50% T2c composite material.

The head assay detail for each of the mine blend samples is shown in Table 13-13.

Table 13-13: Mine Blend Samples Head Assays

Sample ID	Pt g/t	Pd g/t	Au g/t	2E + Au g/t	Cu %	Ni %	Fe %	SiO_2 %	MgO %	S %
Mine blend a	0.89	1.96	0.47	3.31	0.15	0.21	8.49	44.15	18.50	0.55
Mine blend b	0.96	1.78	0.62	3.36	0.10	0.21	8.58	46.21	18.74	0.41
Mine blend c	0.96	2.02	0.45	3.43	0.11	0.16	7.43	45.70	16.31	0.37

13.2.2 Mineralogy

13.2.2.1 Flotation Feed Samples Mineralogical Analyses

Mineralogical analyses was undertaken by MINTEK using automated SEM (autoSEM) on four (4) Waterberg samples namely T2a (T), F4 (F-Central), F-Boundary master composite, and F-North master composite. Each of these samples was analyzed after being milled to a particle size of ~80% -75µm. The T2d sample was milled to 83% -75µm to investigate the benefit of a finer grind.

Particular interest was paid to the Platinum Group Minerals (PGMs) and Base Metal Sulphides (BMS) in each of the samples. The aim of the investigation was to describe and understand the mineralogy of the Platinum Group Metals (PGMs), Base Metal Sulfides (BMS) and associated gangue within each sample. The following analysis was conducted on each sample:

- X-Ray diffraction (XRD).
- Quantitative Evaluation Of Minerals By Scanning Electron Microscopy (QEMSCAN) Modal Analysis
- A QEMSCAN PGM search to investigate the PGM mineralogy with specific focus on the associations with gangue; grain size distribution and liberation characteristics.
- A BMS search to determine the grain size distribution, liberation, mineral associations, as well as the modal abundance of all the minerals present in each sample.

13.2.2.1.1 QEMSCAN Modal Analysis Summary

Modal proportions of minerals present in each sample, as determined by QEMSCAN analyses, are presented in Table 13-14.

Major gangue minerals present in the samples are pyroxene, plagioclase, talc, serpentine and chlorite. The alteration silicates (talc, serpentine and chlorite) are more abundant in the F-Central F4 sample.

Table 13-14: Waterberg QEMSCAN Modal Analyses Summary

	T-zone		F-Central	F-Boundary	F-North
Mineral	**T2a composite**	**T2d composite**	**F4 master composite**	**F-Boundary composite**	**F-North composite**
Pentlandite	0.11		0.44	0.71	0.81
Chalcopyrite	0.33		0.24	0.46	0.56
Pyrrhotite	0.10		0.52	0.45	0.74
Pyrite	0.20		0.14	0.25	0.03
Other Sulphides	0.03		0.04	0.03	
Pyroxene	32.92	11.75	30.40	37.70	56.13
Plagioclase	46.58	50.07	16.40	25.25	17.74
Amphibole	2.19	4.16	1.48	0.98	12.72
Olivine	0.22		3.99	2.16	
Talc	3.30	2.91	10.34	6.36	0.86
Serpentine	1.03		17.08	7.96	0.03
Chlorite	4.54	17.15	14.24	11.44	4.51
Mica	1.89		0.87	1.02	1.20
Quartz	3.05	7.69	0.23	1.71	0.87
Chromite	0.05		0.21	0.19	0.06
Fe/Ti Oxides	0.39		1.41	0.75	
Other Oxides	0.12		0.05	0.04	
Calcite	1.66	4.23	1.07	1.48	1.46
Dolomite		2.04			
Apatite	0.21		0.13	0.08	0.02
Other Silicates	0.83		0.39	0.81	
Magnetite					2.13
Other	0.25		0.34	0.18	0.13

13.2.2.1.2 QEMSCAN PGM Search and Analysis

13.2.2.1.2.1 PGM Types

Several PGM-bearing particles were detected in each of the samples. Mineral identification, grain size, liberation and mode of occurrence data were gathered from each PGM-bearing particle detected. Laurite (RuS_2), although detected during the analyses, has been omitted from these results, since it is a low value PGM that does not contribute to 4E PGE assays.

It was noted that the PGE-bismuth tellurides dominated the F-zone ore types and the T2d T-zone sample, while the PGE-tellurides dominated the T2a T-zone sample. Minor to trace amounts of PGE-sulphides were detected in the F-zone ore types while no PGE-sulphides were detected in the T2a T-zone sample

13.2.2.1.2.2 PGM Grain Size Distribution

Grain sizes are expressed in equivalent circle diameter (ECD), which is defined as the diameter of a circle with the same area as the measured PGM grain.

When considering the PGM grain size distribution data, as presented in Table 13-15, it is noted that except for one relatively large PGM in the F-Central F4 sample, all the PGMs detected in the various samples are smaller than 24 µm ECD.

Table 13-15: Waterberg PGM Grain Size Distribution

Class	T		F-Central	F-Boundary	F-North
(ECD, µm)	T2a	T2d	F4	Composite	Composite
			PGM Volume %		
0 – 3	12.5	8.0	11.7	6.3	7.7
3 - 6	41.1	31.9	26.4	51.3	27.1
6 - 9	31.4	26.0	11.7	19.8	15.4
9 - 12	5.6	10.1	12.5	22.6	12.2
12 - 15	9.5	8.5	17.5		8.4
15 - 18	—	—	—	—	29.2
18 - 21	—	15.6	—	—	—
21 - 24	—	—	20.3	—	—
24 - 30	—	—	—	—	—
>30	—	—	—	—	—

Due to the small amount of PGMs detected, the single large PGM in the F4 sample has the effect of skewing the size data for this sample, and this should be taken into account when interpreting the size distribution results.

13.2.2.1.2.3 PGM Mode of Occurrence

Refer to Table 13-16 for a summary of the PGM mode of occurrence per sample.

Table 13-16: Waterberg PGM Mode of Occurrence

	T-zone		F-Central	F-Boundary	F-North
PGM grain Mode of occurrence	T2a	T2d	F4 master composite	F-Boundary composite	F-North composite
			Volume %		
Liberated	56	68	49	48	40
Associated with liberated BMS	2	0	21	3	3
Attached to Silicate or Oxide gangue particles	30	22	18	32	0
Associated with BMS attached to Silicate or Oxide gangue particles	5	5	9	5	20

PGM grain Mode of occurrence	T-zone T2a	T2d	F-Central F4 master composite	F-Boundary F-Boundary composite	F-North F-North composite
			Volume %		
Associated with BMS locked in Silicate or Oxide gangue particles	0	0	0	0	23
Locked within Silicate or Oxide gangue particles	8	5	3	12	13

The liberated PGMs and PGMs associated with liberated BMS is typically recoverable, whilst the PGMs associated with BMS locked in silicates or gangue together with the PGMs locked within silicate or gangue particles would be lost to tailings. The PGMs attached to silicates and oxide gangue or these associated with BMS attached to silicate or gangue occur in composite particles, and may be recoverable if the floatable components have enough surface exposure to adhere to the froth during flotation. The F-North sample had significantly more PGM association with complex particles compared to the other samples.

13.2.2.1.2.4 PGM Liberating Index

Refer to Table 13-17 for a summary of the Waterberg ores PGM liberation index data.

PGM-bearing particles with liberation indices of >0.4 are likely to comprise the fast-floating fraction of each sample. Particles with liberation indices of <0.2 should be considered non-recoverable, or at best slow-floating, unless further milling is conducted.

Table 13-17: Waterberg PGM Liberation Index

Liberation Index	T T2a	T2d	F-Central F4	F-Boundary Composite	F-North Composite
			Volume %		
<0.2	37.8	15.2	17.2	45.5	36.2
0.2 - 0.4	5.1	—	0.3	0.1	15.3
0.4 - 0.6	—	0.5	10.6	2.0	—
0.6 - 0.8	—	16.1	2.0	1.1	2.1
0.8 - 1.0	57.2	68.2	69.9	51.4	46.3

The F4 and T2d samples showed the highest degree of liberated PGMs and should in turn provide higher PGM recoveries than the other samples at the sample grinds. From the above, it appears that the F-Boundary sample contains the higher volume of non-floating PGMs as well as the higher volume of fast floating PGMs.

13.2.2.1.2.5 PGM Grain Floatability Index

Refer to Table 13-18 for a summary of the PGM grain floatability index data reported on each of the samples analyzed.

Table 13-18: Waterberg PGM Grain Floatability Index

Flotation Index Class	Particle Characteristics	T		F-Central	F-Boundary	F-North
		T2a	T2d	F4	Composite	Composite
		PGM Volume %				
Fast Floating	Liberated PGM's >3µm ECD	52.3	64.9	46.6	46.6	40.1
	Liberated BMS >10µm ECD	1.7	0.0	20.8	2.9	2.3
Slow Floating 1	Liberated PGM's <3µm ECD	3.2	3.3	2.5	1.8	0.0
	Liberated BMS <10µm ECD	0	0.0	0.0	0.2	0.9
	PGM's >3µm ECD attached to gangue	25.1	20.9	14.7	29.8	18.8
	BMS >10µm ECD attached to gangue	5.2	1.7	7.7	4.7	22.4
Slow Floating 2	PGM's <3µm ECD attached to gangue	4.5	0.8	3.3	2.1	0.8
	BMS <10µm ECD attached to gangue	0	2.8	1.4	0	1.1
No Floating	PGM's and/or BMS locked in gangue	8.1	5.6	3.0	12.1	13.6

It is noted that the F-Boundary and F-North samples reported the highest volume percentage of non-floating PGM grains, indicating that lower PGM recoveries can be expected from these samples. The F-Boundary sample reported the highest fraction of slow floating PGM grains, suggesting the need for longer residence times in the flotation circuit.

13.2.2.1.3 BMS Analyses

13.2.2.1.3.1 BMS Liberation Index

The BMS liberation index data is summarized in Table 13-2 below. Liberation classification is based on area percentage proportion of the mineral of interest (e.g. pentlandite) of the total area of a particle. Liberation classes are defined in 11 groups ranging from 0 to 100 area %. These indexes were used to assist in the flowsheet development and design.

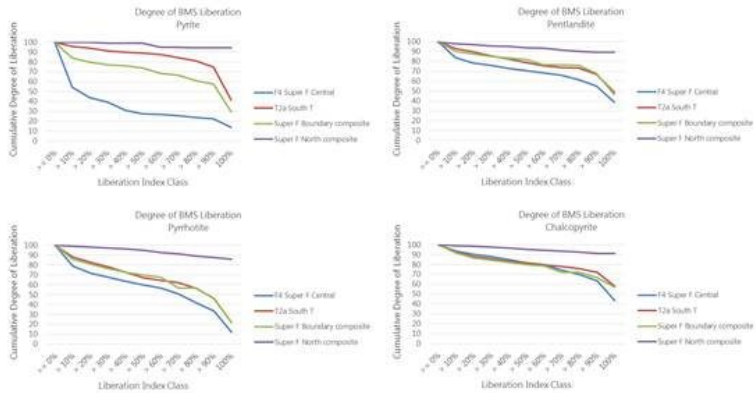


Figure 13-2: Waterberg BMS Liberation Index Summary

13.2.2.2 F-Central Concentrate Mineralogy

As part of the Phase 1a test work scope, a mineralogical investigation was performed on the two concentrate products, i.e. a high grade primary circuit product and a lower grade secondary circuit product, produced from the F-Central F12b master composite sample by utilizing a standard MF2 flowsheet.

The mineralogy search revealed that the PGMs in the primary circuit product were predominantly Pt/Pd-arsenides and Pd-bismuth tellurides, with minor Pt-sulphides. Similarly, in the secondary circuit product the PGMs were mainly Pt/Pd-arsenides and Pd-bismuth tellurides. The PGM mode of occurrence indicated that greater amounts of PGMs were attached to silicates in the secondary circuit product, resulting in lower product grade when targeting high PGM recovery. The modal and base metal search results indicated that both concentrate products comprised mostly of silicates minerals, with talc being the dominant species. The silicates content of the primary circuit concentrate was approximately 64% whilst silicates in the secondary circuit product were approximately 75%. Chalcopyrite was reported as four times higher in the primary circuit product compared to the secondary circuit product. Nickel and copper in the samples were hosted by pentlandite and chalcopyrite respectively. The dominant base metal sulphides were chalcopyrite and pentlandite in the primary and secondary circuit products respectively.

The full chemical analysis, by XRF, did not reveal any deleterious elements of concern in the F-Central concentrate product.

13.2.2.3 F-Central Nickel Deportment Study

A nickel deportment study on the F-Central F4 composite sample was conducted to determine the mode of occurrence of pentlandite in the sample and to quantify the portion of recoverable nickel in the sample. Nickel deportment calculations are used to determine the relative contribution of each nickel-bearing mineral to the total nickel content of the sample.

The Ni deportment results show that a maximum of ~ 70 % of the total Ni content is in a recoverable form (i.e. the pentlandite and Millerite), however, small grain sizes limits the practical recovery.

13.2.2.4 Scavenger Flotation Tailings Study

PGM searches were conducted on the MF2 circuit scavenger flotation tailings products of the T2d composite sample and the F-North composite sample. The following was noted:

- The F-North tailings sample reported are high amount of locked particles (56%) compared to the T2d T-zone sample (24%).
- When considering the class in which PGMs are attached to Silicate or Oxide gangue particles, there was very little volume reported in the F-North sample (13%), whilst the T2d sample reported a much higher volumetric contribution from this class (61%). Based on this, it is possible that the T-zone material is amenable to finer grinding. It does not appear that the F-North sample is amenable to finer grinding.
- The F-North tailings sample indicated that 18% (volume) of the PGMs present was liberated, however, the liberation data revealed that 72% of the PGMs detected had grain sizes < 6µm, and all of the PGMs grain sizes was smaller than 9µm. The T2d T-zone tailings sample indicated that 8% (volume) of the PGMs present was liberated, whilst the liberation data revealed that 74% of the PGMs detected had grain sizes < 6µm, and all of the PGMs grain sizes was smaller than 9µm. 42% of the PGMs detected had grain sizes < 3µm. It is noted that this results are based on volumetric contributions and that it can be misleading since there were only 2 PGM grains detected which were between 3µm to 10µm size fraction. This could make the data statistically unreliable.

13.2.3 Comminution Test work Summary

Comminution test work on each of the following samples was conducted at Mintek during the course of the different test work phases: T-zone T2a sample, F-Central F4 sample, F-Boundary cores, and F-North cores. Refer to Section 13.2.1 for details pertaining to the comminution sample selection process.

The comminution characterization test work scope included; SAG Mill Comminution (SMC) tests, uniaxial compressive strength (UCS) tests, Bond crushability work index (CWi) tests, Bond abrasion index (Ai) tests, Bond rod work index (BRWi) tests, Bond ball work index (BBWi) test and Mintek grind mill tests.

Due to the metallurgical drill core sample being available in different core sizes and fractions (i.e. half core, ¾ core, or full core), the samples were not all subjected to identical testing. As a minimum however, each sample was subjected to BBWi and Mintek grindmill testing. This allows comparison and benchmarking of the different sample against each other by means of various simulation methods.

Refer to Table 13-19 for a summary of the tests conducted per sampler as well as the associated results.

Table 13-19: Summary of Waterberg Samples Comminution Test Results

Waterberg Ore Type (Sample Reference)	SG t/m³	SMC A*b	UCS Min MPa	UCS Max MPa	UCS Avg. MPa	CWi Avg. kWh/t	Ai Avg. g	BRWi 1180µm kWh/t	BBWi 106µm kWh/t	BBWi 75µm kWh/t
T zone (T2a sample)	2.92	51.6	63.4	120.1	83	10.8	0.194	16.28	19.54	21.63
F-Central FH Upper (F1)	2.98	30.8	87.1	244.9	196	11.0	0.162	20.12	24.37	24.96
F-Central FH Lower (F2)	3.03	32.1	56.9	268.8	172.2	10.6	0.183	19.82	21.98	22.90
F-Boundary	2.96	—	—	—	—	—	0.200	19.75	22.67	24.13
F-North	—	—	—	—	—	—	—	—	20.24	20.03

The comminution test work results can be summarized as:

- The SMC test classified the T-zone sample as being of medium hard competency, whilst both the F-Central samples were classified as being of hard competency.
- The UCS test classified the T-zone sample as soft while the F-Central samples were classified as hard.
- The CWi test results classified the sample tested as soft.
- The Bond abrasion index test results indicated that all of the Waterberg samples tested were moderately abrasive.
- BRWi and BBWi test results classified all of the samples tested as hard to very hard.

13.2.4 Flotation Test Work Summary

Various test work campaigns were conducted during which the flotation response of the various Waterberg ore types were tested and compared. Two flotation flowsheets were tested during each of the different campaigns, i.e.:

- MF1 circuit utilizing Oxalic acid and Thiourea, and
- MF2 circuit utilizing typical Southern African PGM reagent suites.

The MF1 circuit applied by SGS, under the management of JOGMEC, during the course of the 2013/2014 campaign was used as a starting point for the subsequent development of the MF1 circuit flowsheet.

13.2.4.1 Phase 1a: Concentrate production for Product Off-take Discussions

A number of open circuit batch flotation tests were conducted on the F12a and F12b master composite samples (refer to Section 13.2.1.1). The results from these tests are summarized in Table 13-20.

The MF1 (single milling and flotation stage) circuit utilizing Oxalic acid and Thiourea generally resulted in higher iron and sulphur content in the final product, with similar final concentrate metal grades. The reagent suite results in higher metal recoveries when compared to the MF2 tests.

The combined concentrate product produced by the MF2 open circuit tests, namely MF2 T22 and MF2 T23, achieved a slightly higher final product grade (> 100 g/t 2E + Au), and was thus submitted for a comprehensive chemical and mineralogical characterization. The aim of the mineralogy investigation was to characterize the mode of occurrence of the PGMs, gangue and base metal sulphides (BMS) in the final concentrate product produced by the MF2 circuit. Refer to Section 13.2.2.2 for more information.

13.2.4.2 Phase 1b: F-Central Flotation Flowsheet Development Test work

Refer to Section 13.2.1.1 for details pertaining to the sample selection and preparation of the F-Central F4 master composite sample used in this phase of test work.

Head grade analysis, using a variety of analytical methods, resulted in notable assay variability despite a number of re-assay checks. This is most likely attributable to coarse nugget effects, mostly noted on the Au and Pd assays. The calculated head grades (based on concentrate and residue analysis) for each of the open circuit batch flotation tests performed during the campaign was included and compared to the assayed head grade. The head grade assays and calculated values for the F4 sample is presented in Table 13-21.

Table 13-20: Summary of Waterberg Phase 1a Campaign Test Results

Test ID Description	Sample ID	Calculated Head Grade 2E+Au g/t	Mass Pull %	Grade 2E+Au g/t	Grade Cu %	Grade Ni %	Grade Fe %	Grade S %	Recovery 2E+Au %	Recovery Cu %	Recovery Ni %	Recovery Fe %	Recovery S %
T1_Modified JOGMEC MF1 JOGMEC MF1 with reduced conditioning time and 3-Stage cleaning	F12a	2.66	1.63	125.4	3.76	5.05	19.68	11.83	77.07	82.87	46.90	3.69	61.59
T2_JOGMEC MF1 JOGMEC MF1 with 3-Stage cleaning	F12a	2.65	1.29	145.4	4.49	5.88	21.31	13.48	70.61	80.50	37.20	3.20	55.20
MF2 T6 repeat Split cleaning circuit with 3-stage secondary cleaning. Iron rejection	F12b	3.26	2.31	115.3	2.59	3.62	14.24	7.56	81.77	—[3]	38.50	3.93	65.95
MF2 T8 Split cleaning circuit with 3-stage secondary cleaning. Iron rejection	F12b	3.22	2.69	91.9	2.11	2.77	13.26	6.14	76.76	—[4]	35.53	4.11	52.16
T11_Modified JOGMEC MF1 JOGMEC MF1 with reduced conditioning time and 2- stage cleaning	F12b	3.13	2.29	109.7	2.41	3.95	16.88	9.74	80.15	73.84	38.01	4.61	65.93
T12;T15;T18;T21;T24_Modified JOGMEC MF1 JOGMEC MF1 with reduced conditioning time and 2- stage cleaning	F12b	3.24	2.70	97.2	2.39	3.23	15.45	8.17	81.02	86.92	38.12	5.04	65.23
T22; T23_MF2 Split cleaning circuit with 2-stage secondary cleaning. Iron rejection.	F12b	3.61	2.76	102.7	2.23	2.83	15.75	7.36	78.68	83.14	38.38	5.00	65.22

[3] Lower detection limit reached on assay instrument
[4] Lower detection limit reached on assay instrument

Table 13-21: F4 Flotation Sample Head Grade

Analytical Method Used	Pt g/t	Pd g/t	Au g/t	2E+Au g/t	S %	Cu %	Ni %	MgO %	Fe %	SiO_2 %
3E (Pt, Pd, Au) [5]	0.88	2.13	0.16	3.17	0.31	0.08	0.22	23.75	8.32	43.00
4E (Pt, Pd, Au, Rh) [6]	0.92	1.46	0.13	2.51						
6E (Pt, Pd, Au, Rh, Ir, Ru) [7]	0.76	1.40	0.10[8]	2.26						
Calculated from test work	0.85	2.02	0.17	3.05	0.33	0.09	0.22	23.26	8.40	44.96
F4 composite average [9]	0.88	1.91	0.16	2.95	0.32	0.08	0.21	23.51	8.36	43.98

This flotation test work campaign aimed to provide the necessary process design parameters needed to establish what the optimum flowsheet for the Waterberg project would be. During this campaign, test work was conducted concurrently on the MF1 circuit and a MF2 circuits.

13.2.4.2.1 MF2 Open Circuit Test work

The following bench scale test work was undertaken using an MF2 circuit:

- Primary rougher flotation development work including reagent screening, residence time determination, effect of grind and effect of depressant dosage.
- Primary cleaner circuit configuration.
- Secondary rougher circuit development work which aimed to determine the effect of particle grind, residence time, collector and depressant addition rates.
- Effect of mainstream regrinding, and application for ultrafine grinding.
- Secondary cleaner circuit configuration.

During the development of the main stream circuit (primary rougher and secondary rougher) test work revealed that the addition of 35 g/t Sodium Isobutyl Xanthate (SIBX) in the secondary circuit (Test M10), and a finer secondary grind of 90% -75µm (vs 80% -75µm) (Test M11) resulted in between 1% and 2% higher 2E+Au recovery. Even though a higher grade was obtained from the shorter primary rougher residence time (Test M9), a lower 2E+Au and Ni recovery was achieved in these tests, when compared to the other tests.

The 2E+Au and Ni grade-recovery curves for the tests performed during the primary and secondary rougher development work is presented in Figure 13-3 and Figure 13-4, respectively.

5 Fire Assay via Pb/Ag collection followed by the hot plate acid dissolution of the silver prill in an aqua-regia. Analysis of the Pt, Pd, and Au with ICP-OES.
6 Fire Assay via Pb/Ag collection followed by the high pressure oven acid dissolution of the silver prill in a sealed glass tube. Analysis of the Pt, Pd, Rh and Au with ICP-OES.
7 Fire Assay via NiS collection followed by acid leaching of the crushed Ni button on a boiling water bath, filtration and hot plate dissolution of the PGMs, followed by analysis of the PGEs with ICP-OES
8 Au assay may not be accurate due to poor collection of Au by the NiS method
9 6E values excluded

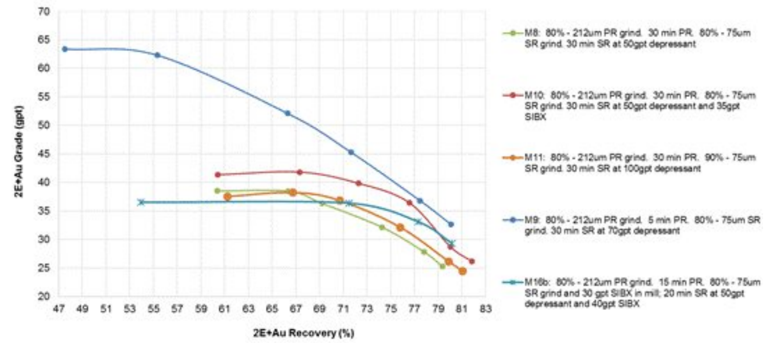


Figure 13-3: 2E+Au Grade-Recovery Curve (Main stream only)

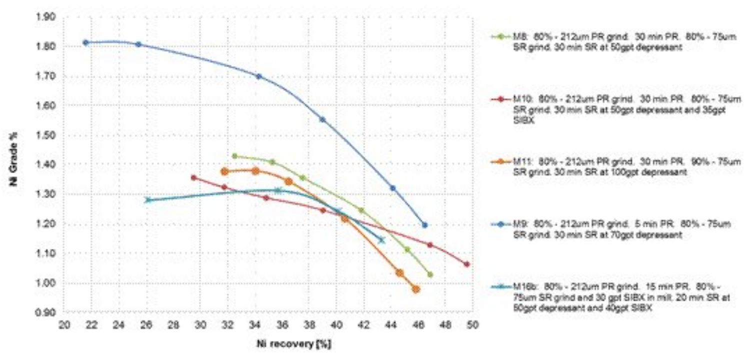


Figure 13-4: Ni Grade-Recovery Curve (Main stream only)

The final MF2 flowsheet tested as part of Phase 1b (Tests MF2 N5, MF2 N6 and MF2 N7) included the optimized MF2 circuit tested during the Phase 1a. Extended flotation times were tested in the secondary rougher, scavenger, and scavenger cleaner streams to target the slow floating fractions as observed in the PGM grain floatability data (Refer to Section 13.2.2.1.2.5). In an attempt to improve on final concentrate grade, without compromising on achievable metal recovery, the primary re-cleaner and secondary re-cleaner stages' residence times were reduced in test MF2 N7.

Four (4) different MF2 circuit options were evaluated as part of the Phase 1b campaign, namely:

- Complex MF2 circuit (Test M22_b as per Figure 13-5)
- Phase 1a MF2 with inclusion of a regrind stage (as per Figure 13-6)
- MF2 circuit with extended scavenging and scavenger cleaning capacity (Test MF2 N5 and Test MF2 N6 as per Figure 13-7)
- MF2 circuit with extended scavenging and scavenger cleaning capacity and reduced final cleaner residence times and mass pulls (Test MF2 N7 as per Figure 13-8)

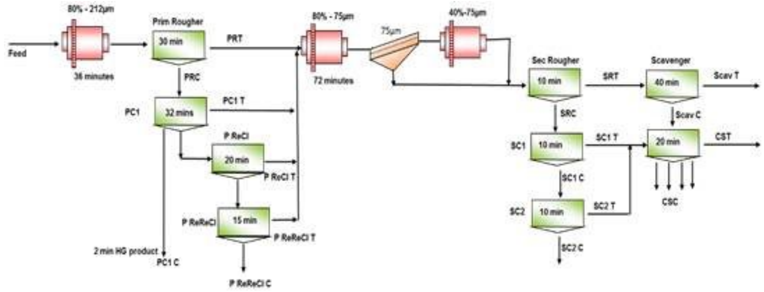


Figure 13-5: Complex MF2 Circuit (Test M22b)

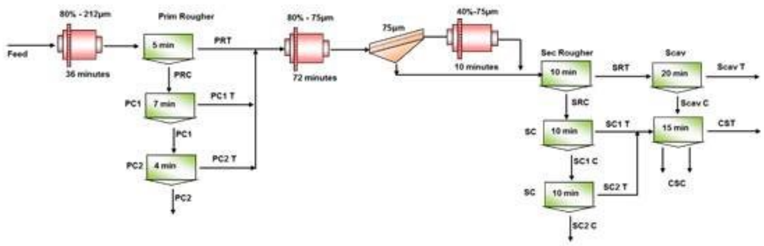


Figure 13-6: MF2 Campaign 1 with Regrind

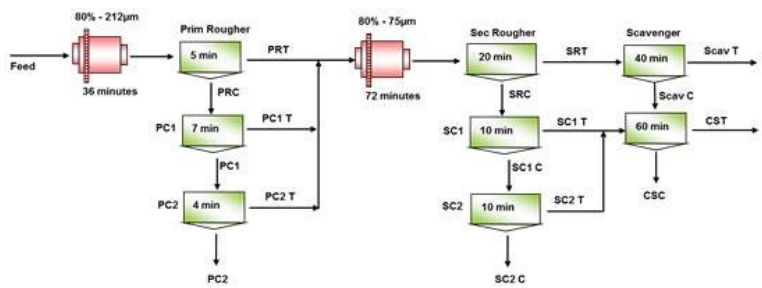


Figure 13-7: Test MF2 N5/N6 Flowsheet

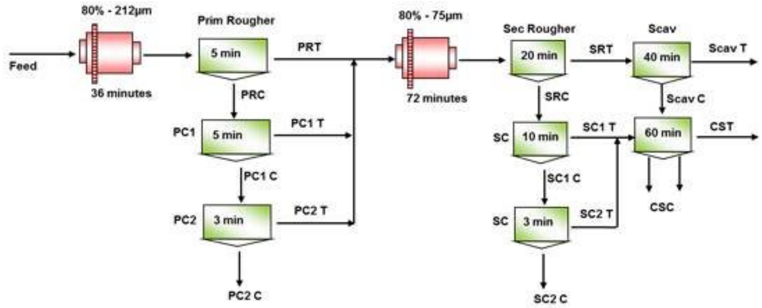


Figure 13-8: Test MF2 N7 Flowsheet

The grade-recovery relationship derived for each of the above MF2 configurations was compared against the base-case Phase 1a MF2 circuit results. The resultant grade-recovery (2E+Au) curves for each circuit is presented in Figure 13-9.

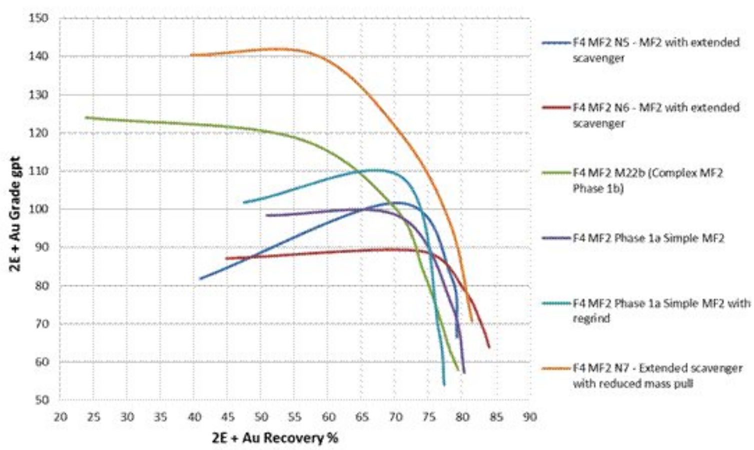


Figure 13-9: 2E+Au Grade-Recovery Curve (MF2 comparison)

The Phase 1b MF2 tests revealed that that extensive scavenger and cleaner circuit capacity is essential, while low primary re-cleaner and secondary re-cleaner mass pulls are to be targeted (as per F4 MF2 N7 test) in order to maximize the final product grade. This circuit did however result in lower Ni recovery (35% vs 38.5%), compared to the F4 MF2 N5 and F4 MF2 N6 tests in which the cleaner residence times where longer. When comparing Cu recoveries across the different flowsheets tested, the F4 MF2 N7 test achieved the highest Cu recovery of ~80%.

Based on the higher final concentrate grade and PGE recovery achieved, the flowsheet as per Figure 13-8 (F4 MF2 N7) was selected as the optimum MF2 flowsheet.

13.2.4.2.2 MF1 Open Circuit Test work

As part of the Phase 1b campaign, a number of open circuit batch tests were performed in order to evaluate the effect of minor modifications to the base case MF1 circuit as tested during the Phase 1a campaign (as per Figure 13-10). These included:

- The use of an alternative xanthate collector (SIPX or Sodium Isopropyl Xanthate as opposed to SIBX or Sodium Isobutyl Xanthate), called Tests M13a and M13c.
- In-mill conditioning versus cleaner circuit conditioning of Oxalic acid and Thiourea, Test M14a and M14a repeat, as per Figure 13-11.
- The effect of Oxalic acid and Thiourea addition to the circuit — compared to no addition of these reagents, Test M15.
- The effect of a finer primary grind (90% passing 75µm when compared to the base-case of 80% passing 75µm), Test M18.
- The effect of a stirred milling regrind circuit on the slow floating streams prior to scavenger cleaning, Test M19 as per Figure 13-12.

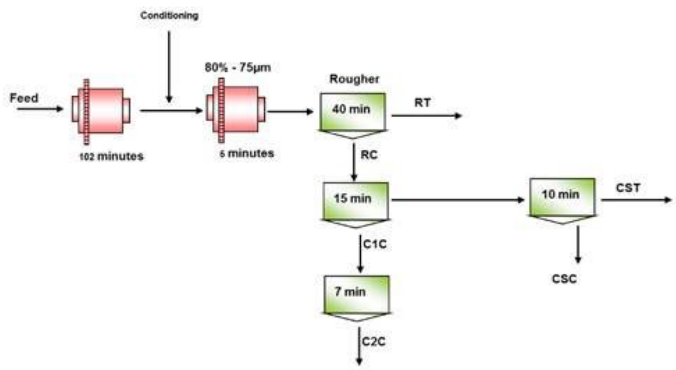


Figure 13-10: MF1 Base Case Flowsheet, M13a

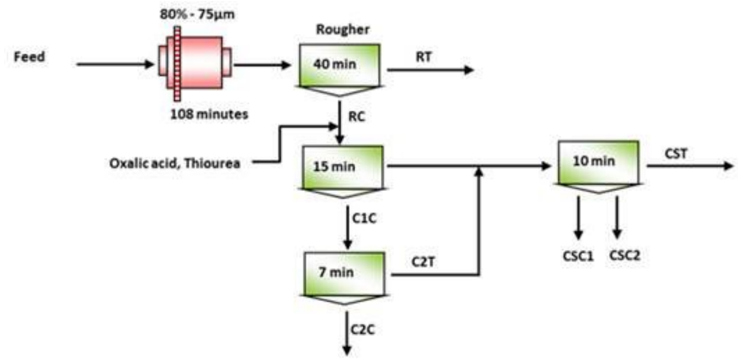


Figure 13-11: MF1 Test M14a Flowsheet

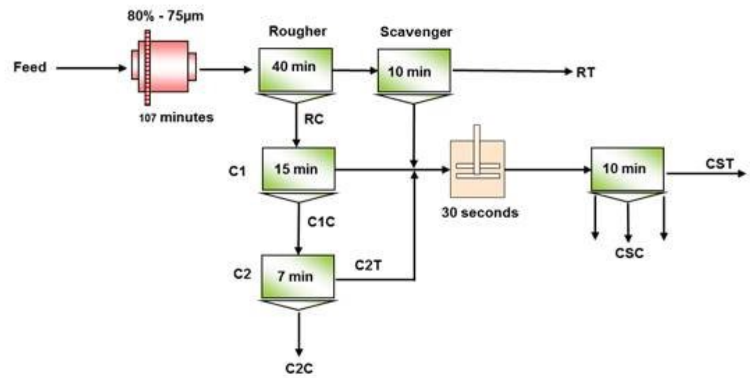


Figure 13-12: MF1 Test M19 Flowsheet

Based on the test work results the following observations and conclusions could be made:

- The alternative collector (SIPX) improved both the PGE and nickel flotation recoveries at similar PGE grades, although it also resulted in significantly higher iron content in the final product. Depending on the iron mode of occurrence (i.e. oxide vs sulphide), this could result in increased furnace and converter matte fall during smelter and subsequently affect the third party treatment costs. No mineralogy assessment on this specific concentrate sample was conducted, however, when considering the mineralogy results as summarized in Section 13.2.2.2, the majority of the iron in that specific MF2 final concentrate product was present as a sulfide (pyrrhotite).

- Conditioning with Oxalic acid and Thiourea in the milling circuit is not essential. Comparative results were obtained with conditioning prior to the cleaner flotation steps, resulting in reagent consumption reduction and reduced operational complexity. Improved nickel recoveries were also noted by conditioning prior to the cleaner circuit.

- The addition of Oxalic acid and Thiourea results in an increase in PGE recovery and grade, when compared to the test in which these reagents were excluded from the flowsheet. Improved nickel recoveries where however reported for the test in which the Oxalic acid and Thiourea were excluded.

- Regrinding of the slow floating fraction prior to scavenger cleaning did not improve the results obtained in the base-case flowsheet and resulted in both lower metal recoveries and lower concentrate grades.

A summary of the results of these tests are presented in Figure 13-13 in the form of a grade-recovery curves for 2E+Au.

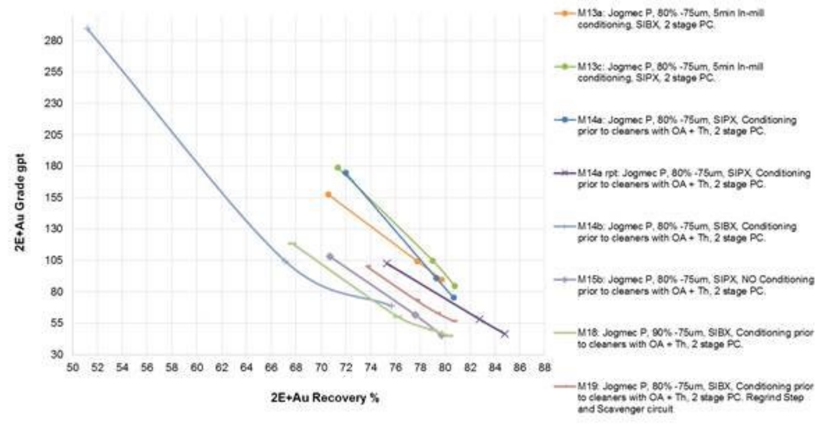


Figure 13-13: 2E+Au Grade-Recovery Curve (Phase 1b MF1 Investigation)

13.2.4.2.3 F-Central MF1 vs MF2

When comparing the results achieved in the MF2 open circuit tests to the MF1 open circuit tests, it was noted that on the F-Central F4 composite sample tested, the PGE flotation performance were similar between the two circuits. Refer to Figure 13-14 for a comparison of the 2E + Au Upgrade Ratio (UGR) vs 2E + Au recovery for both circuits. The MF1 circuit achieved the higher nickel recovery (42% vs 38%), whilst the MF2 circuit achieved the higher copper recovery (~80% vs ~66%).

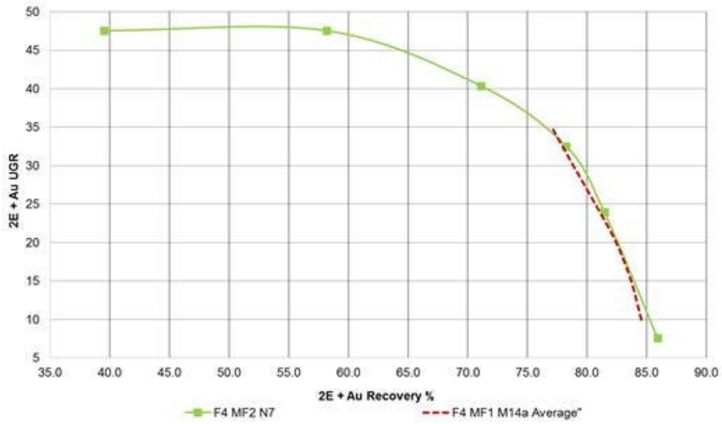


Figure 13-14: 2E+Au UGR-Recovery Curve (F-Central MF1 vs MF2)

13.2.4.3 Phase 2: F-Central MF1 Circuit Collector Optimization

The Phase 2 test work campaign focused on determining the metallurgical response benefit of various reagent collector schemes using the MF1 flowsheet. The aim was to improve the recovery of both the precious metals and nickel, without compromising on the final concentrate grade. The test work was conducted using the F-Central F4 master composite sample, as per Section 13.2.1.1

The MF1 flowsheet, as per Phase 1b Test M14a (see Figure 13-11), was used as the base case flowsheet to measure the modifications against.

The following conditions were tested:

- Impact of dosing Oxalic acid and Thiourea in the rougher circuit as opposed to the cleaner circuit.
- Effect of copper sulphate addition, as an activator (addition in the rougher circuit) and/or as a froth modifier (addition in the cleaner circuit).
- Effect of dosing the following collectors in the rougher circuit:
  - Gamcol 2015 (mixed collector),
  - C12 Mercaptin, and
  - MBT (more selective collector).

The following conclusions were drawn from the Phase 2 test work campaign:

- There is no support for the use of Oxalic acid and Thiourea in the rougher stage. The effect of dosing different collectors to the rougher circuit did not improve the recovery of nickel, when compared to the baseline test. The result is supported by the mineralogical characterization work, which indicated that the pentlandite is locked in fine gangue minerals.
- The addition of $CuSO_4$ to the rougher circuit resulted in ~1% higher PGE recovery.

13.2.4.4 Phase 3: F-North Flotation Test work

As per the Phase 1b campaign, the calculated head grades from the individual tests (based on concentrate and residue analysis) was included and compared to the assayed head grade. The head grade assays and calculated values for the F-North sample is presented in Table 13-22.

Table 13-22: F-North Composite Sample Head Grade

Analytical Method Used	Pt g/t	Pd g/t	Au g/t	2E+ Au g/t	S %	Cu %	Ni %	MgO %	Fe %	SiO_2 %
3E (Pt, Pd, Au) [10]	0.93	2.40	0.18	3.50	0.49	0.09	0.22	20.63	8.41	47.58
Calculated from test work	0.92	2.42	0.18	3.52	0.51	0.12	0.26	19.86	8.63	46.53
F-North composite average	0.93	2.41	0.18	3.51	0.50	0.11	0.24	20.25	8.52	47.05

13.2.4.4.1 MF2 Flotation Test work

The optimal circuit configuration (F4 MF2 N7, Figure 13-8) from the Phase 1b test work campaign was used as the baseline flowsheet for the Phase 3 flotation test campaign (PH3 EDF MF2 T1). The baseline test results indicated that similar 2E+Au rougher recoveries (approximately 86%) could be expected for both the F-North composite sample and the F-Central F4 sample. The tests did, however, highlight that significantly lower upgrade ratios (2E+Au UGR of 8 at 86% recovery) could be expected for the F-North ore.

Following the baseline test, a number of optimization tests were conducted as described in Table 13-23, to improve the upgrade potential of the material.

Table 13-23: Phase 3 MF2 Flotation Optimisation Tests

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH3 EDF [11] MF2 T1	Phase 3 MF2 baseline test applying the Phase 1 MF2 New T7 flowsheet and test conditions
PH3 EDF MF2 T2	Baseline test conditions with a reduction in depressant to the primary cleaner circuit
PH3 EDF MF2 T3	Baseline test conditions with a reduction in depressant to the primary re-cleaner circuit
PH3 EDF MF2 T5	Baseline test conditions in primary circuit with increased depressant in primary re-cleaner. Increased secondary rougher residence time and SIBX addition to the secondary rougher stage. Scavenger cleaning stage omitted.
PH3 EDF MF2 T7	Repeat of T5 with the addition of a scavenger cleaning stage.

[10] Fire Assay via Pb/Ag collection followed by the hot plate acid dissolution of the silver prill in an aqua-regia. Analysis of the Pt, Pd, and Au with ICP-OES.
[11] EDF refers to the F-North composite sample

Test 7 delivered the best result by achieving a high grade final product of 133 g/t (2E+Au) at 71% recovery and 1.8% mass pull, or a lower grade 53 g/t (2E+Au) product at 81% recovery and 5.2% mass pull. The copper and nickel recoveries were 88% and 54% respectively for the lower grade product. It was noted that the F-North material PGE recovery is very sensitive to product grade and mass pull.

13.2.4.4.2 MF1 Flotation Test work

The optimal circuit configuration (MF1 M14a rpt, Figure 13-11) from the Phase 1b test work campaign was used as the baseline flowsheet for the MF1 testing (PH3 EDF MF1 T4). Following the baseline test, an additional test was conducted to evaluate the effect of varying the addition ratio of Oxalic acid to Thiourea in the cleaner circuit, as summarized in Table 13-24.

Table 13-24: Phase 3 MF1 Flotation Tests

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH3 EDF MF1 T4	Phase 3 MF1 baseline test applying the Phase 1 MF1 M14a rpt flowsheet and test conditions
PH3 EDF MF1 T6	Baseline test conditions with Oxalic acid and Thiourea, in a ratio of 60:40, added to the cleaner circuit

Test 4 delivered the best MF1 result by achieving a high grade final product of 91 g/t (2E+Au) at 76% recovery and 1.1% mass pull, or a lower grade 56 g/t (2E+Au) product at 81% recovery and 5% mass pull. The copper and nickel recoveries were 87% and 56% respectively for the lower grade product.

13.2.4.4.3 F-North MF1 vs MF2

Refer to Figure 13-15 for a comparison of the 2E + Au Upgrade Ratio (UGR) vs 2E + Au recovery for both circuits.

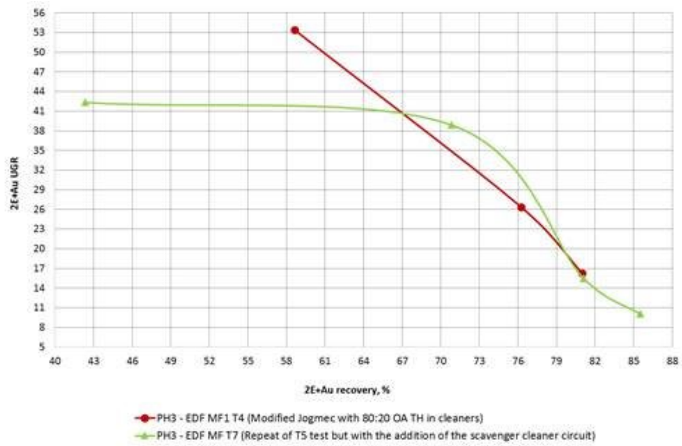


Figure 13-15: 2E+Au UGR-Recovery Curve (F-North MF1 vs MF2)

When comparing the results achieved in the MF2 open circuit tests to the MF1 open circuit tests, it was noted that on the F-North composite sample tested, the PGE flotation performance were marginally better for the MF2 circuit. The MF1 circuit achieved the higher nickel recovery (56% vs 54%), whilst both circuits achieved similar copper recoveries of ~88%.

13.2.4.5 Phase 4: Further Flotation Optimization Test Work

As per the Phase 1b campaign, the calculated head grades from the individual tests (based on concentrate and residue analysis) was included and compared to the assayed head grade. The head grade assays and calculated values for the T-zone samples tested is presented in Table 13-25 and Table 13-26.

Table 13-25: T-zone Composite Sample T2c Head Grade

Analytical Method Used	Pt g/t	Pd g/t	Au g/t	2E+ Au g/t	S %	Cu %	Ni %	MgO %	Fe %	SiO_2 %
3E (Pt, Pd, Au) [12]	1.49	2.15	1.14	4.78	0.56	0.23	0.13	8.12	6.10	48.24
Calculated from test work	1.29	2.04	1.04	4.37	0.49	0.23	0.14	7.86	6.07	47.48
T T2c composite average	1.39	2.10	1.09	4.57	0.53	0.23	0.13	7.99	6.08	47.86

[12] Fire Assay via Pb/Ag collection followed by the hot plate acid dissolution of the silver prill in an aqua-regia. Analysis of the Pt, Pd, and Au with ICP-OES.

Table 13-26: T-zone Composite Sample T2d Head Grade

Analytical Method Used	Pt g/t	Pd g/t	Au g/t	2E+ Au g/t	S %	Cu %	Ni %	MgO %	Fe %	SiO_2 %
3E (Pt, Pd, Au) [13]	1.18	2.23	0.77	4.17						
Calculated from test work	1.06	2.02	0.80	3.87	0.41	0.15	0.11	9.08	6.34	47.50
T T2d composite average	1.12	2.12	0.78	4.02	0.41	0.15	0.11	9.08	6.34	47.50

13.2.4.5.1 T-zone MF2 Open Circuit Test work

The Phase 4 baseline conditions was based on the optimum test conditions and flowsheet established during the Phase 1b (Test VT8) T-zone test campaign — refer to Section 13.3.4.

Based on the Phase 1b test observations relating to pyrrhotite recovery and grade dilution, higher depressant dosages were employed with the aim of depressing the pyrrhotite. The test work, conducted on the T-zone, T2c sample showed flotation performance could be improved, despite the presence of pyrrhotite.

A recovery of 82% (2E+Au) and corresponding concentrate grade of 70 g/t was achieved in the baseline test at high mass pull of 5.4%. The product grade improvement was attributed to the following:

- Differences in sample performance between the Phase 4a T-zone, T2c composite sample and the Phase 1b T-zone, T2b individual core samples.
- Increased depressant addition, to control the froth, improved process control when compared to the Phase 1b test campaign.

Based on the results from the baseline test, additional MF1 cleaner tests, as per Figure 13-16, were conducted aimed at optimizing the primary circuit reagent additions in order to reduce pyrrhotite recovery.

Table 13-27 provides a summary of the test conditions aimed at evaluating the effect of different depressant types on pyrrhotite rejection and metal recovery.

[13] Fire Assay via Pb/Ag collection followed by the hot plate acid dissolution of the silver prill in an aqua-regia. Analysis of the Pt, Pd, and Au with ICP-OES.

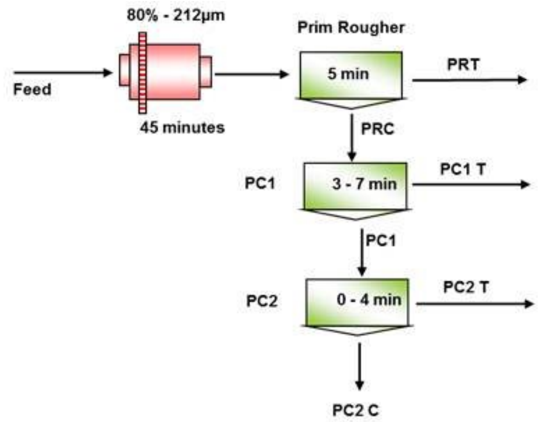


Figure 13-16: MF1 Cleaner Testing Flowsheet

Table 13-27: Phase 4a Effect of Depressant Type on T-zone Ore Pyrrhotite Recovery

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4 T2c MF2 T1	Phase 4 baseline test conditions, based on Phase 1b test VT8, together with higher depressant dosage rates
PH4 T2c MF2 T2	Primary circuit baseline test using 30 g/t Sendep 30E depressant in the rougher circuit, 100 g/t in the primary cleaner, and 60 g/t in the primary re-cleaner stage
PH4 T2c MF2 T3	Using 30 g/t Sendep 30E and 250 g/t Dextrin Starch depressant in the rougher circuit, 100 g/t Sendep 30E and 100 g/t Dextrin Starch depressant in the primary cleaner stage
PH4 T2c MF2 T4	Using 30 g/t Sendep 30E and 65 g/t Cytec 7261 depressant in the rougher circuit, 50 g/t Sendep 30E and 15 g/t Cytec 7261 depressant in the primary cleaner, and 30 g/t Sendep 30E and 10 g/t Cytec 7261 depressant in the primary recleaner stage
PH4 T2c MF2 T5	Using 30 g/t Sendep 30E and 12 g/t KU92 guar depressant in the rougher circuit, 50 g/t Sendep 30E and 12 g/t KU92 guar depressant in the primary cleaner stage

The conditions used in the baseline test resulted in poor metal recovery and upgrading in the rougher circuit. It is believed that this was due to the coarse grind being applied in the primary circuit (80% - 212µm) rather than the additional depressant addition aimed at depressing pyrrhotite.

The primary rougher sulphur recovery was evaluated to determine if the various depressants were able to reduce the pyrrhotite recovery. The following was noted from the results:

- The use of Dextrin Starch resulted in higher 2E+Au recoveries (53% vs 49%) when compared to the baseline test conditions, with comparative primary cleaner concentrate grades. The mass pulls varied only slightly between the two tests (1.71% vs 1.76%). The sulphur recovery in the rougher concentrate was marginally lower compared to the baseline test (85% vs 86%).

- The use of the Cytec 7261 depressant resulted in higher 2E+Au recovery to the primary cleaner concentrate (48% vs 39%) when compared to the baseline test at similar PGE upgrade ratios. However, 2E+Au recovery loss to the primary circuit tailings was higher compared to the test in which Dextrin Starch was used. The sulphur recovery to the rougher concentrate product was significantly lower compared to the baseline test (80% vs 86%).

- The use of KU92 Guar depressant resulted in higher primary circuit 2E+Au recovery at similar PGE upgrade ratios compared to the baseline test conditions, but lower compared to the test which used the Cytec 7261 depressant. The sulphur recovery to the rougher concentrate was lower than the baseline test and similar to the test in which the Cytec 7261 depressant was used.

The objective of the above testing was to find an operating condition in the primary rougher stage in which there was not a significant PGE loss during further upgrading. All four tests reported significant PGE losses in the primary cleaning stages; 23% to 30% of the PGEs recovered in the rougher concentrate reported to the primary cleaning tailings. However, it was noted that the conditions used in Tests 4 and 5 produced rougher concentrates with the lowest S recovery. Based on this observation, it was decided to continue further MF2 tests utilizing KU92 guar depressant, as per Table 13-28.

Table 13-28: Phase 4 MF2 Circuit Optimisation Testing on the T2c Sample

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4 T2c MF2 T1	Phase 4 baseline test conditions, based on Phase 1b test VT8, together with higher depressant dosage rates
PH4 T2c MF2 T6	T1 baseline conditions with an extended cleaner scavenger residence time of 60 minutes
PH4 T2c MF2 T7	T6 conditions with the use of guar and lime in the secondary scavenger circuit
PH4 T2c MF2 T10	T7 conditions with additional guar and lime in the secondary scavenger circuit

It is noted that all of the above tests included a single primary cleaner stage (compared to 2-stage primary cleaning required for the F4 sample). Test 7 achieved the best results with a 2E+Au recovery of 80% at a product grade of 83 g/t. The copper and nickel recoveries were 85.6% and 42.7% respectively.

Further test work was conducted on the T-zone material, however, due to sample availability a new sample, T2d, was prepared for the additional test work. Refer to Section 13.2.1.3 for more details on the sample selection and preparation.

The additional tests conducted on the T2d sample is summarized in Table 13-29.

Table 13-29: Phase 4 MF2 Circuit Optimisation Testing on the T2d Sample

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4b T2d MF2 T1	Phase 4b baseline test conditions, based on Phase 4 T2c MF2 T1, to assess variability between the T2c and T2d samples, at a secondary grind of 80% -75µm, and guar addition to the scavenger cleaner circuit
PH4b T2d MF2 T2	T1 baseline conditions utilising a finer secondary grind of 83% - 75µm, and the addition of guar depressant in the scavenger cleaner circuit
PH4b T2d MF2 T3	T1 baseline condition with the addition of $CuSO_4$ to the secondary rougher circuit and guar depressant in the scavenger cleaner circuit
PH4b T2d MF2 T4	T1 baseline condition with additional depressant in the primary and secondary rougher circuits and guar depressant in the scavenger cleaner circuit
PH4b T2d MF2 T5	T1 baseline conditions utilising a finer secondary grind of 90% - 75µm and no guar addition
PH4b T2d MF2 T8	Repeat of T5
PH4b T2d MF2 T9	T1 baseline conditions with no guar addition
PH4b T2d MF2 T10	Repeat of T9

The above tests indicated that the T-zone sample was amenable to a finer secondary grind (90%- 75µm) as the 2E + Au recovery of Test 5 was ~ 1% higher compared to Test 9 and Test 10 at an upgrade ratio of 22. It was also noted that the test, which applied a secondary grind of 83% -75µm, resulted in lower PGE recoveries when compared to the base case. The finer grind further resulted in increased copper recoveries, with similar nickel recoveries noted.

13.2.4.5.2 T-zone MF1 Open Circuit Test work

A single MF1 test (PH4 T2c MF1 T8) was conducted on the T-zone, T2c composite sample. The test achieved a final product grade of 60 g/t 2E+Au (upgrade ratio of 15.5) at a recovery of 77%. The copper and nickel recoveries were 89% and 58% respectively. It is noted that poor Pt and Au test accountabilities were recorded on this test. If the assays are aligned to within 10% test accountability, the test could have possibly achieved a 65 g/t 2E+Au grade at a 78% recovery.

Further MF1 circuit testing, as per Table 13-30, was conducted on the T-zone, T2d composite sample (Refer to Section 13.2.1.3)

Table 13-30: Phase 4 MF1 Circuit Optimisation Testing on T2d Sample

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4b T2d MF1 T1	Baseline circuit based on the Phase 2 MF2 T10 flowsheet (i.e. M14a flowsheet with the addition of CuSO4 to the rougher circuit)

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4b T2d MF1 T2	T1 baseline conditions utilising a finer secondary grind of 83% - 75µm
PH4b T2d MF1 T3	Repeat of T1

Results from the above tests indicated that a finer grind in the MF1 circuit did not result in higher PGE recoveries. Test 1 achieved the best results at a 2E + Au recovery of 73% at a product grade of 102 g/t (upgrade ratio of 28), at a mass pull of 2.65%. The associated copper recovery was 89% whilst 45% of the nickel was recovered.

13.2.4.5.3 T-zone MF1 vs MF2

Refer to Figure 13-17 for a comparison of the 2E + Au Upgrade Ratio (UGR) vs 2E + Au recovery for both circuits.

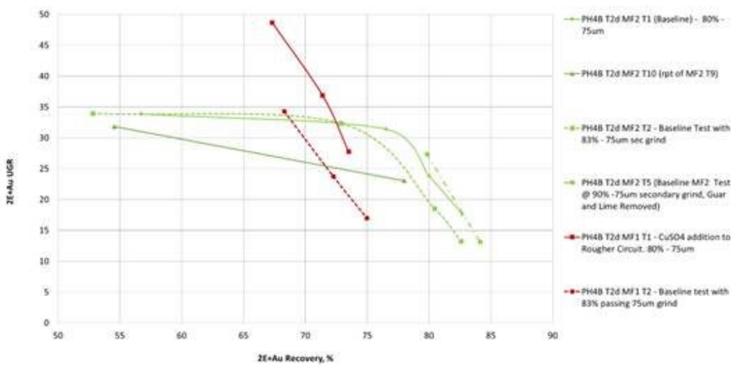


Figure 13-17: 2E+Au UGR-Recovery Curve (T MF1 vs MF2)

When comparing the results achieved in the MF2 open circuit tests to the MF1 open circuit tests, it was noted that on the T-zone composite sample tested, the PGE flotation performance were substantially better for the MF2 circuit. The MF1 baseline circuit achieved the higher copper recovery when compared to the MF2 baseline circuit (88% vs 84%) whereas the MF2 circuits achieved slightly higher nickel recoveries (47% vs 45%).

13.2.4.5.4 F-Boundary MF2 Open Circuit Test work

Due to the limited amount of flotation test work conducted on individual F-Boundary drill core material as part of Phase 1, further open circuit batch test work was conducted on the F-Boundary composite sample (Refer to Section 13.2.1.2).

During these tests, superior performance was noted compared to the Phase 1b results achieved. A 2E + Au recovery of ~85% at a mass pull of 4.2% to produce a product grade of 71 g/t (upgrade ratio of 20) when targeting 80% - 75µm secondary grind.

13.2.4.5.5 Mine Blend MF2 Open Circuit Test work

Two additional open circuit tests were conducted on the Mine Blend c sample (Refer to Section 13.2.1.5) to investigate the effect of a finer secondary grind (90% -75µm) on a mine blend sample. The head grade assays and calculated values for the Mine Blend c sample tested is presented in Table 13-31. Refer to Table 13-32 for a summary of the tests conducted.

Table 13-31: Mine Blend C Composite Sample Head Grade

Analytical Method Used	Pt g/t	Pd g/t	Au g/t	2E+ Au g/t	S %	Cu %	Ni %	MgO %	Fe %	SiO_2 %
3E (Pt, Pd, Au) [14]	0.98	1.93	0.45	3.36	0.36	0.11	0.16	16.33	7.32	45.78
Calculated from test work	0.95	2.10	0.45	3.50	0.39	0.11	0.16	15.98	7.54	45.62
Mine Blend c composite average	0.96	2.02	0.45	3.43	0.37	0.11	0.16	16.16	7.43	45.70

Table 13-32: MF2 Tests to Investigate the Effect of Grind on Mine Blend c Sample

Test Reference (Phase-Sample-Circuit-Test ID)	Test Description
PH4b Mine Blend c T6	Phase 4b T2d MF2 T5 conditions (90% - 75µm secondary grind)
PH4b Mine Blend c T7	Phase 4b T2d MF2 T5 conditions (80% - 75µm secondary grind)

The associated 2E + Au recovery — UGR curves for each of the tests are illustrated in Figure 13-18. The results indicated that a secondary grind of 90% - 75µm was detrimental to the 2E + Au recovery, as a 4% lower recovery was reported at an upgrade ratio of 20 (~ 70 g/t 2E+Au product). The finer grind resulted in increased copper recovery (88% vs 86%), however, the finer grind had a negative affect the nickel recovery reported (42% vs 46%).

[14] Fire Assay via Pb/Ag collection followed by the hot plate acid dissolution of the silver prill in an aqua-regia. Analysis of the Pt, Pd, and Au with ICP-OES.

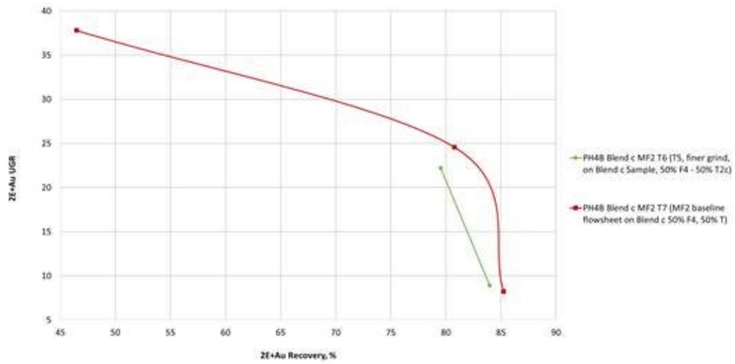


Figure 13-18: 2E+Au UGR-Recovery Curve for MF2 Tests Conducted on Mine Blend c Sample

13.2.4.5.6 General Notes on Phase 4 Flotation Test work

The following is noted about the Phase 4 test work, in addition to the above discussions:

- Different individual metal recoveries were noted for the precious metals. Platinum metal recovery is generally higher than Palladium recovery on by a few basis points (between 3% — 7% on the T2c and T2d samples). Gold metal recovery is generally the lowest, being between 12% - 18% lower than the Platinum metal recovery.
- Reagent optimization test work on the T-zone material, in the primary circuit, was conducted with the aim at depressing pyrrhotite and improving the product grade. The results indicated that this could not be achieved without compromising on precious metal recovery. The use of a KU92 guar depressant showed potential to reduce sulphur recovery and can possibly be incorporated into the secondary flotation circuit of an MF2 configuration.
- Longer secondary scavenger cleaner residence times were necessary during the F-Boundary test work to improve the overall 2E+Au recovery, when compared to the F-Central flowsheet.

13.2.4.6 Flotation Locked Cycle Testing

13.2.4.6.1 MF2 Circuit

Locked cycle tests, which are defined as being repetitive batch flotation tests, were used to simulate a continuous circuit on a number of samples.

Table 13-33 provides a summary of the head grades for each of the MF2 locked cycle tests conducted. The results obtained from each locked cycle test are summarized in Table 13-34.

Table 13-33: Sample Head Grades for MF2 Locked Cycle Tests

	Pt	Pd	Au	Rh	4E	S	Cu	Ni	MgO	Fe
Test Reference	g/t	g/t	g/t	g/t	g/t	%	%	%	%	%
F-Central F4 Sample MF2 Locked cycle test #1	0.86	2.27	0.19	0.06	3.38	0.31	0.07	0.18	22.80	8.43
F-Central F4 Sample MF2 Locked cycle test #2	0.93	2.24	0.23	NR	3.40	0.32	0.07	0.18	22.38	8.43
T T2c Sample MF2 Locked cycle test	1.15	1.86	1.13	NR	4.14	0.41	0.19	0.11	7.87	6.02
F-Boundary Sample MF2 Locked cycle test #1	1.23	2.64	0.18	0.09	4.13	0.89	0.13	0.29	20.75	9.37
F-North Sample MF2 Locked cycle test	0.92	2.41	0.19	NR	3.51	0.49	0.11	0.24	20.17	8.53
Mine Blend a Sample MF2 Locked cycle test #1	0.89	1.96	0.47	NR	3.31	0.51	0.15	0.25	18.50	8.49

Table 13-34: Locked Cycle Test Results for MF2 Flowsheet

		Final Product Grade						Recovery		
	Mass	4E	Cu	Ni	S	Fe	MgO	4E	Cu	Ni
Test Reference	%	g/t	g/t	g/t	g/t	%	%	%	%	%
F-Central F4 Sample MF2 LCT #1 (Low mass pull option)	2.1	127.8	2.9	3.8	9.3	16.5	17.2	78.2	77.7	39.4
F-Central F4 Sample MF2 LCT #2 (High mass pull option)	4.0	64.1	2.5	2.1	5.5	12.5	20.8	78.4	82.0	39.0
T T2c Sample MF2 Locked cycle test	4.0	69.9	3.7	1.3	7.9	13.3	8.6	72.5	84.9	49.3
F-Boundary Sample MF2 Locked cycle test #1	2.4	103.5	4.0	5.4	11.7	17.7	16.9	65.0	80.5	44.7
F-North Sample MF2 Locked cycle test (Low mass pull option)	1.9	129.3	5.0	5.9	14.8	17.4	13.7	70.4	87.2	47.5

		Final Product Grade						Recovery		
Test Reference	Mass %	4E g/t	Cu g/t	Ni g/t	S g/t	Fe %	MgO %	4E %	Cu %	Ni %
F-North Sample MF2 Locked cycle test	5.4	52.1	1.8	2.4	6.4	12.5	18.4	80.7	92.3	55.7
Mine Blend a Sample MF2 Locked cycle test #1	3.9	65.7	3.2	2.9	8.6	14.6	17.3	76.6	88.8	46.8

The T-zone T2c sample locked cycle operation using the flowsheet and conditions as per the PH4 T2c MF2 T7 open circuit test was not successful in improving on the recovery or grade prediction achieved in the open circuit test.

It is expected that better liberation can result in better recoveries and upgrading, as per Section 13.2.2.4.

13.2.4.6.2 MF1 Circuit

Table 13-35 provides a summary of the measured head grades for each of the Modified JOGMEC MF1 locked cycle tests conducted. The results obtained from each locked cycle test are summarized in Table 13-36.

Table 13-35: Sample Head Grades for MF1 Locked Cycle Tests

Test Reference	Pt g/t	Pd g/t	Au g/t	Rh g/t	4E g/t	S %	Cu %	Ni %	MgO %	Fe %
F-Central F4 Sample MF1 Locked cycle test	0.89	2.06	0.16	NR	3.11	0.31	0.08	0.18	24.82	8.76
Mine Blend a Sample MF1 Locked cycle test	0.89	1.96	0.47	NR	3.31	0.51	0.15	0.25	18.50	8.49

Table 13-36: Locked Cycle Test Results for MF1 Flowsheet

		Final Product Grade						Recovery		
Test Reference	Mass %	4E g/t	Cu g/t	Ni g/t	S g/t	Fe %	MgO %	4E %	Cu %	Ni %
F-Central F4 Sample MF1 Locked cycle test	2.5	89.7	2.2	3.2	8.6	11.0	41.3	77.0	76.5	41.5
Mine Blend a Sample MF1 Locked cycle test	2.7	91.8	4.5	4.3	13.0	17.5	17.1	76.8	79.4	46.7

13.2.5 Other Test Work

Additional test work was conducted as part of the Phase 1a and Phase 1b campaign to provide process design data and guide flowsheet development. The following additional test work was conducted:

- Magnetic separation testing on a final concentrate product produced from a F4 sample by applying a standard MF2 processing route, to investigate the possibility of iron reduction in the final product and to assess the effect of such a process on the PGE recoveries.

- Tailings dewatering test work on a flotation tailings sample (at a grind of 80% passing 75µm) was conducted using the F4 sample (60:40 F-Central FH upper: F-Central FH lower). The sample was submitted, in May 2015, to Vietti Slurrytec in South Africa for particle size and high level mineralogical characterization, thickening and filtration test work.

- As part of the thickening test work, Vietti Slurrytec prepared a typical thickener underflow sample, which was in turn submitted to Paterson & Cooke Consulting Scientists in South Africa for rheological characterization test work.

- Heavy Liquid Separation (HLS) test work was conducted at MINTEK during December 2014 on a single F-Central drill core sample to assess the amenability of the F-Central material to density pre-concentration.

13.2.5.1 Magnetic Separation

The combined concentrate product, as produced during the Phase 1a campaign, by the MF1 open circuit tests T12, T15, T18, T21, and T24 was submitted for magnetic separation amenability test work. The investigation was aimed at reducing the iron content in the flotation concentrate by means of magnetic separation method. This would reduce subsequent furnace matte fall and produce an enhanced final product. The test work comprised of hand-held magnetic separation, Davis tube tests and Wet High Intensity Magnetic Separation (WHIMS).

The, hand-held magnetic separation as well as the WHIMS testing revealed limited amenability of the concentrate product to magnetic separation treatment due to high losses of precious metals.

PGE losses to the iron fraction of between 15% and 38% (hand-held and WHIMS tests respectively) was reported. Further investigations were thus abandoned.

13.2.5.2 Tailings Dewatering Test work

The following findings and conclusions were made from the thickening test work:

- The material was found to be naturally dispersive (non-settling) in the unflocculated state, due to the presence of smectite and talc clays, together with the low conductivity value of the water15 used to prepare the slurry.

- Magnafloc 1597, at a dosage of 200 g/t, was selected as the conditioning agent (coagulant) for the tailings slurry.

- The optimum flocculant for thickening was selected as Magnafloc 919, at a dosage of 20 g/t.

- The optimum thickener feed solids concentration of 10% w/w was noted.

[15] Rand Water Board water was used as water source during the sample preparation at MINTEK.

- The optimum solids flux rate when utilizing a high rate thickener was determined as 0.4 t/h/m2.
- An underflow slurry, with a solids concentration of 60% w/w, was predicted by Vietti Slurrytec for a high rate thickening application.
- The optimum solids flux rate when utilizing a paste thickener was determined as 0.5 t/h/m2.
- An underflow solids concentration of 67% w/w was noted for a paste thickening application.
- The un-sheared vane yield stress of the Waterberg Tailings sample was 197 Pa under high rate conditions and 356 Pa under Paste conditions at an underflow solids concentrations of 63% w/w and 71% w/w respectively.

13.2.5.3 Tailings Filtration Test work

The following findings and conclusions were made from the tailings filtration test work:

- The material does dewater under vacuum filtration, although test work highlighted that it is imperative to thicken the slurry ahead of filtration.
- Low filtration rates were achieved for vacuum filtration, and Polymer coagulation is required
- A filter cake moisture of 24% by mass was achieved during testing with a design flux of 0.410 t/h/m2.

13.2.5.4 Heavy Liquid Separation Test work

HLS test work was conducted at MINTEK during December 2014 on a single F-Central drill core sample, namely, WD151-D2. Test work was performed at various density intervals and 3 different top sizes, namely; -30mm + 1mm, -20mm + 1mm, and -10mm + 1mm, to assess the amenability to density pre-concentration.

The results from the heavy liquid separation test work indicate limited scope for pre-concentration based on density. Albeit that waste rejection of up to 40% could be achieved, high precious metal losses (in excess of 20%) will render the application uneconomical. Further investigations were thus abandoned.

13.3 Process Plant Recovery Estimate

The expected process plant recovery estimates were derived using both open and closed circuit data obtained from test work on the various main Waterberg deposit lithology units. All data was obtained using proven, laboratory scale, testing techniques. The recovery correlations derived are based on the open circuit test results, for an MF2 circuit. The results from the confirmatory locked cycle tests conducted were compared to the open circuit recovery data. Test work on ore type blends (to reflect the expected mining blends) was also considered and compared to the results obtained from tests conducted on individual geological units.

13.3.1 Flowsheet Selection

As described in the preceding test work discussions, both the MF1 and MF2 flowsheets were evaluated. A techno-economic trade-off study, which considered capital costs, operating costs and revenue differentials, was conducted to select the optimal flowsheet for the Waterberg project. The trade-off relied on the results obtained during the test work phases, along with a number of other cost estimates.

Figure 13-19 provides a summary of the PGE recoveries and corresponding upgrade ratio each flowsheet option and for different ore types.

Test work has indicated that, for the required product specification, the MF1 circuit achieved marginally higher PGE recoveries on the F-Central ore, while the MF2 flowsheet resulted in higher PGE recoveries on the remainder of the ore types. Flotation test work conducted on the Mine Blend a and b samples (refer to Section 13.2.1.5), returned superior recoveries and upgrade ratios with the MF2 circuit when compared to the MF1 circuit.

Albeit that a single milling stage was employed to simulate the MF1 circuit on laboratory scale, two stages of milling in series (as a Mill-Mill-Float) would be required to achieve the target grind of 80% passing 75µm. To achieve the plant throughput, the high mill power requirements can only be achieved in a single milling step by utilizing large, uncommon grinding equipment sizes. Because of both circuits have two stages of milling to achieve the target grind, the capital cost differential between the milling circuits is expected to be low.

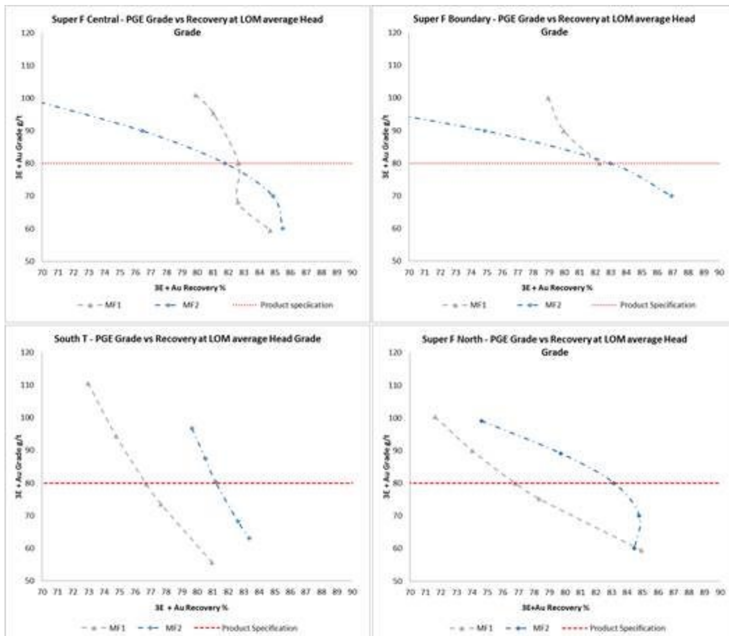


Figure 13-19: PGE Recovery Potential for Waterberg Ores at the Life of Mine Average Head Grade

The following comments about the major process equipment in both circuits are noted:

- Identical run-of-mine ore receiving and crushing circuits will apply to either circuit.
- Similar milling circuits will apply as the final grind on the flowsheets are identical.
- The MF1 circuit will require a less elaborate and costly flotation circuit due to fewer cleaning circuits, accompanied by fewer low pressure flotation air blowers.
- The tailings thickening and handling circuits for both circuit will be identical.
- The concentrate thickening and handling circuits for both circuits will be identical.
- The services and infrastructure requirements for both circuits will be identical.
- The reagent make-up, storage and distribution circuits for the MF1 circuit will be more costly due to more reagents being required when compared to the MF2.

The following differences in operating costs were noted:

- Approximately 10% higher power consumption is expected for the MF2 circuit due to the additional flotation mechanical items
- Based on reagent consumption rates and typical supply costs to site, the MF2 circuit reagent costs were calculated as being approximately 5% higher when compared to the MF1 circuit

A high level financial evaluation was performed which took into account the differences in recoveries, operating costs and capital expenditure between the two flowsheets under consideration, based on the mine blend test work. Refer to Table 13-37 details surrounding the inputs into this financial evaluation.

Table 13-37: Flowsheet Selection Financial Model Input Summary

Financial Model Input Parameter	Unit of Measure	Input Value	
		MF2 circuit	MF1 circuit
ROM grade	4E	3.58	
Mill feed tonnage	tpm	300 000	
Mill feed grade	4E	3.58	
Final Product grade	4E	90	
Mass pull	% of Mill Feed	3.20	3.02
Metal Recoveries			
Pt	%	80.2	73.6
Pd	%	83.1	79.4
Au	%	70.2	63.1
Rh	%	63.6	58.5
Cu	%	84.6	83.5
Ni	%	45.1	46.5
Metal Prices			
Pt	USD/oz.	1203	
Pd	USD/oz.	708	

Financial Model Input Parameter	Unit of Measure	Input Value MF2 circuit	MF1 circuit
Au	USD/oz.	1231	
Rh	USD/oz.	973	
Cu	USD/lb	6.9	
Ni	USD/lb	2.9	
Rate of Exchange	R/USD	15	
Process plant operating cost	USD/t milled	9.43	8.88
Process plant capital cost estimate	USD	$ 200 000 000	Variable

It is noted, that for the purpose of this evaluation, the final product specification was selected as 90 g/t 4E, as the mass pull achieved in the MF1 mine blend test modelled was not high enough to achieve a 80 g/t final product (as per Figure 13-20).

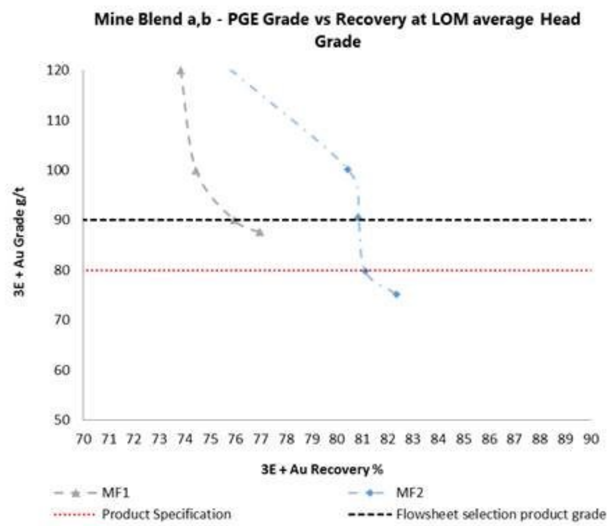


Figure 13-20: PGE Recovery Potential for Waterberg Mine Blend a and b Samples

Identical metal payability factors were applied to each of the flowsheet options evaluated while the capital cost requirement of the MF1 circuit was varied. The evaluation showed that the MF1 circuit would result in a superior NPV over the MF2 circuit, if the MF1 circuit capital cost requirement where ~35% of the MF2 circuit capital cost requirement. Refer to Figure 13-21 below for an illustration of the evaluation outcome.

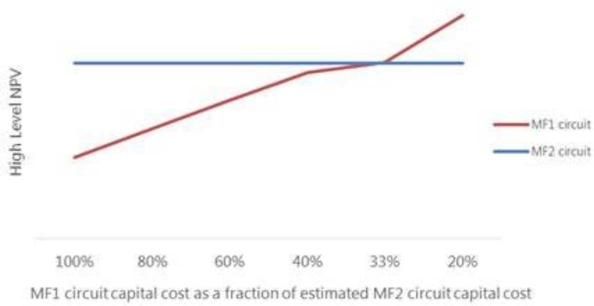


Figure 13-21: Summary of Flowsheet Selection Financial Evaluation

Based on the above, the MF2 flowsheet was selected as the basis for the Waterberg PFS.

13.3.2 MF2 Recovery Test work

The MF2 circuit test results as per Table 13-38 below were used to derive the plant recovery estimates for a MF2 circuit.

Table 13-38: MF2 Test work Data Used for Recovery Modelling

Test work Phase	Ore Type	Circuit	Test Description (Phase-Sample-Circuit-Test ID)	Test Type
Phase 1b	F-Central	MF2	PH1 F4 MF2 New Test 6	Open circuit
Phase 1b		MF2	PH1 F4 MF2 LCT#1	Locked cycle
Phase 4	F-Boundary [16]	MF2	PH4 F-Boundary Test 1	Open circuit
Phase 1b	F-North [17]	MF2	PH1 F-North MF2 LCT	Locked cycle
Phase 3		MF2	PH3 EDF MF2 T7	Open circuit
Phase 3		MF2	PH3 EDF MF2 LCT	Locked cycle
Phase 4	T-zone	MF2	PH4 T2c MF2 T1	Open circuit
Phase 4		MF2	PH4 T2c MF2 LCT	Locked cycle
Phase 1b	Mine Blend 25% T:50% F-Central: 25% F-Boundary	MF2	PH1 Blend a MF2 New Test 9	Open circuit
Phase 1b		MF2	PH1 Blend b MF2 New Test 9 [18]	Open circuit
Phase 1b		MF2	PH1 Blend a MF2 New Test 9 LCT	Locked cycle

[16] F-Boundary material were referred to as "F-North" in earlier phases of test work
[17] F-North material were referred to as "Early Dawn F" in earlier phases of test work
[18] Platinum and copper assays are not used in the recovery estimate due to poor accountabilities in the specific test.

No test work has been conducted on the F-South material during the PFS test work campaigns. Recovery correlations as per F-Central ore have been applied to the F-South material. The F-South material contributes only 1% of the tonnage during the first 5 years of operation, and only 10% of the total tonnage over life-of-mine.

13.3.3 Mine and Plant Feed Schedules

The process plant will comprise of two (2) parallel modules, each capable of processing 300 000 tonnes of run of mine material per annum. Aligned with the mining production schedule, the first module is scheduled to be bought online in month 36 whilst production from the second module is scheduled to start in month 53. A plot of the preliminary plant feed schedule for Module 1 and Module 2 (Figure 13-22 and Figure 13-23 respectively) and combined 4E, Cu and Ni plant feed grades (Figure 13-24) is presented below.

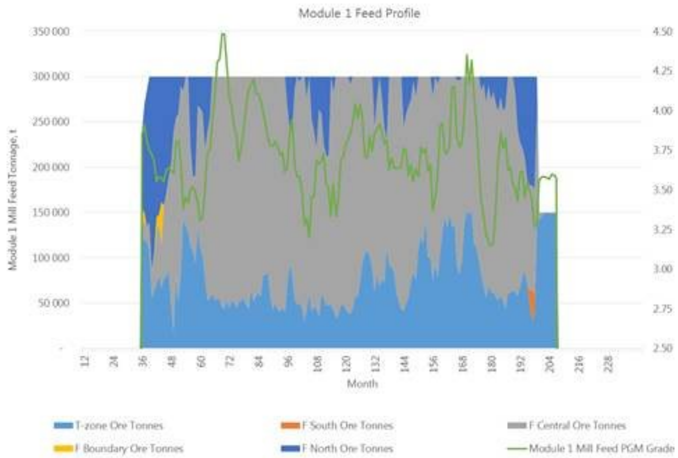


Figure 13-22: Module 1 Life-of-Mine Plant Feed Profile

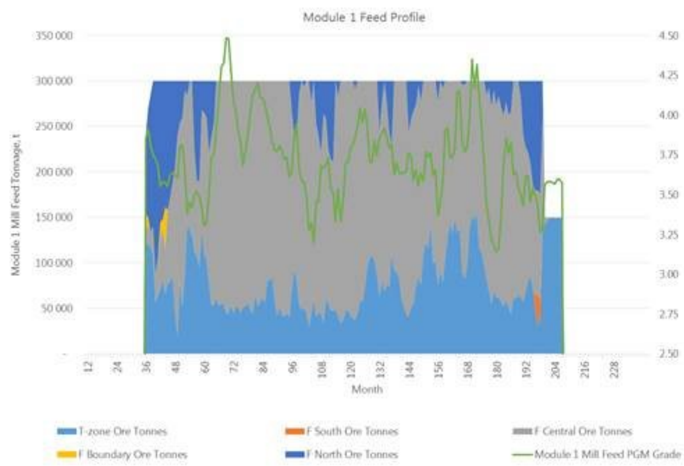


Figure 13-23: Module 2 Life-of-Mine Plant Feed Profile

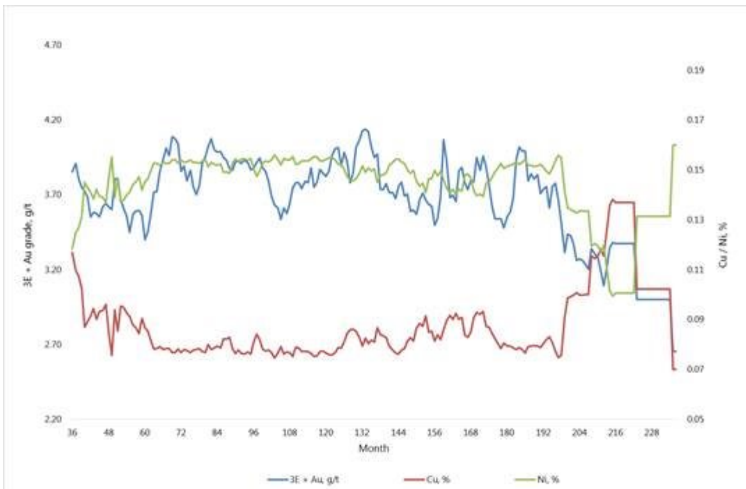


Figure 13-24: Life-of-Mine Metal Mill Feed Grade Profile

The following is noted from the preliminary plant feed schedule:

- Lithologies being treated include:
  - T-zone
  - F South
  - F-Central
  - F-Boundary
  - F-North
- The PGE (4E) mill feed grade is expected to vary between 2.65 g/t and 4.14 g/t with a life-of-mine average value of 3.73 g/t.
- The copper mill feed grade is expected to vary between 0.07% and 0.14% with a life-of-mine average value of 0.08%.
- The nickel mill feed grade is expected to vary between 0.10% and 0.16% with a life-of-mine average value of 0.15%
- The blend being processed during the first 5 years of production includes roughly 40% of F-Central, 15% T-zone, 15% F-Boundary, and 30% F-North.

13.3.4 Basis of Recovery Estimate

As part of the Phase 1b test work campaign, individual drill core samples were tested on a MF2 flowsheet to investigate the effect of varying head grade on the flotation performance. These included samples from Central FH Upper, Central FH Lower, T-zone, F-Boundary and the Mine Blend a sample (see Section 13.2.1.5). It is noted that these tests were not conducted on the optimum MF2 flowsheet as per Figure 13-8, as these tests were conducted prior to the conclusion of the MF2 circuit development work.

When considering the relationship between the mass pull and PGE upgrade ratio achieved it was noted that the upgrade ratio is generally independent of the sample head grade, for mass pull higher than 2%. This was true for the F-Central, F-North and F-Boundary ore types. The test work done on the T-zone material did not report a similar correlation, however, at the time of conducting this test work only minor work had been done on the T-zone material.

Refer to Figure 13-25 to Figure 13-28 for an illustration of the trends noted.

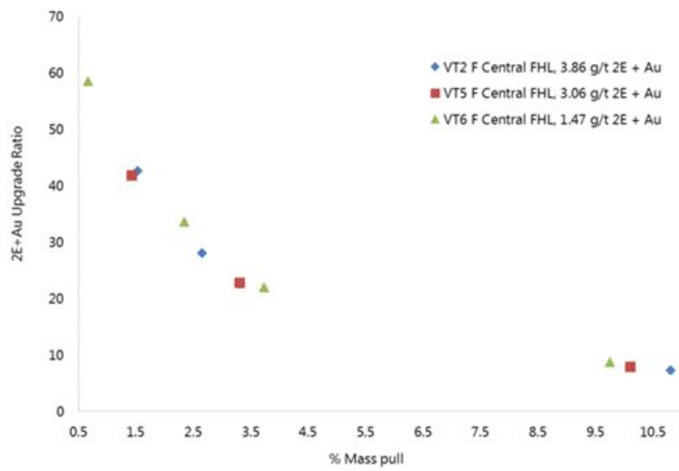


Figure 13-25: Correlation between Mass Pull and PGE UGR for F-Central FH Lower Ore

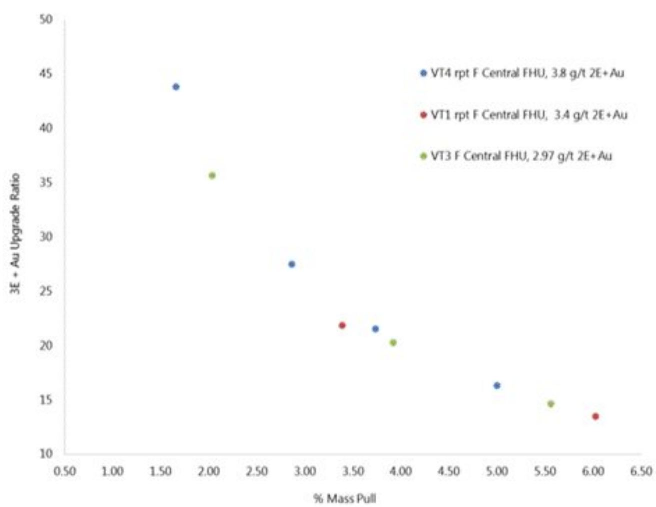


Figure 13-26: Correlation between Mass Pull and PGE UGR for F-Central FH Upper Ore

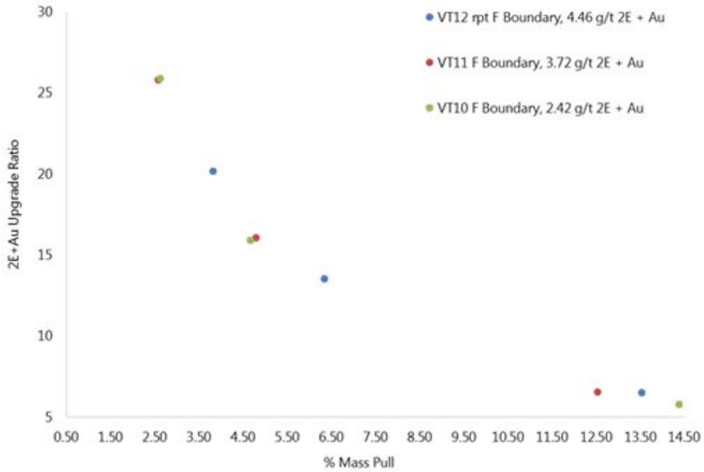


Figure 13-27: Correlation between Mass Pull and PGE UGR for F-Boundary Ore

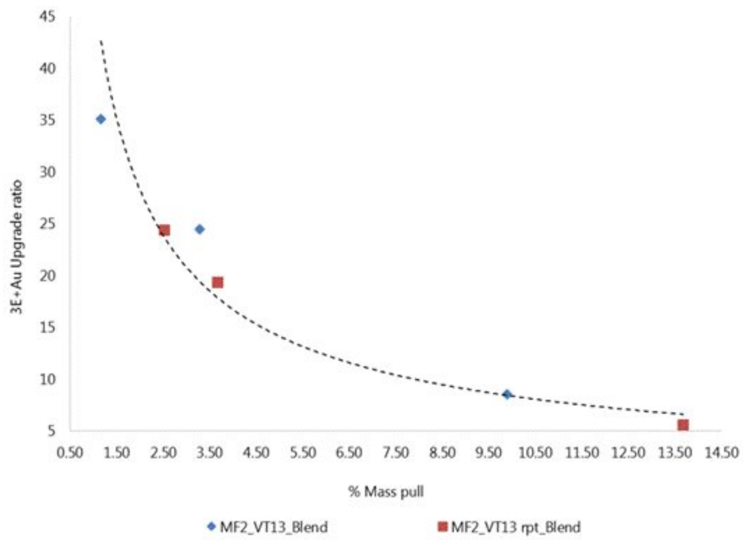


Figure 13-28: Correlation between Mass Pull and PGE UGR for Mine Blend a Sample

The above methodology, i.e. PGE upgrade ratio vs mass pull is independent of head grade, was used as a basis to model the expected recoveries from optimum test work results per ore type.

For each of the lithologies, a correlation between concentrate mass pull and platinum upgrade ratio (ratio between mill feed grade and final concentrate grade) were derived. The platinum UGR (upgrade ratio) was used as the basis, since the test work accountabilities for the platinum results were more consistent when compared to palladium, rhodium and gold.

Once the correlation between concentrate mass pull and platinum UGR was established, correlations between the platinum UGR and the other individual PGEs (Pd, Au, and Rh [19]) were established and used to determine the individual elemental recoveries, as well as the associated final product grades expected at different mass pulls. The recoveries for copper and nickel were based on correlations derived between the concentrate mass pull and the respective base metal UGRs. Correlations were also derived to determine required mass pulls at different PGE head grades, in order to produce a final product with of least 80 g/t.

The monthly blend's PGE recoveries were calculated based on weighted averages of the individual recoveries for each lithology, as it was found that this approach was within acceptable accuracy. However, for copper, the recoveries were based on the correlations derived from the mine blend test work data as it was noted that a weighted average of the individual lithology base metal recoveries under estimated the blend performance. Refer to Table 13-39 for a comparison between the Mine Blend modelled and calculated values, based on an 80 g/t product.

Table 13-39: Comparison between Mine Blend Modelled vs Calculated Recoveries

	Head Grade				Recoveries (%)						
	4E	Cu %	Ni %	Mass %	4E	Pt	Pd	Au	Rh	Cu	Ni
Mine Blend Modelled	3.58	0.09	0.14	3.64	81.1	80.7	83.3	70.0	64.4	87.8	46.6
Weighted Average Calculated	3.58	0.09	0.14	3.65	81.8	81.9	83.5	73.1	56.8	78.1	46.7
Variance	—	—	—	0.37%	0.83%	1.5%	0.2%	4.5%	11.8%	11.0%	0.2%

The locked cycle test results were plotted against the derived correlations to verify the accuracy of the developed models.

13.3.5 MF2 Circuit Recovery Equations for 80 g/t 4E Final Product

The methodology as outlined above was followed to derive correlations between final metal recovery and plant feed grade to produce a final concentrate containing 80 g/t (4E). The correlations are valid over the head grade range of between 2.5 g/t to 4.2 g/t, as validated by test work. These equations, specific to each lithology, are summarized in Table 13-40.

[19] Rh assays were conducted as part of the test work to assess the impact of sample head grade on recovery.

Table 13-40: MF2 Elemental Recovery Equations per Lithology – 80 g/t product

T-zone:
Mass pull % = -0.0976*(4E Head Grade) ^{3} + 1.0955*(4E Head Grade) ^{2} – 2.8462*(4E Head Grade) + 4.1944
Mass pull range = 2.36 % to 5.40 %

Metal	Elemental Recovery as a function of 4E head grade
Pt Recovery	= -9.4353*(Pt Head Grade) ^{2} + 30.52*(Pt Head Grade) + 64.738
Pd Recovery	= -3.0459*(Pd Head Grade) ^{2} + 16.518*(Pd Head Grade) + 60.34
Au Recovery	= -16.273*(Au Head Grade) ^{2} + 40.421*(Au Head Grade) + 53.41
Rh Recovery	= -0.2925*(4E Head Grade) ^{2} + 3.7959*(4E Head Grade)+ 18.195
Cu Recovery	= 1.2859*(Mass pull) ^{2} - 1.7101*(Mass pull) + 76.903
Ni Recovery	= 0.9637*(Mass pull) ^{3} - 10.394*(Mass pull) ^{2} + 37.745*(Mass pull)

F-South & F-Central:
Mass pull % = -0.32*(4E Head Grade) ^{2} + 3.6068*(4E Head Grade) – 5.082
Mass pull range = 1.93 % to 4.16 %

Metal	Elemental Recovery Equation
Pt Recovery	= -95.861*(Pt Head Grade) ^{4} + 563.04*(Pt Head Grade) ^{3} – 1235.2*(Pt Head Grade) ^{2} + 1199.3*(Pt Head Grade) – 350.46
Pd Recovery	= -4.9528*(Pd Head Grade) ^{4} + 61.581*(Pd Head Grade) ^{3} – 286.08*(Pd Head Grade) ^{2} + 588.4*(Pd Head Grade) - 365.87
Au Recovery	= -96958*(Au Head Grade) ^{4} + 95829*(Au Head Grade) ^{3} – 35357*(Au Head Grade) ^{2} + 5769.5*(Au Head Grade) - 281.86
Rh Recovery	= -0.1776*(4E Head Grade) ^{4} + 5.2546*(4E Head Grade) ^{3} - 50.775*(4E Head Grade) ^{2} + 201.86*(4E Head Grade) – 209.98
Cu Recovery	= 1.2859*(Mass pull) ^{2} - 1.7101*(Mass pull) + 76.903
Ni Recovery	= -1.5677*(Mass pull) ^{2} + 15.264*(Mass pull) + 6.0285

F-Boundary:
Mass pull % = -0.748*(4E Head Grade) ^{2} + 7.0599*(4E Head Grade) - 12.01
Mass pull range = 1.8 % to 4.2 %

Metal	Elemental Recovery Equation
Pt Recovery	= -344.39*(Pt Head Grade) ^{4} + 1797.9*(Pt Head Grade) ^{3} - 3581.6*(Pt Head Grade) ^{2} + 3224.9*(Pt Head Grade) - 1024.7
Pd Recovery	= -18.541*(Pd Head Grade) ^{4} + 206.41*(Pd Head Grade) ^{3} - 876.21*(Pd Head Grade) ^{2} + 1680*(Pd Head Grade) - 1137
Au Recovery	= -18.541*(Au Head Grade) ^{4} + 206.41*(Au Head Grade) ^{3} - 876.21*(Au Head Grade) ^{2} + 1680*(Au Head Grade) - 1137

F-Boundary:
Mass pull % = -0.748*(4E Head Grade) ^{2} + 7.0599*(4E Head Grade) - 12.01
Mass pull range = 1.8 % to 4.2 %

Metal	Elemental Recovery Equation
Rh Recovery	= -1.5737*(4E Head Grade) ^{4} + 27.566*(4E Head Grade) ^{3} - 186.52*(4E Head Grade) ^{2} + 576..18*(4E Head Grade) – 624.57
Cu Recovery	= 1.2859*(Mass pull) ^{2} - 1.7101*(Mass pull) + 76.903
Ni Recovery	= -1.3989*(Mass pull) ^{2} + 20.084*(Mass pull) + 4.0806

F-North:
Mass pull % = -0.0119*(4E Head Grade) ^{4} + 0.2217*(4E Head Grade) ^{3} - 1.6683*(4E Head Grade) ^{2} + 6.9274*(4E Head Grade) - 7.9303
Mass pull range = 1.76 % to 5.24 %

Metal	Elemental Recovery Equation
Pt Recovery	= 126.1*(Pt Head Grade) ^{3} - 500.24*(Pt Head Grade) ^{2} + 658.38*(Pt Head Grade) - 202.01
Pd Recovery	= 12.778*(Pd Head Grade) ^{3} - 108.15*(Pd Head Grade) ^{2} + 303.73*(Pd Head Grade) - 198.86
Au Recovery	= 26827*(Au Head Grade) ^{3} - 17715*(Au Head Grade) ^{2} + 3869.4*(Au Head Grade) - 197.22
Rh Recovery	= 2000000*(4E Head Grade) ^{3} - 287227*(4E Head Grade) ^{2} + 12924*(4E Head Grade) - 134.78
Cu Recovery	= 1.2859*(Mass pull) ^{2} - 1.7101*(Mass pull) + 76.903
Ni Recovery	= 2.9098*(Mass pull) ^{3} - 37.225*(Mass pull) ^{2} + 154.95*(Mass pull) - 152.19

13.3.6 PFS Process Plant Recovery

This PFS recovery estimate is based on the following inputs:

- 2 x 300ktpm MF2 concentrator plants, phased as detailed in Section 13.3.1
- Mine schedule detailing total tonnages and run of mine grades mined per ore type as detailed in Section 13.3.3.
- PGE, nickel and copper recoveries calculated as detailed in Section 13.3.3.
- Ramp-up and commissioning losses is included on each of the individual 3E + Au elements as well as copper and nickel, for each concentrate module, as follows:
  - Month 1 after mill start-up : 3%
  - Month 2 and month 3 after mill start-up : 2% per month
  - Month 4 and month 5 after mill start-up : 1% per month

The recovery estimate for this PFS is summarized below:

Table 13-41: Discounted Recoveries over Life-of-Mine

Element	Mill feed grade	Mass pull	Final Product Grade	Discounted Recovery (%)
3E + Au	3.73 g/t	3.78 %	81.1 g/t	82.1 %
Platinum	1.07 g/t	3.78 %	24.3 g/t	82.5 %
Palladium	2.20 g/t	3.78 %	50.5 g/t	83.2 %
Gold	0.28 g/t	3.78 %	5.7 g/t	75.3 %
Rhodium	0.04 g/t	3.78 %	0.6 g/t	59.4 %
Copper	0.08 %	3.78 %	2.0 %	87.9 %
Nickel	0.15 %	3.78 %	1.9 %	48.8 %

13.4 Concentration Specification

The flotation concentrate final product target specification is a 4E grade of at least 80 g/t. The expected mass pull to achieve an 81 g/t 4E product is 3.73 % based on a life-of-mine mill feed grade of 3.73 g/t 4E.

It is evident from the test work on the various ore types that the recoveries are very sensitive to changes in mass pull.

13.5 Metallurgical Variability

Limited variability test work was conducted as part of this PFS scope of work. Test work was conducted to investigate the variance in flotation performance between the various ore types.

13.6 New Metallurgical Test Work

The following test work is recommended for the next phase of the project:

- Flotation test work using water from the envisaged raw water sources to ensure the flotation performance is not negatively affected.
- Testing of the MF2 circuit using an Oxalic acid and Thiourea reagent scheme
- Comminution variability test work on the individual ore types
- Comminution variability test work on various possible mine blends
- Flotation open circuit batch variability test work on the individual ore types
- Flotation open circuit batch variability test work on various possible mine blends
- Concentrate thickening and filtration test work

13.7 Risks and Opportunities

The test work programmes undertaken for the Waterberg pre-feasibility study was of a suitable standard for a pre-feasibility study and was conducted at reputable institutions.

Data obtained from the various test work campaigns, and subsequent modelling and simulation allowed the following design activities to take place:

- Selection of a process flowsheet and reagent suite
- Mass and water balance development for a 2 x 300ktpm concentrator modules
- Sizing of major mechanical equipment
- Estimation of plant operating cost over life of mine

Portions of the plant operating costs and expected overall plant recoveries were derived from the laboratory test results. Based on the test work and engineering design performed as part of the pre-feasibility study a number of processing risks and opportunities have been identified.

13.7.1 Flowsheet

The fact that extensive test work has been conducted on two different flowsheets introduces a level of flexibility and opportunity.

There is opportunity to take advantage of the lower capital cost and operating cost potential on the MF1 flowsheet, if one were able to dedicated F-Central material only to one of the concentrator plant modules.

13.7.2 Nugget Effect on Assaying

Head grade analysis, specifically on the Central F material, using a variety of analytical methods, resulted in notable assay variability despite a number re-assay checks. This is most likely attributable to coarse nugget effects, mostly noted on the Au and Pd assays.

In order to minimize the impact of the coarse nugget effect on result interpretation, an overall sample rolling average head grade for each of the samples tested where used to determine the accountability of each test conducted. The rolling average head grades for each sampler where based on the average of the measured head grade assays and the back-calculated head grade values (based on the assayed concentrate products and tailings streams) for each of the tests conducted, per sample.

It is further noted that the tests used as basis for the recovery correlations, as per Section 13.3.2, all reported acceptable accountability values as illustrated in Table 13-42.

Table 13-42: Test Accountabilities for Recovery Estimate Test work

Ore Type	Test Description (Phase-Sample-Circuit-Test ID)	Pt %	Pd %	Au %	2E + Au %	Cu %	Ni %	Fe %
F-Central	PH1 F4 MF2 New Test 6	10.3	7.0	-1.7	7.5	5.3	5.7	2.9
F-Boundary	PH4 F-Boundary Test 1	-8.1	0.5	18.7	-0.7	-7.7	-3.5	-1.4
F-North	PH3 EDF MF2 T7	-2.4	-2.1	-6.8	-2.4	8.0	5.2	4.0
T-zone	PH4 T2c MF2 T1	-2.4	4.9	2.5	2.1	8.1	11.2	-0.8
Mine Blend	PH1 Blend a MF2 New Test 9	-10.4	-6.1	14.6	-4.4	-3.9	9.7	8.5

Ore Type	Test Description (Phase-Sample-Circuit-Test ID)	Pt %	Pd %	Au %	2E + Au %	Cu %	Ni %	Fe %
	PH1 Blend b MF2 New Test 9	-12.6	-13.3	11.6	-9.7	23.5	-7.7	-2.3

In general, the 2E + Au accountabilities were all within 10% of the samples average head grades. The Au accountability for F-Boundary test used was poor (~18% higher than the sample average). This can result in a 1.5% overstatement of the Au recovery for the F-Boundary material.

The copper value for the Mine Blend b Test 9 was note used in the recovery modelling due to poor accountability.

13.7.3 Recovery Estimate

The recovery estimate derived for the pre-feasibility study as presented in Section 13.3.6, was based on the results achieved from various open circuit tests on individual composite samples from each lithology (excluding the F-South material) and minor work on mine blends were included.

Further testing on different mine blends, as well as the F-South material would be required to better quantify the effect off blending of the different ore types on recovery. Further to this, detailed variability testing would be required to more accurately quantify the expected recovery and highlight what degree of variability could be expected.

Flotation recovery for full scale operations can be lower than that achieved in a laboratory due to operational inefficiencies such as those listed below:

- Variation in ore types/blends
- Power: The laboratory flotation cell power (and air) inputs are extremely high (typically 10 kWh/m ^{3}). This may tend to give higher recoveries due to the improved fines (<20 mm) recovery.
- Milling type: The milling in the laboratory is generally undertaken using rod mills, as opposed to the actual plant, which is often undertaken with ball milling. The difference in particle size distribution between these two types may have an effect on performance.
- Operating conditions: Laboratory operation is undertaken under controlled, 'ideal' conditions. Operational disturbances on full scale operations such as starting and stopping of the plant undoubtedly cause loss of recovery.
- Operational skills: The bench scale laboratory tests are supervised by 'expert' operators. In the actual plant, recovery losses may occur because of bad operational practices.

In order to address as many of these problems as possible the plant design will allow a high level of instrumentation and control within the flotation and milling circuit with the allowance for installation of a mass pull process control system to allow for improved flotation control. Process operators need to be trained and supervised to reduce the occurrence of losses due to bad operational practices.

Variability testing during the feasibility phase will focus on testing to quantify the variability across the deposit and between each of the geo-metallurgical units as identified in the mine plan.

13.8 Comments on Section 13

It is the opinion of the qualified person responsible for Section 13 of the technical report, Mr. Gordon Cunningham, that sufficient test work to support the Waterberg Platinum pre-feasibility has been undertaken. Bench scale test work conducted, on a variety of ore types and blends, has demonstrated that a saleable final concentrate containing at least 80 g/t (4E) can be produced, with 4E recoveries in excess of 80% being expected at the proposed mill feed grades.

The presence of coarse nuggets, most noticeably gold and palladium, has introduced some uncertainty in assaying, although adequate measures have been taken by the metallurgical testing facility and process team to verify and validate metal recoveries. Future ore characterization should aim to shed light on the extent and continuity of the coarse nuggets with specific reference to the palladium, as this would have the greatest impact on the metallurgical recoveries reported.

Extensive metallurgical test work has been conducted on two different flowsheets, namely the MF1 and MF2 flowsheets, with encouraging results obtained from both. Test results have demonstrated that some of the ore types respond better to a particular configuration. Should the mining strategy, and practicality thereof, lend itself to delivering discreet ore types to the process plant then additional flexibility and optimization opportunities could be realized. There is opportunity to take advantage of the lower capital cost and operating cost potential on the MF1 flowsheet, should F-Central material be mined discreetly and processed through one of the concentrator plant modules proposed for the study.

Test work results have indicated that a fair degree of scatter is to be expected around the recovery estimates provided. Future feasibility level test work will aim to describe the metallurgical response variability (on specific ore types and ore blends) across the orebody, with specific reference to comminution and flotation variability.

14. Mineral Resource Estimate

The mineral resource estimates for the T- and F-Zones were updated incorporating the additional and infill drilling since the April 2016 mineral resource estimation. Table 14-1 summarizes the updated Mineral Resources for the F- and T-Zones.

The data that formed the basis for the mineral resource estimate was an exploration database that contained the details of geological logging and assay values derived from a surface drilling programme.

For this estimate, the provided T- and F-Zone wireframes were used as a basis for estimation. The mineralised zones were first identified and labelled as TZ and FZ respectively. The wireframes were created by making a top and bottom pick of mineralization where assays exceeded 1 g/t threshold grade. The wireframes were established by Company geologists and reviewed in detail by the QP. The wireframes were validated by visual inspection vs. the drill holes.

Table 14-1: Summary of Mineral Resources effective October 2016 on 100% Project Basis

T-Zone

Cut-off 4E	Tonnage	Grade							Metal 4E	
		Pt	Pd	Au	Rh	4E	Cu	Ni		
g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
					Indicated					
2	37.788	1.05	1.77	0.76	0.04	3.62	0.16	0.08	136,793	4.398
2.5	31.540	1.13	1.90	0.81	0.04	3.88	0.16	0.08	122,375	3.934
3	23.321	1.24	2.10	0.90	0.04	4.28	0.16	0.08	99,814	3.209
					Inferred					
2	21.865	1.06	1.79	0.77	0.04	3.65	0.16	0.08	79,807	2.566
2.5	19.917	1.10	1.86	0.80	0.03	3.79	0.16	0.08	75,485	2.427
3	13.527	1.24	2.10	0.90	0.04	4.28	0.16	0.08	57,896	1.861

F-Zone

Cut-off		Grade							Metal	
4E	Tonnage	Pt	Pd	Au	Rh	4E	Cu	Ni	4E	
g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	Kg	Moz
					Indicated					
2	292.906	0.91	1.95	0.15	0.03	3.04	0.07	0.16	890,434	28.628
2.5	186.725	1.05	2.23	0.17	0.04	3.49	0.07	0.16	651,670	20.952
3	115.499	1.19	2.53	0.20	0.04	3.96	0.07	0.16	457,376	14.705
					Inferred					
2	164.056	0.83	1.77	0.14	0.03	2.76	0.04	0.12	452,795	14.558
2.5	77.295	1.01	2.16	0.17	0.03	3.37	0.04	0.12	260,484	8.375
3	39.409	1.19	2.55	0.20	0.04	3.98	0.04	0.12	156,848	5.043

4E = platinum Group Elements (Pd+Pt+Rh and Au). The cut-offs for Mineral Resources have been established by a qualified person after a review of potential operating costs and other factors. The Mineral Resources stated above are shown on a 100% basis, that is, for the Waterberg Project as a whole entity. Conversion Factor used — kg to oz = 32.15076. Numbers may not add due to rounding. Resources do not have demonstrated economic viability. A 5% and 7% geological loss have been applied to the indicated and inferred categories respectively.

Based on the available data a mineral resource estimate was undertaken. Prior to declaration of the mineral resource, CJM took into consideration the prospect that the project "has a reasonable prospect for eventual economic extraction" as required by the SAMREC and CIM Codes.

1) Mineral Resources are classified in accordance with the SAMREC standards. There are certain differences with the "CIM Standards on Mineral Resources and Reserves"; however, in this case the QP believes the differences are not material and the standards may be considered the same. Mineral Resources that are not mineral reserves do not have demonstrated economic viability and inferred resources have a high degree of uncertainty.

2) Mineral Resources are provided on a 100% project basis. Inferred and indicated categories are separate. The estimates have an effective date of 17 October 2016. Tables may not add perfectly due to rounding.

3) A cut-off grade of 2.5 g/t 4E (platinum, palladium and gold) for the T-Zone and 2.5 g/t 4E (platinum, palladium, rhodium and gold) for the F-Zone is applied to the selected base case Mineral Resources. Prior to July 20, 2015, a 2 g/t cut-off was applied to resource estimates. For comparison with earlier resources, a 2 g/t cut-off on the updated resource model is presented above. Cut-off grades of 3.0 g/t 4E are also presented as certain mining plans in early years may apply higher cut-offs for the Pre-Feasibility Study.

4) Cut-off grade for the T- and the F-Zones considered costs, smelter discounts, concentrator recoveries from the previous and ongoing engineering work completed on the property by the Company and its independent engineers. Spot and three-year trailing average prices and exchange rates are considered for the cut-off considerations. Metallurgical work indicates that an economically attractive concentrate can be produced from standard flotation methods.

5) Mineral Resources were completed by Charles Muller of CJM Consulting and a NI 43-101 technical report for the Mineral Resources reported herein, effective 17 October 2016

6) Mineral Resources were estimated using Indicator Kriging (IK) for mineralized envelopes and Ordinary Kriging (OK) for grade domains created in Datamine Studio3 from 303 mother holes and 483 deflections.

7) The estimation of Mineral Resources has taken into account environmental, permitting, and legal, title, and taxation, socio-economic, marketing and political factors. The Mineral Resources may be materially affected by metals prices, exchange rates, labor costs, electricity supply issues or many other factors detailed in the Company's Annual Information Form.

8) The following prices based on an approximate recent 3-year trailing average (31 July 2016) in accordance with U.S. Securities and Exchange Commission ("SEC") guidance was used for the assessment of Resources; USD Pt 1,212/oz, Pd 710/oz, Au 1,229/oz, Rh 984/oz - see Cautionary Note.

9) Estimated grades and quantities for by-products will be included in recoverable metals and estimates in the on-going pre-feasibility work. Copper and Nickel are the main value by-products recoverable by flotation and for indicated resources are estimated at 0.16% copper and 0.08% nickel in the T zone 0.07% copper and 0.16% nickel in the F zone.

The data that formed the basis of the estimate are the drill holes drilled by PTM which consist of geological logs, the drill hole collars, the downhole surveys and the assay data. The area where each layer was present was delineated after examination of the intersections in the various drill holes.

There is no guarantee that all or any part of the Mineral Resource that is not converted to Reserves here will be converted to a Mineral Reserve in the future.

14.1 Key Assumptions and Parameters

Generation of the Waterberg Resource was conducted using the following steps:

- Coding of the drill holes to reflect the main mineralised zones,
- Delineate geological/geostatistical domains
- Statistical analysis to provide a basis for data verification and to establish specific information on population distributions and checks for anomalous values,
- Variogram modelling for the grade values,
- Kriging,
- Environmental, permitting and legal, title, taxation, socio-economic, marketing and political factors have been taken into account. The Mineral Resources may be materially affected by metals prices, exchange rates, labor costs, electricity supply issues or many other factors detailed in the Company's Annual Information Form.
- A depth cut-off of approximately 1500 m was implemented based on a preliminary economic assessment.

14.1.1 Data Used

Data used in this estimated comprised 303 original drill holes with 483 deflections. Of these 186 intersections occurred in the T zone ranging from approximately 140m to 1380m in depth below surface. 506 intersections in the F zone were used ranging from approximately 200m to 1500m in depth.

The drill hole file received by CJM Consulting from the Client underwent several main steps before Mineral Resource estimation could be carried out, including:

- Reef coding,
- Compositing with SG weighting,
- Determination of reef cuts,

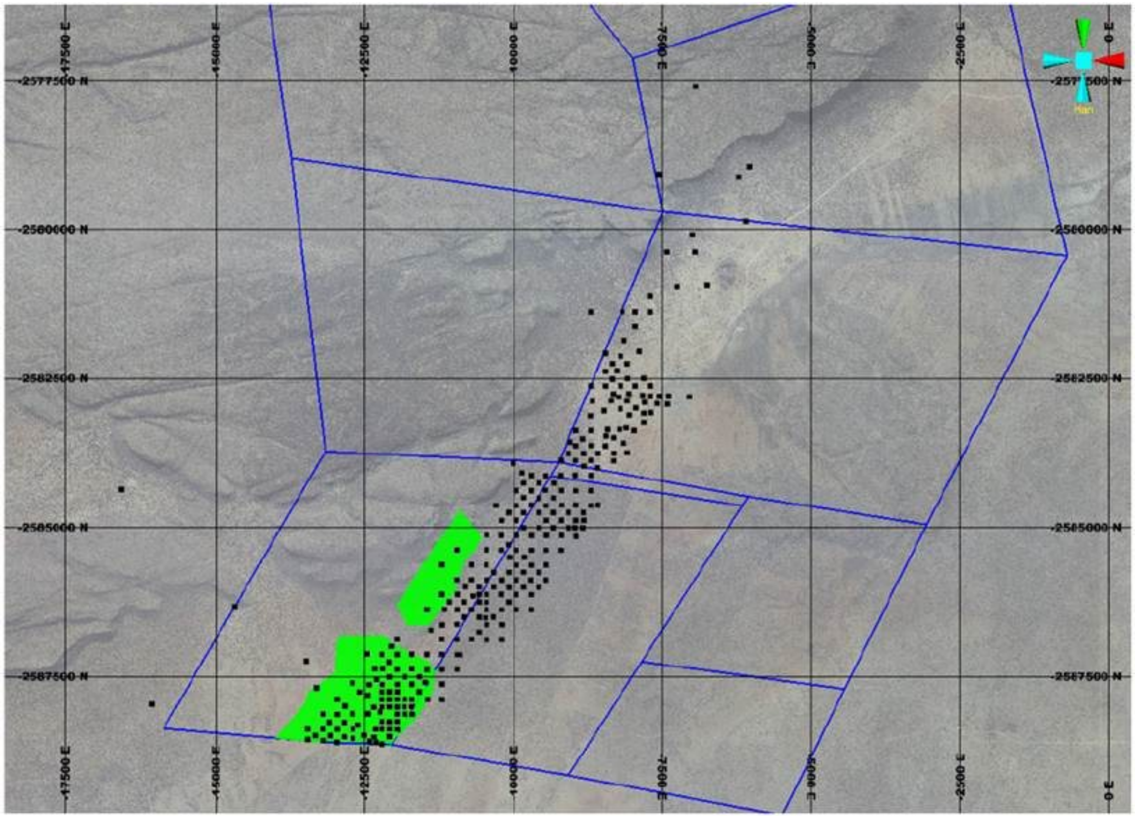


Figure 14-1: Area underlain by the T-Zone

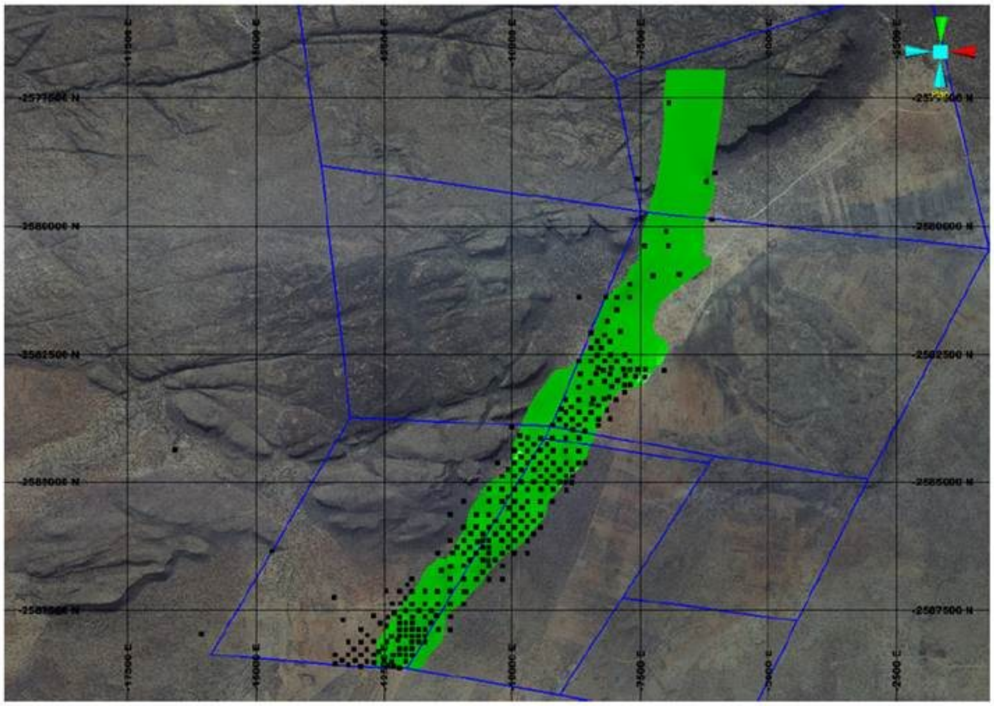


Figure 14-2: Area underlain by the F-Zone

14.1.2 Reef Coding Methodology

As the mineralization is not continuous throughout each of the delineated F and T-Zones and the portions that are mineralised can vary from top to bottom over various distances it was necessary to delineate a mineralised envelope within each zone. In this way poorly mineralised or un-mineralised portions were separated from well mineralised portions. An Indicator Kriging approach was used to estimate the mineralised envelope within each zone. This procedure prevents smearing of high grades into areas, which are not actually mineralised.

Figure 14-3 shows the discontinuous nature of the mineralization. The distribution shown in Figure 14-4 shows if all data is included for a particular F-Zone, the grades are much lower than in reality and are smeared across the higher grade ranges.

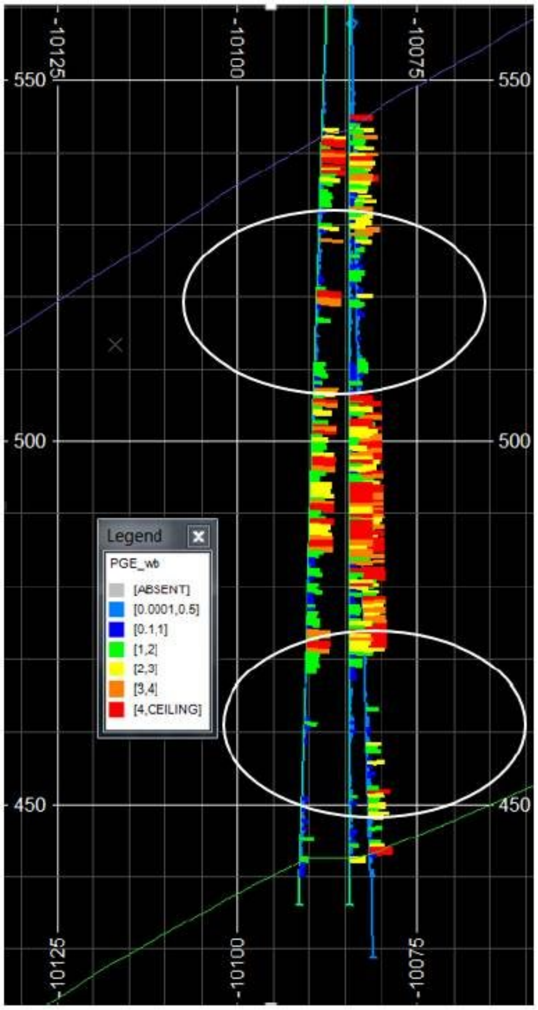


Figure 14-3: Discontinuous Mineralisation in the F-Zone

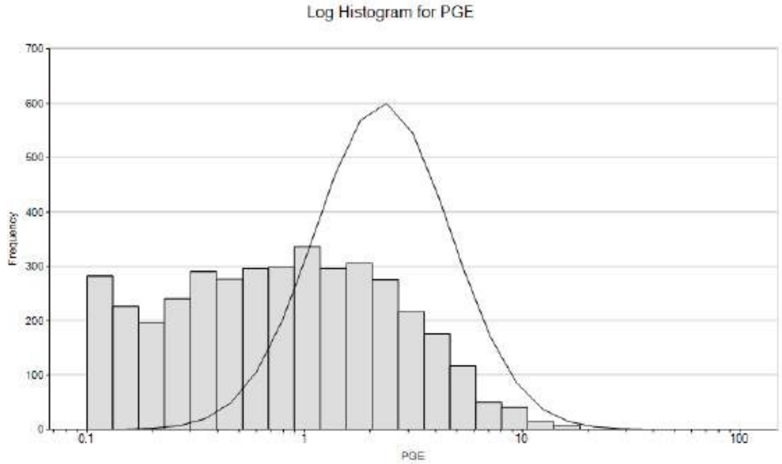


Figure 14-4: Skewed Histogram incorporating Low Grade Data

A 1g/t cut-off grade was selected as representative of the mineralised envelope within each specific F and T-Zones. Up to 2m of waste was included where it could not be separated from the mineralised envelope.

Thereafter, the data was rotated on a horizontal plane in order to improve the sample selection within the defined search ellipses.

Indicator Kriging was then used to delineate mineralized zones within each F and T-Zones. The indicators were estimated in the horizontal plane to produce a probability model. An appropriate probability level was then selected to define the mineralized envelope for final estimation.

The Ordinary Kriged estimate was carried out on a regularized cell size of 25mx25mx1m.

14.1.3 Geological or Geostatistical Domains

The project area consists of distinct zones of mineralization that vary in different parts of the project area. The F-Zone varies from thick (20m – 60m), well mineralised and continuous mineralization (Super F-Zones) to intermediate thickness (10m – 20m) less continuous to thin zones with scattered lower mineralization. The T-Zone is generally thinner (5m – 10m) with higher grades than the F-Zone. There is also a thicker T-Zone of up to 20m and well mineralised and continuous (Super T-Zone).

The following criteria were used for the delineation of the different geological/Geostatistical Domains: -

- Thickness of the total delineated zone (vertical)
- Total metal content within total vertical thickness
- The continuation of grades in the vertical and horizontal direction
- Metal content in the top 20m
- Visual comparison of grade continuity between drill holes on section

Using the criteria above, fourteen geological/geostatistical domains have been delineated for the F-Zone (Figure 14-5) and two domains for the T-Zone (Figure 14-6). For the estimation process, each domain was considered on its own for statistical analysis, variography and kriging.

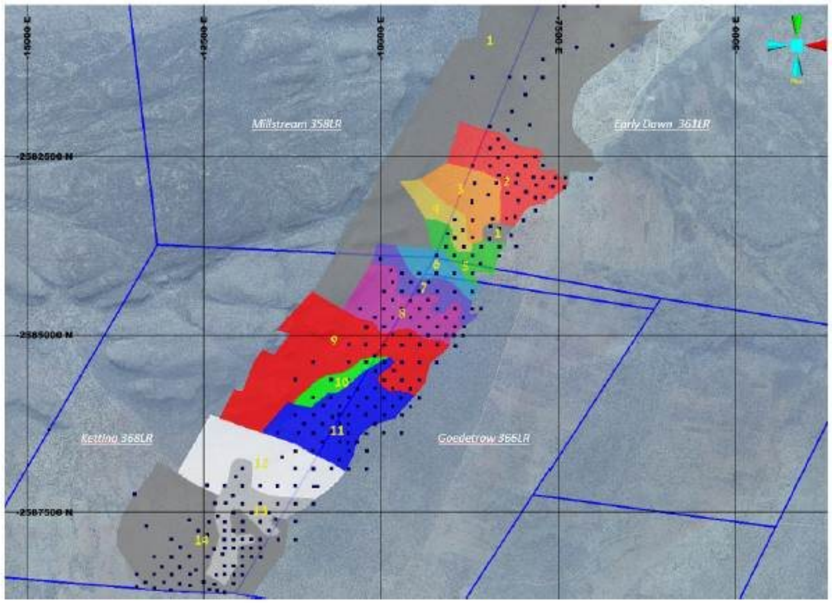


Figure 14-5: Geological Domains for the F-Zone

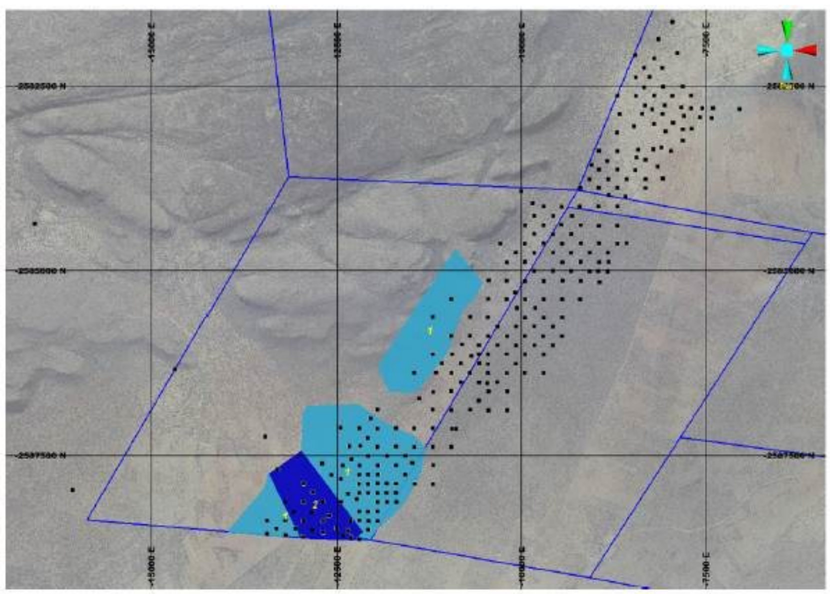


Figure 14-6: Geological Domains for the T-Zone

14.1.4 Density

For the T-Zone a constant density value of 2.8 was assigned based on limited measurements taken to date, and considering the host rock.

For the F-Zone, density was krigged for each block in the model similarly to grade. The average density of the F-Zone was slightly higher than 2.8, and is supported by sufficient data collected using the Archimedes immersion method on site at the core yard. There are cases where density was not measured. As a result, there are some gaps in the data. The gaps were assigned values according to their lithology and an analysis to determine average values for each lithological unit.

The density values are considered by the QP to be appropriate for Bushveld type mineralization.

14.2 Compositing

The drill hole intersections for both the F and T-Zone intersections were composited for PGE+Au, Pt, Pd, Au, Cu and Ni on a 1m interval. The compositing utilized the weighting of density and sample length.

14.3 Descriptive Statistics: Composites

Detailed descriptive statistical analysis was completed based on the total composite data for the mineralised layers. Table 14-2 shows the raw descriptive statistics for the T and F-Zones.

Table 14-2: Descriptive Statistics for the F- and T-Zones

Parameter	Reef	Domain	Records	Min	Max	Av	Var	CoV
3PGE _AU (g/t)	FZ	1	738	0.0303	31.31	1.54	2.7048	1.06
3PGE_AU (g/t)	FZ	2	2545	0.0303	31.47	2.57	4.6046	0.84
3PGE_AU (g/t)	FZ	3	607	0.0303	13.35	1.92	3.8000	1.02
3PGE_AU (g/t)	FZ	4	198	0.0303	15.83	1.43	2.3126	1.07
3PGE _AU (g/t)	FZ	5	1191	0.0303	25.86	2.41	4.1286	0.84
3PGE_AU (g/t)	FZ	6	573	0.0303	7.62	1.73	1.7910	0.77
3PGE_AU (g/t)	FZ	7	198	0.0303	7.35	1.39	1.3704	0.84
3PGE_AU (g/t)	FZ	8	2077	0.0303	17.48	2.07	3.5818	0.92
3PGE _AU (g/t)	FZ	9	402	0.0303	13.67	1.76	2.6417	0.92
3PGE_AU (g/t)	FZ	10	226	0.0303	17.63	1.83	5.6693	1.30
3PGE_AU (g/t)	FZ	11	4612	0.0303	31.74	2.30	5.1014	0.98
3PGE_AU (g/t)	FZ	12	313	0.0303	10.23	1.60	2.2156	0.93
3PGE_AU (g/t)	FZ	13	963	0.0303	25.65	2.47	9.4373	1.24
3PGE_AU (g/t)	FZ	14	553	0.0303	16.41	1.65	2.7012	1.00
2PGE_AU(g/t)	TZ	1	1352	0.0300	23.85	1.62	6.9851	1.63
2PGE_AU(g/t)	TZ	2	2091	0.0030	30.44	2.33	16.849	1.76

The histograms for all of the F and T-Zones show strong positively skew distributions. Care should be taken to apply any linear relationship to these distributions such as a straight average, inverse square distance for estimation etc.

14.4 Outlier Analysis

The histogram and probability plots have been used to determine the values to be top-cut (values greater than the top-cut value are set to the top-cut value) for the various domains. Table 14-3 shows the top-cut values applied for the F-Zone and Table 14-4 for the T-Zone.

Table 14-3: Top-cut Values for the F-Zone

REEF	DOMAIN	PARAMETER	TOPCUT (g/t)
FZ	1	4E	7
FZ	2	4E	12
FZ	3	4E	8
FZ	4	4E	6
FZ	5	4E	12
FZ	6	4E	6
FZ	7	4E	4
FZ	8	4E	14
FZ	9	4E	7
FZ	10	4E	6
FZ	11	4E	17
FZ	12	4E	7
FZ	13	4E	16
FZ	14	4E	9

Table 14-4: Top-cut Values for the T-Zone

REEF	DOMAIN	PARAMETER	TOPCUT (g/t)
TZ	1	2PGE+Au	25
TZ	2	2PGE+Au	35
TZ	3	2PGE+Au	—

Base metals are not top cut due to their minor role in the project's economics.

14.5 Mineral Resource Modelling

14.5.1 Modelling Methodology

Modelling was carried out using Datamine Studio ™ ver21 and Minesoft's geostatistical package 'RES ver4'.

Pt(g/t), Pd(g/t), Au(g/t), Rh(g/t), PGE(g/t), Cu(%), Ni(%) and SG(t/m³) were estimated using Ordinary Kriging techniques. Detailed checks were carried out to validate kriging outputs, including input data, Kriged estimates and kriging efficiency checks.

14.5.2 Variography

Variograms are a useful tool for investigating the spatial relationships of samples. Variograms for 4E (4E Grade) was modelled during the estimation process (Figure 14-7 to Figure 14-9). All variograms are omni-directional spherical semi-variograms.

Type	Variance	Range
Nugget	1.826	-
Spherical	1.039	156
Spherical	1.674	384

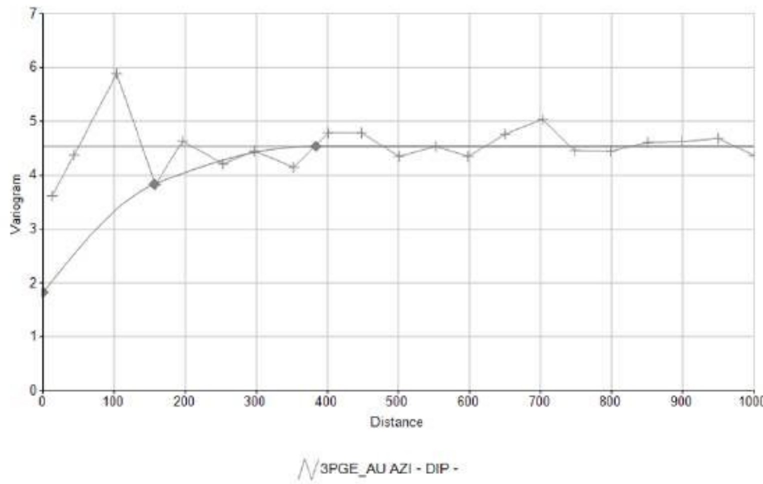


Figure 14-7: Variogram — F-Zone

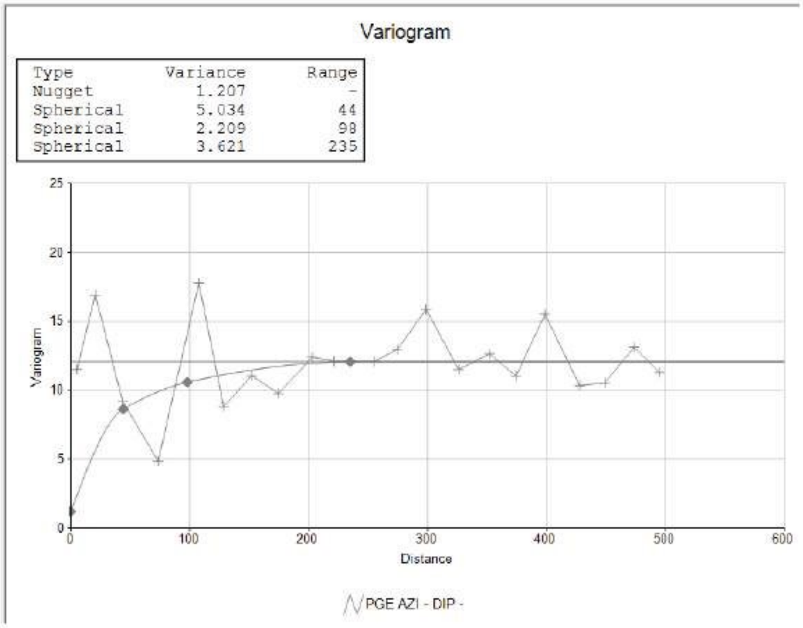
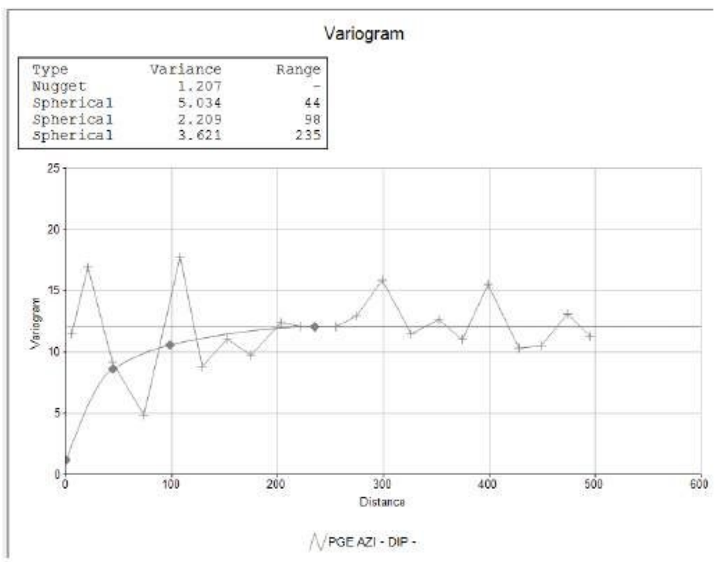


Figure 14-8: Variogram — T-Zone



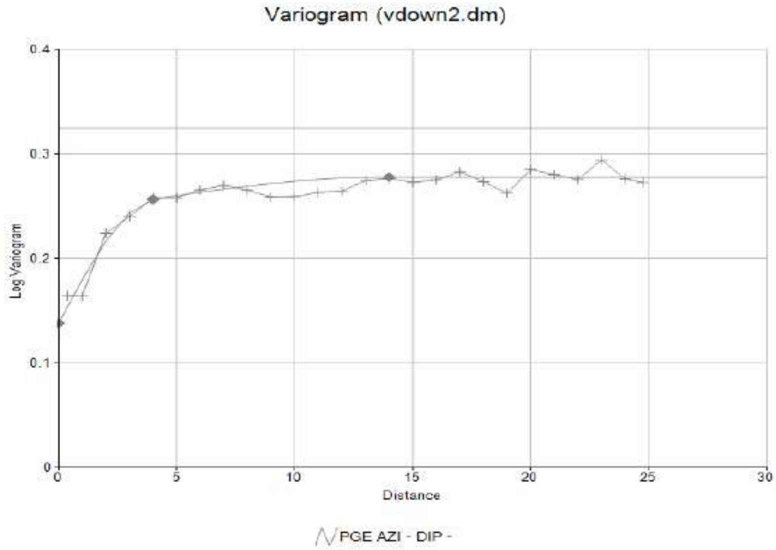


Figure 14-9: Downhole Variogram

14.6 Grade Estimation

14.6.1 Modelling Methodology

The following applies to the Mineral Resource area and was undertaken using Minesoft (Pty) Ltd.'s 'RES' geostatistical program. The following parameters were used in the kriging process for both Project Areas:

- 25m x 25m x 1m block size,
- 3D estimation was conducted
- Search ellipses aligned with the variogram ranges. Search ranges between 200m and 400m.
- Minimum number of Samples 18
- Maximum number of samples 30
- Interpolation methods — Ordinary Kriging,

The following explains the terminology of certain of the parameters that were used in the kriging process:

- ***Search range —*** As range of variogram decreases to approach zero (pure nugget), the required neighbourhood configuration for good estimation will become progressively larger, and vice versa. A limited search range will result in a block estimate that is progressively uncorrelated to the true grade as the variogram range tends to zero. Using the variogram range or slightly larger than variogram range allows the search volume to have a long range relative to the block dimensions, thereby accessing samples particularly in areas of data scarcity.

- ***Discretization* —** Used to divide the block area into many points to allow improved block estimates from point data. The block is divided into many points and then individual point estimates are averaged to get an average over the block. Spatial locality of point data relative to the block to be estimated is hence entertained.

- ***Parent cell estimation*** — When the block model is created, sub-celling of the parent cells is used to allow for an improved representation of the volume. The grade of the parent cell is estimated and that value is assigned to all the (sub) cells inside the parent.

- ***Negative kriging weights*** — at the edges of the ore body / domains, the kriging weights will be small, even negative. The distance required to search before negative weights are encountered progressively increases as the nugget increases. In general, negative weights are not problematic in an estimation model if the number of negative weights is a small proportion, typically less than 2%. Re-setting the negative weights to zero allows conditional bias to be incorporated in the estimation exercise.

14.7 Classification

CJM considers that within the T and F-Zones there are areas that can be classified as Inferred Mineral Resources and others classified as Indicated. The primary criteria differentiating these areas is the spacing of drill hole data, confidence in the kriging estimate which is derived from the kriging efficiencies and regression slope values. Infill drilling has increased the confidence in the structure and the perceived continuity of the layering of mineralization within each Zone. The data is of sufficient quality and the geological understanding and interpretation are considered appropriate for this level of mineral resource classification. The resource was classified according to the criteria below:

- Sampling — QA and QC
  - Measured: high confidence, no problem areas,
  - Indicated: high confidence, some problem areas with low risk,
  - Inferred: some aspects might be of medium to high risk.
- Geological Confidence
  - Measured: high confidence in the understanding of geological relationships, continuity of geological trends and sufficient data,
  - Indicated: good understanding of geological relationships,
  - Inferred: geological continuity not established.
- Number of Samples Used to Estimate a Specific Block
  - Measured: at least eight drill holes within semi-variogram range and minimum of twenty-seven 1m composite samples,
  - Indicated: at least four drill holes within semi-variogram range and a minimum of twelve 1m composite samples,
  - Inferred: less than three drill holes within the semi-variogram range.

- Distance to Sample (Semi-variogram Range)
  - Measured: at least within 60% of semi-variogram range,
  - Indicated: within semi-variogram range,
  - Inferred: further than semi-variogram range.
- Kriging Efficiency
  - Measured: > 60%,
  - Indicated: 20 – 60%,
  - Inferred: < 20%.
- Regression Slope
  - Measured: >90%
  - Indicated: 60 – 90%
  - Inferred: <60%

Figure 14-10 and Figure 14-11 shows the indicated and inferred resource categories respectively.

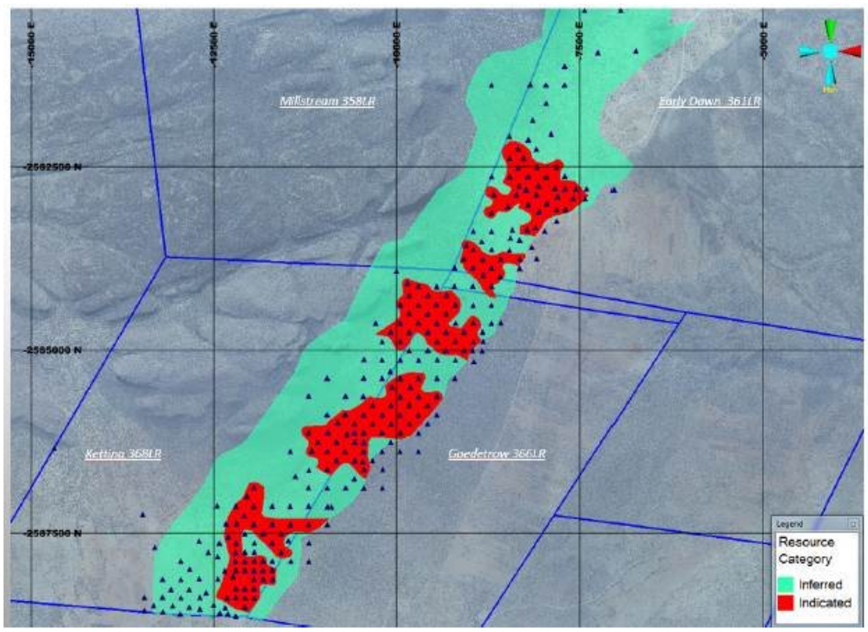


Figure 14-10: F-Zone Mineral Resource Categories

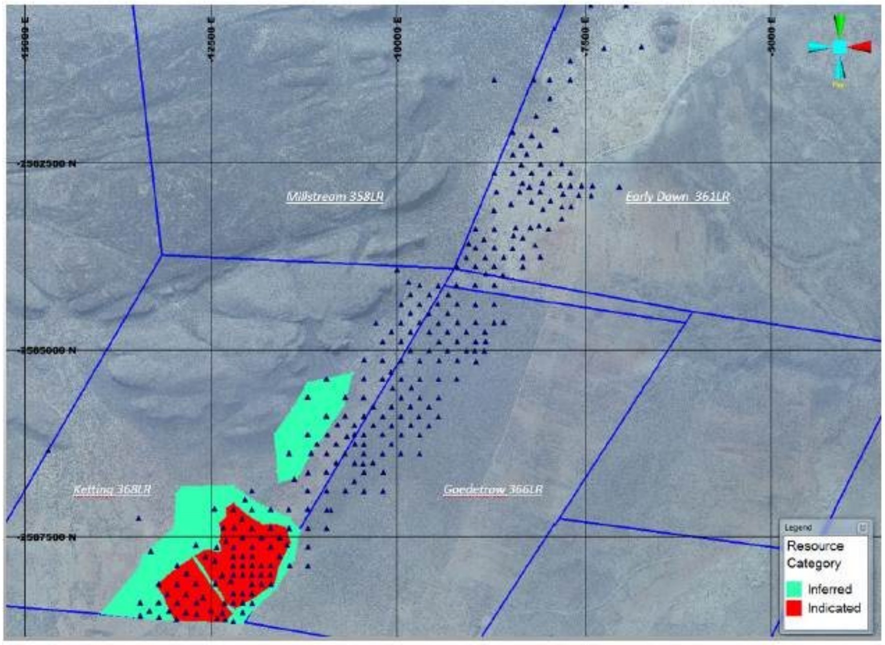


Figure 14-11: T-Zone Mineral Resource Categories

The classification of the mineral resource estimate was underlain in accordance with requirements and guidelines of The South African Code for the Reporting of Exploration Results, Mineral Resources And Mineral Reserves (The SAMREC Code) (2007 Edition as amended July 2009). The reconciliation of the SAMREC Code classification with the CIM Standards (2014) indicates that the criteria for classification and the classes of mineral resource are compatible. The CIM 2014 standard adds a condition that further exploration could reasonably be expected to upgrade the Inferred Mineral Resource to Indicated Mineral Resource. The Mineral Resource reported here meets the requirements of the current 2014 CIM standard.

It should be noted that an Inferred Mineral Resource has a degree of uncertainty attached. No assumption can be made that any part or all of mineral deposits in this category will ever be converted into mineral Reserves.

14.8 Mineral Resource Reporting

Metal contents and block tonnages were accumulated and formed the basis for reporting the mineral resource estimate. The results are presented in Table 14-1.

Mineral Resources, which are not mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

There is no guarantee that all or any part of the Mineral Resource will be converted to a Mineral Reserve.

The independent Qualified Person responsible for the mineral resource estimate in this report is Charles Muller. Mr. Muller is a geologist with some 30 years' experience in mine and exploration geology, resource and Reserve estimation and project management in the minerals industry (especially platinum and gold). He is a practicing geologist registered with the South African Council for Natural Scientific Professions (Pr.Sci.Nat.) and is independent of Platinum Group Metals Ltd as that term is defined in Section 1.5 of the Instrument.

All Mineral Resources have been classified as Indicated and Inferred Mineral Resources, according to the definitions of the SAMREC code and CIM Guidelines for NI 43-101.

Inferred Mineral Resources have been classified. However, no addition of the Inferred Mineral Resources to other Mineral Resource categories has taken place.

14.9 Metal Equivalents

3E (platinum, palladium and gold) and 4E (platinum, palladium, rhodium and gold) estimates of platinum, palladium, gold and rhodium are commonly used in SAMREC Resource estimates. The metal split for the T-Zone is Pt:Pd:Au Rh 29:49:21:1 and the F-Zone Pt:Pd:AuRh:Au 30:64:5:1

14.10 Effect of Modifying Factors

Modifying factors such as taxation, socio-economic, marketing or political factors have been taken into account as disclosed in this report at a Resource assessment level. No environmental, permitting, legal or title factors that are not disclosed will affect the estimated Mineral Resource.

Initial metallurgical, socioeconomic, community, political and metal marketing factors create no known current fatal impediments to the project.

These factors are considered in greater detail at a Reserve consideration level.

The Resources may never be classified as Reserves or be upgraded without a PFS or further exploration.

15. Mineral Reserve Estimates

15.1 Resource to Reserve Calculation

The Mineral Reserve Estimate for Waterberg is based on the block model files provided by PTM. The block model is divided into the two different mineralized zones of the deposit: F Zone and T Zone. Only Indicated material was used to generate the estimate. The block models represent the deposit by means of 5m x 5m x 1m cells that contain density values and an equivalent grade field: 4E. The composition of the 4E field is summarized in Table 15-1.

Table 15-1: Prill Split

	Prill Split					Grade	
	Pt %	Pd %	Au %	Rh %	Total %	Cu %	Ni %
T-Zone	29	49	21	1	100	0.16	0.08
F-Zone	30	64	5	1	100	0.07	0.16

The resource block model was used as an input into the Mineable Shape Optimizer (MSO) software, which identified mineable areas based on grade cut-off, the minimum mining height and geotechnical middling. These results were used as guidance in the detailed design.

To the tonnes and grade evaluated from this design, modifying factors were applied which represent the practical losses and dilution expected when mined and have resulted in the Mineral Reserve Estimate.

15.1.1 Identifying Mineable Areas - MSO

The tool used to identify mineable areas was Mineable Shape Optimizer (MSO). It uses the resource model and the cut-off grade as the main inputs. MSO then takes the block model, slices it at predefined intervals within a specified grid, and then evaluates these slices against the resource model to obtain tonnage and grade. The final step is to group as many slices as possible, while still being above the cut-off grade and within the geotechnical and mining criteria specified.

To initiate the Resource to Reserves process a planning stope pay limit analysis was carried out using the parameters set out in Table 15-2:

Table 15-2: Planning Stope Pay Limit

Inputs	Value	Unit
Pt Price (3yrs avg.)	1212	USD/oz
Pd Price (3yrs avg.)	710	USD/oz
Au Price (3yrs avg.)	1229	USD/oz
Rh Price (3yrs avg.)	984	USD/oz
Basket Price (4E)	899	USD/oz
Total Production Costs	53.33	USD/t
Metal recovery	82	%
Smelter recovery	85	%

Inputs	Value	Unit
Dilution	5	%
Stoping Pay Limit	2.62	g/t

For the purpose of this study a strategy was adopted to initially target only >3.0g/t ore. At the end of the mine life, >2.5g/t ore would also be included.

The parameters used for MSO are shown in Table 15-3.

Table 15-3: MSO Parameters

Parameter	Value	Unit
1. Shape Framework	XY	axis
2. Slice interval	1	m
3. Stope Shape	5x5	m
4. Minimum Mining Height	3	m
5. Minimum Middling	20	m

15.1.2 Shape Framework

The Shape Framework is the orientation in which MSO will carry out its analysis in the 3D coordinate system. There are three options of framework:

- East-West (XZ axis)
- North-South (YZ axis)
- Horizontal (XY axis)

The ore body strikes to the northeast (~30° from North) at an average dip of 40°, therefore the East-West framework is eliminated from the options. The images below show a representative plan and section view of the largest block in F-Zone, F Central.

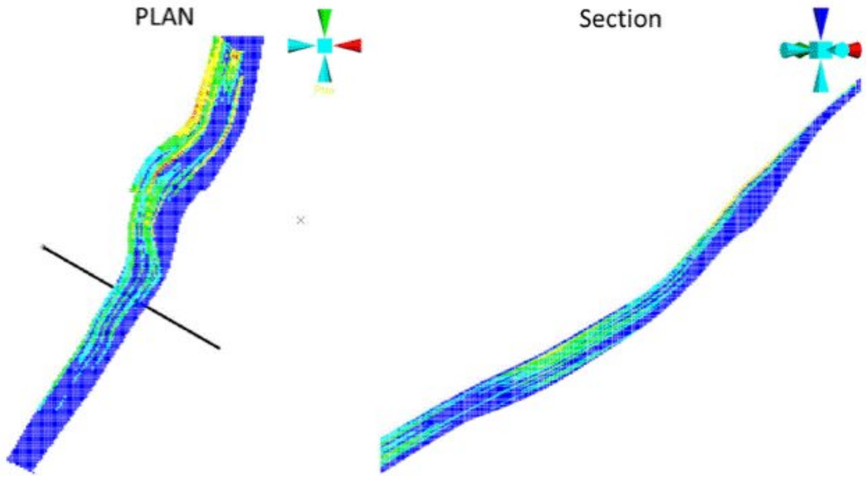


Figure 15-1: Example of Plan and Section for F Central

To make a final decision on which Shape Framework to use, a detailed examination of the blocks was required. This ore body was modelled using 5m x 5m x 1m cells (X,Y,and Z).

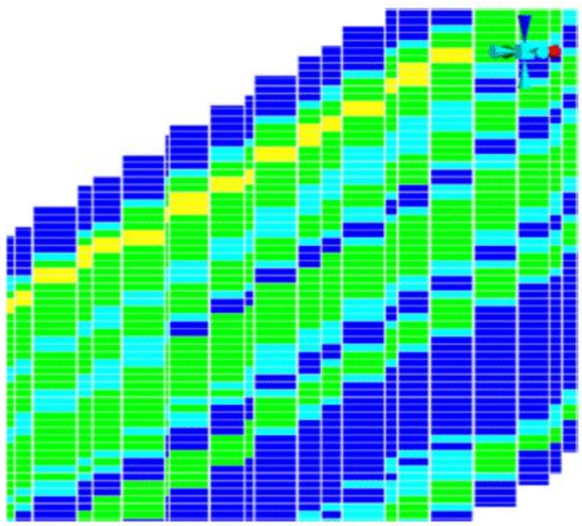


Figure 15-2: Example of Block Model showing Cell Arrangement

Considering the aspect of the ore body and the shape of the cells used to model it, the XY Shape Framework was selected as ideal to provide the most representative results. This Shape Framework will allow MSO to create slices that perfectly enclose the model cells. This will be detailed in the sections to follow.

15.1.3 Slice Interval

The Slice Interval is the spacing that determines the minimum resolution of the MSO analysis. This interval is specified in metres.

The Shape Framework selection in 0 determined that the X and Y dimensions will be dealt with as the stope shape (X and Y, which is explained in the following heading), therefore the Slice Interval will have effect on the Z axis. The cells in the block model are 5m x 5m x 1m, so this determines that the minimum slice interval to use is 1m. There is no gain in accuracy slices taken smaller than the cell size is used.

MSO will benefit from larger Slice Intervals as it will reduce processing time, however to pursuit highest accuracy the minimum Slice Interval of 1m was used.

15.1.4 Stope Shape

The Stope Shape determines the other two dimensions that will complement the Slice Interval, i.e. the X and Y dimensions. MSO, as the name states, is focused on determining mineable shapes; therefore, it is recommended to use practical stope shapes that could potentially be used directly into the mine design. This, however, is not always possible due to the nature of the model and the shape of the ore body, which was the case with Waterberg.

This project uses three different mining methods to economically exploit the resource, and the method is selected based on the dip and thickness of the ore body. Each method has its own shape and size of stopes, making it impossible to determine mineable shapes in a single MSO run. In addition to this issue, cogniscence of the natural orebody shapes, as defined by the drill hole spacing, larger stopes where edge induced dilution was implicit were eliminated, in favor of smaller more orebody hugging designs.

All modifying factors are applied after the design is finished, therefore this modelling dilution would overestimate the expected dilution and the results would be unnecessarily penalized. The image below (Figure 15-3) illustrates the corners of blocks (in green) that breach the planned stope limits (red surface), thus causing misalignment between resource and Reserve estimates.

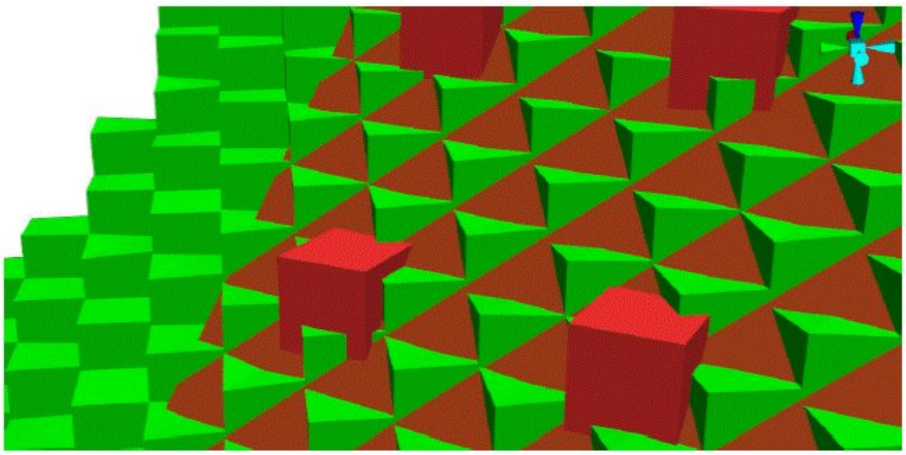


Figure 15-3: Example of Block Model Corners going through the Planned Stope Hanging Wall

To overcome this issue, MSO was run on the smallest possible optimization size to determine mineable packages, which is a 5 x 5m shape. This shape wraps perfectly around the block model cells without any losses, resulting in raw, undiscounted mineable areas and will be aligned with the resource estimates, and to which modifying factors can be applied to estimate the Reserves.

A risk of using the smallest possible shape is that there must be continuity in the results in order to maintain a practical mining scenario. If the variability of the results is too high within a small area, this methodology cannot be taken further. The Sections 15.1.5 and 15.1.6 further describe the purpose and confirm that the continuity of the shapes is well within acceptable range for this level of study.

15.1.5 Minimum Mining Height and Minimum Middling

The minimum vertical mining height is determined by the mining methods being applied. The minimum vertical mining height considered for this study is 3m. This was due to the practical limitations of mechanized mobile equipment, especially when the associated dip is also considered. Therefore, this was the Minimum Mining Height defined in MSO.

The ore body is made up of layered mineralized zones, and a Minimum Middling is required to force MSO to select the largest possible mining package. This Minimum Middling was set to 20m as recommended by the geotechnical model, so MSO only selected adjacent mining areas if they were 20m apart vertically.

15.1.6 Results

MSO generates results in the form of graphical wireframes, strings and tabulated results. The key values in the mineable areas identified by MSO are:

- Coordinates of mineable area
- Mining height
- Tonnage
- PGE value

The MSO results are well aligned to the Resource estimates prior to geological losses. The divergence in the MSO results are derived from not meeting the requirements specified, e.g. Minimum Mining Height and Minimum Middling.

To interpret the coordinates and the mining height of the results one must observe the wireframe output. The image below illustrates each Zone separately in plan view.

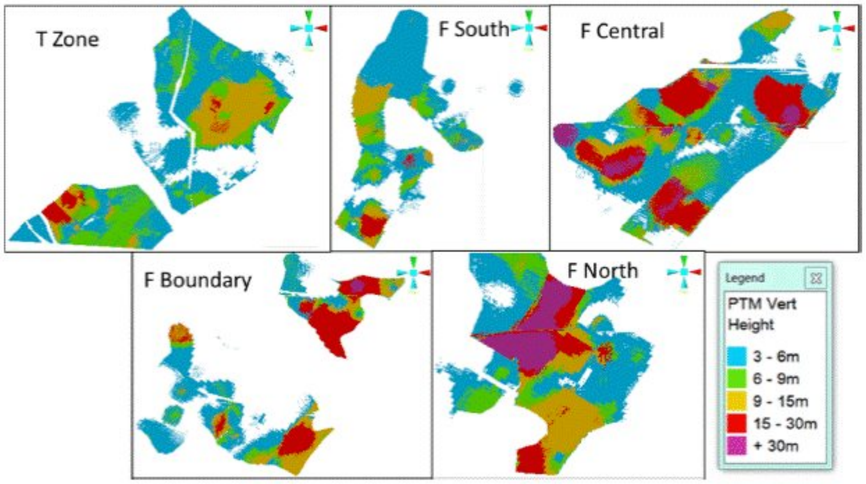


Figure 15-4: MSO Results for Each Area coloured on Vertical Height (not to scale)

These results form the foundation of the mine design. They are to be used to easily identify areas of certain ore thickness (vertical height) and combined with dip of the ore body can delineate areas that will have specific mining methods.

When observing a section of the MSO results it is possible to get a better understanding on how the optimizer works. The following image shows an East-West section of the F-North results. The block model is loaded with transparency and colored on PGE grade to highlight the 1m high cells, and the MSO results are shown as outlines that contour certain blocks in order to form mineable shapes. These shapes have been colored based on Vertical Height to assist with the interpretation of results.

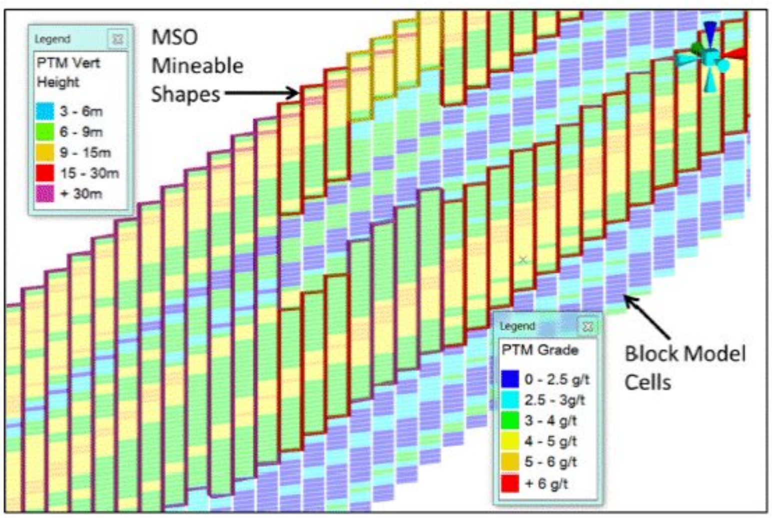


Figure 15-5: Section of MSO Shapes and the Block Model

To the East of this section (to the right of the image) the MSO shapes are divided into two potentially minable stopes with a 20m minimum middling. As the ore body gets deeper, to the West, MSO makes the decision to combine the two seams into one massive shape (+30m vertical thickness). This section also shows the complexity of the mineralization, emphasizing the importance of an optimization tool to provide a basis for the mine design.

15.1.7 Design

The wireframes resulting from the MSO runs were used to create artificial footwall and hanging wall contact zones from which the mine design could be digitized. This was done by linking the horizontal centroids of the MSO-identified shape outline.

Three mining methods were identified to cover the mineable areas of the resource: Blindhole Longitudinal Retreat (BLR), Longitudinal Sub-level Open Stoping (LSLOS) and Transverse Sub-level Open Stoping (TSLOS).

The criteria for each of these methods will be detailed in Section 16, but can be summarized by Table 15-4.

Table 15-4: Selection Criteria for Mining Method

Mining Method	Dip	Vertical Thickness
BLR	≤ 35°	3 - 15m
LSLOS	> 35°	3 - 15m
TSLOS		> 15m

The MSO wireframes provided the boundaries to which each mining method is applied. These boundaries along with the artificial contact zones were used in Studio 5D Planner to create the detailed mined design.

The design maximized the recovery of material identified from MSO while honoring the geotechnical guidelines proposed by rock engineering, all geotechnical losses therefore were designed for and would not require additional factors.

To obtain initial tonnage and grades, the mine design was evaluated against the block model and the results were exported to Enhanced Production Scheduler (EPS) for scheduling and reporting.

15.1.8 Modifying Factors

Along with the intentional geotechnical losses that were designed for (pillars), the *in-situ* tonnage and grade from the design have to be modified in order to accurately represent realistic mining practices. The following Modifying Factors were applied to the design results:

- Mining Geological Losses
- Stope Overbreak
- Other Mining Losses

15.1.8.1 Mining Geological Losses

Geological Losses are applied both to the Resource and to the Reserve results, however care was exercised that they are not double accounted and therefore there is no duplication of geological losses.

The Resource is extracted from the Block Model and then discounted by 5% geological losses to derive to the final result. The Mineable Reserves are also extracted from the same Block Model, thus the need to apply the same geological loss in order to ensure consistency between the results. This 5% Mining Geological Loss was applied to the ore tonnes.

15.1.8.2 Stope Overbreak

Stope overbreak is used to account for additional tonnage that will be blasted with every stoping excavation. Overbreak in general has been considered in both development and stoping. Development overbreak has been designed for by increasing both the height and width of the excavations by 5%, and therefore does not require additional factoring for overbreak. This section will focus on the stope overbreak that has been applied to the mined tonnes in the form of a modifying factor.

Contrary to the development overbreak that has been physically designed for, the Stope overbreak has been estimated for each individual mining method and for each Zone based on the average thickness and dip of the stopes.

The average dip and thickness of each mining method per Zone was estimated and used to design a typical section for that particular case. From this report section, the stope hanging wall and footwall were expanded by a certain distance to estimate the effect of the overbreak. The expansion distance was determined by the thickness of the ore body. For true thickness up to 6m, the overbreak was applied at 0.3m on the hangingwall and similarly the footwall. For true thickness greater than 6m, the overbreak was estimated at 0.5m on the hanging wall and the footwall. This difference in overbreak thickness assigned is due to the drilling pattern accuracy for different stope thicknesses. Stopes larger than 6m thick require ring drilling that leads to more overbreak when compared to the parallel drilling used in smaller stopes.

The image below shows an example from the Longitudinal Sub-level Open Stoping areas of F North, orange represents the ore and the pink edges show the designed overbreak. This area has a thickness of 10.8m therefore the pink overbreak was designed at 0.5m on each side. At the bottom, one can observe a green margin representing the ballast loss, which will be discussed in the following headings.

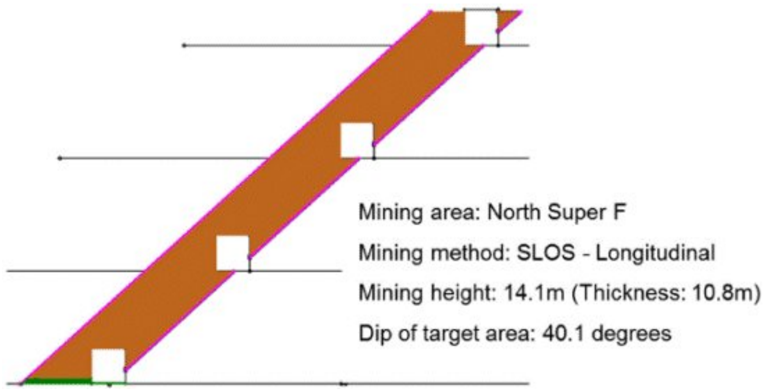


Figure 15-6: Example of overbreak design

The area of this expansion along with all other areas was then measured in the design software.

Table 15-5: Measurements for overbreak calculation

Method	Vert Thickness (m)	Avg Dip °	Overbreak (m)
LSLOS	14.1	40.09	0.5

Planned stope	931.6	m²
Ballast Loss	14.4	m²
Overbreak	93.0	m²

These results were used to determine the percentage of dilution in relation to the ore tonnes for each case.

The results of the Stope overbreak estimation are shown in Table 15-6.

Table 15-6: Final Stope Overbreak factors

Area	Factor		Stope Overbreak Tonnes
T-Zone	BLR		15.5%
	SLOS	Longitudinal	13.3%
		Transverse	7.5%
South	BLR		16.2%
	SLOS	Longitudinal	16.4%
		Transverse	7.2%
Central	BLR		15.3%
	SLOS	Longitudinal	11.9%
		Transverse	6.1%
Boundary	BLR		15.4%
	SLOS	Longitudinal	16.0%
		Transverse	5.9%
North	BLR		14.4%
	SLOS	Longitudinal	10.0%
		Transverse	3.7%

Due to the mineralization of the material hosting the stopes, a separate exercise was done to estimate what grade would be applied to the overbreak dilution.

This exercise consisted of translating the current stope design vertically in both directions by 0.5m and evaluating the results against the block model. The results provide an estimate on the effect of the grade when the stopes are shifted up or down. Since the overbreak is applied to both the hanging wall and the footwall, the effect on grade must be combined to obtain the estimated final dilution.

Table 15-7: Estimated Grade Dilution for Overbreak

Dilution Estimation

Case	Dilution
T-Zone	2.1%
F-South	2.2%
F-Central	0.7%
F Boundary	1.1%
F-North	0.3%

15.1.8.3 Other Mining Losses

Additional factors have to be considered to accurately estimate losses in the stopes, mainly the angle of repose loss, the cleaning loss and the ballast loss.

15.1.8.3.1 Repose Loss

The repose loss occurs due to the natural angle of repose of the broken ore. This angle is estimated to be 38°. If the dip of the stope is less than the angle of repose, there will be losses due to material accumulating on the footwall.

This loss was calculated using a similar method as the overbreak loss. The repose loss is designed where applicable and the area ratio is used to establish a percentage factor to be applied as depicted in Figure 15-7.

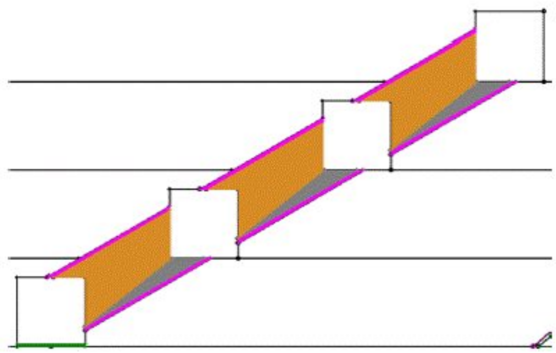


Figure 15-7: Repose Loss being represented in grey

15.1.8.3.2 Ballast Loss

Ballast loss refers to the material that is left behind in the footwall of the excavations in order to provide a smooth surface on which machinery can operate. Ballast loss has been assumed 0.3m from the footwall of the excavation.

15.1.8.3.3 Lock-up Loss

Loading in the stopes will be accomplished via remotely controlled LHDs and reduces the ability to load efficiently; therefore, a lock-up loss was applied to the stopes. This loss has been estimated at 1m only at the bottom level of the stope panel.

Figure 15-6 from a previous heading shows the ballast loss and lock-up loss in both development and stoping in green.

15.1.8.3.4 Footwall Loss

The footwall loss is considered in cases where there is no repose loss. It is the small accumulation of material on the footwall of the stope, even if the dip is greater than the angle of repose. It has been estimated by considering that ore will form a 15cm layer on the footwall.

All factors have been combined into a single category.

Table 15-8: Other Mining Losses factor

Area	Factor		Other Mining Losses
T-Zone	BLR		11.6%
	SLOS	Longitudinal	2.4%
		Transverse	1.5%
South	BLR		11.8%
	SLOS	Longitudinal	2.9%
		Transverse	1.5%
Central	BLR		13.2%
	SLOS	Longitudinal	3.4%
		Transverse	1.4%
Boundary	BLR		11.9%
	SLOS	Longitudinal	3.1%
		Transverse	3.5%
North	BLR		10.4%
	SLOS	Longitudinal	2.7%
		Transverse	1.2%

15.1.8.4 Grade Adjustment Factor

The Waterberg orebody dips between 35° and 40° on average. Since the model is made of 5m x 5m x 1m cells, the hanging wall and footwall contact zones do not have the required resolution to align with a practical stope shape. When observing a section of the model it is possible to observe a “staircase effect” where ore is excluded and waste is included in the design:

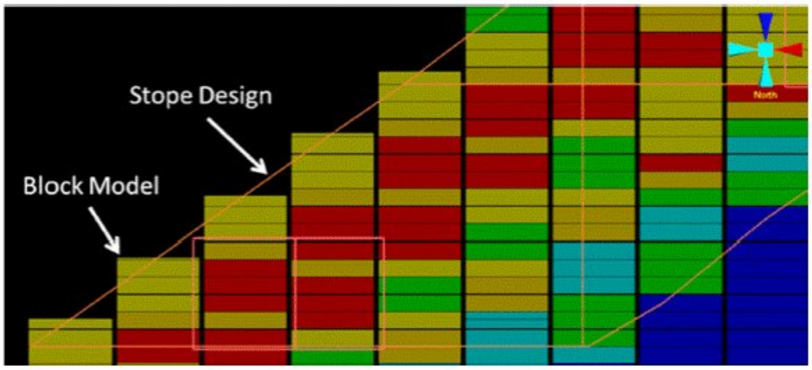


Figure 15-8: Misalignment between Block Model and Stope Design

Grade Adjustment Factors were calculated and applied to all the stoping tonnes to account for this phenomenon. The results are tabulated in Table 15-9.

Table 15-9: Grade Adjustment Factor due to Loss and Excess Overbreak from MSO run

Grade Adjustment Factor

Area	Original 4E	Factor	Adjusted 4E
F South	3.94g/t	10.76%	4.36 g/t
F Central	3.44 g/t	9.3%	3.76 g/t
F Boundary	3.73 g/t	5.7%	3.94 g/t
F North	3.70 g/t	6.6%	3.94 g/t
T Zone	4.07 g/t	11.7%	4.55 g/t

15.1.8.5 Summary of Modifying Factors

All modifying factors applied to the mine design are shown in Table 15-10.

Table 15-10: All Modifying Factors

Area	Factor		Mining Geological Losses	Stope Overbreak Tonnes	Mining Loss (Ballast, Cleaning)	Grade Adjustment Factor
T-Zone	BLR		5%	15.5%	11.6%	11.7%
	SLOS	Long	5%	13.3%	2.4%	
		Trans	5%	7.5%	1.5%	
South	BLR		5%	16.2%	11.8%	10.8%
	SLOS	Long	5%	16.4%	9.251%	
		Trans	5%	7.2%	5.675%	
Central	BLR		5%	15.3%	13.2%	6.6%
	SLOS	Long	5%	11.9%	10.758%	
		Trans	5%	6.1%	9.251%	
Boundary	BLR		5%	15.4%	11.9%	5.7%
	SLOS	Long	5%	16.0%	6.604%	
		Trans	5%	5.9%	10.758%	
North	BLR		5%	14.4%	10.4%	9.3%
	SLOS	Long	5%	10.0%	2.7%	
		Trans	5%	3.7%	1.2%	

Each factor has been applied to the corresponding stoping methods and areas to achieve a representative Reserve Estimate.

15.1.9 2.5 g/t 4E Cut-off Ore

Once the mine has depleted its high grade reserves, there is potential to recover some of the lower grade Indicated Resources between 2.5g/t and 3g/t in F Zone, and deeper T resources. A study was carried out to quantify this potential. All the material that has been estimated in this portion of the study has a lower level of mine design when compared to the +3g/t material due to the fact that a detailed design and schedule has not been compiled for it. The detailed mine plan schedule for this material has small value impact since it is at the end of the mine life. This ore is considered part of the reserves and represents approximately 11% of the total 2.5 g/t 4E cut-off reserve.

15.1.9.1 T Zone

MSO was re-run using the same parameters as the 3g/t optimizations, but at the reduced cutoff of 2.5g/t. The results were fenced around the current design to result in the total upside potential for additional ore reserves in T Zone.

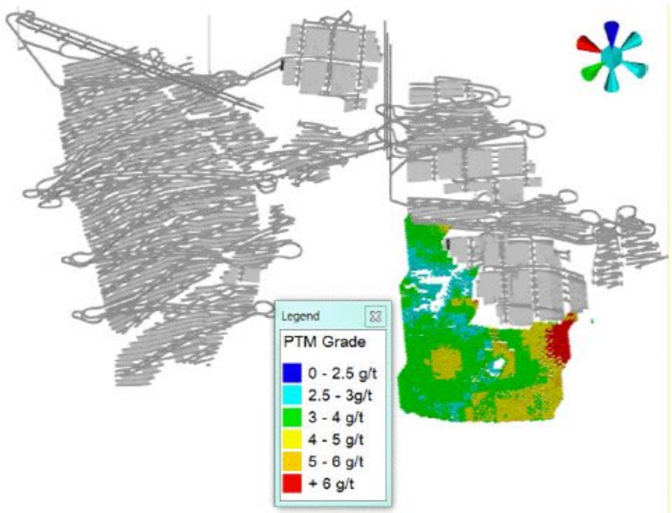


Figure 15-9: Additional Indicated Material for T-Zone (not to scale)

All areas were deemed as economical to mine as they consist of a consolidated area, as opposed to isolated mining blocks. A design factor was applied to account for the dilution that would normally occur from an MSO result to a complete stope design. In the image below, it is possible to see a few empty spaces in the middle of the MSO results. These voids would be mined through in practice, which would result in additional tonnes, thus the need for such an adjustment factor.

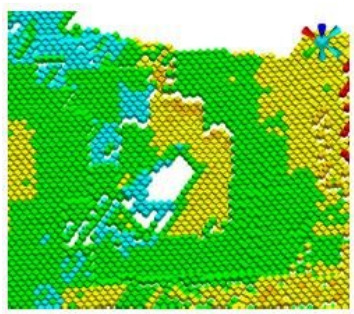


Figure 15-10: Void Material that will be factored into the Results (not to scale)

To determine this factor, a perimeter was defined around the entire area from the MSO results, which included the void areas. The area of the perimeter was calculated and multiplied by the average height of the MSO stopes for that particular location. This resulted in the total volume estimated from that area (by means of a full, continuous solid). This volume was compared to volume inside the MSO blocks within the same perimeter. The design factor was calculated from the difference between these two volumes.

This design factor was calculated to be 5% for T Zone. These additional tonnes are added at zero grade to maintain the same metal content. This is in reality a conservative approach; given that most of the material surrounding the mining blocks does present some grade that will result in additional metal content.

This potential material was classified by Mining Methods, and the modifying factors were applied as per the 3g/t material (pillar loss, geoloss, overbreak and mining loss).

15.1.9.2 F Zone

The 2.5g/t cut-off ore reserves for F Zone were estimated on the same Indicated resource model used for the 3g/t reserve estimation. To establish a target area for the upside, precautions were taken to ensure that no overlaps would occur with the 3g/t design, which would result in double-accounting of reserves.

An artificial footwall was created at the base of the 3g/t design, and this surface was translated vertically down by 20 meters. This would be the battery limit of this exercise for F Zone. Anything above this surface was considered conflicting with the 20 meter middling required by the geotechnical studies and therefore was excluded from the exercise.

The remaining model below the artificial surface was submitted to an MSO run at 2.5g/t cut-off. The results are shown below:

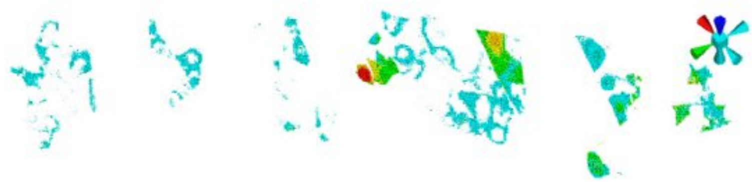


Figure 15-11: F-Zone 2.5g/t Late Stage Indicated Material (not to scale)

This potential resulting from MSO was then visually analyzed to assess how economical certain areas are to be practically mined, i.e. all isolated blocks were removed. Below is an enlarged image of examples (inside the circles) of areas being defined as uneconomical. The marked area has been removed from the study. Naturally, this exercise suggests that additional information (drill hole intercepts) may assist in further clarification of these areas.

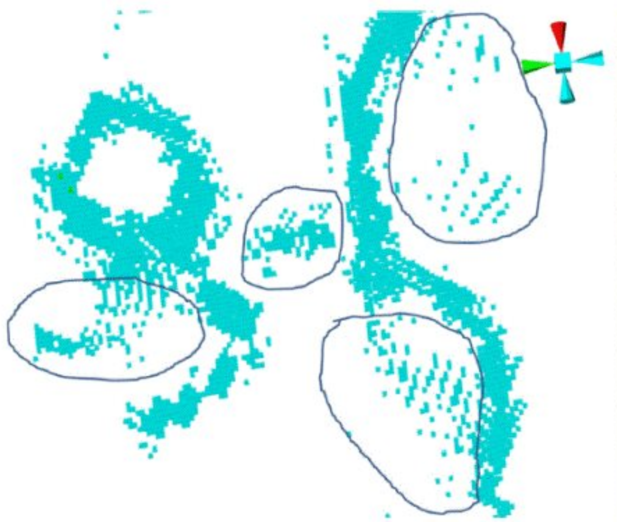


Figure 15-12: Uneconomical Areas removed from the Exercise (not to scale)

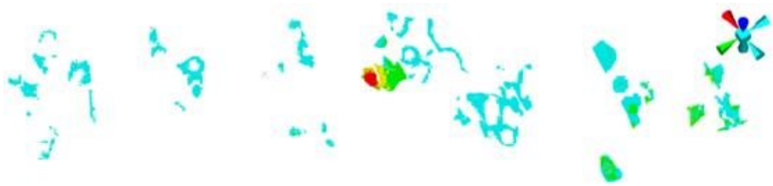


Figure 15-13: F-Zone 2.5g/t Remaining Economical Material (not to scale)

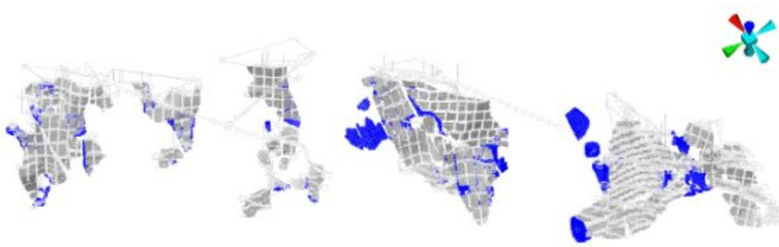


Figure 15-14: Mineable Portion of F-Zone 2.5g/t Late stage Indicated Material (not to scale)

The F Zone results present a higher density of voids in-between mining areas when compared to T Zone, therefore a design factor of 7.5% was obtained for the F Zone blocks, which is slightly higher than the factor determined for T Zone. The modifying factors were applied as per the 3g/t material (pillar loss, geoloss, overbreak and mining loss). The results are shown in the table below:

15.2 Basic Mining Equation

The Resource has been modified into a Reserve in the process shown in the tables below.

Table 15-11: Basic Mining Equation for T-Zone at 2.5 g/t 4E Cut-off

Description	Tonnes	4E (g/t)	Content (kg)	Content (Moz)
Geological Resource at 2.5g/t (no Geoloss)	33 200 000	3.88	128 816	4.14
- < 20m Middling and < 3m Mining Height	3 818 864	3.63	13 854	0.45
+ Internal Low Grade < 2.5g/t	862 197	2.35	2 027	0.07
- Uneconomical Areas	5 707 087	2.70	15 427	0.50
- Estimated Pillar loss (from Blueprints)	7 804 346	4.46	34 809	1.12
- Geological Loss (5% of Extractable Resource)	836 595	3.99	3 338	0.11
+ Stope Overbreak	1 001 527	3.36	3 361	0.11
- Stope Mining Loss	394 799	4.26	1 680	0.05
Mineable Reserve at 2.5g/t	**16 502 033**	**3.94**	**65 097**	**2.09**

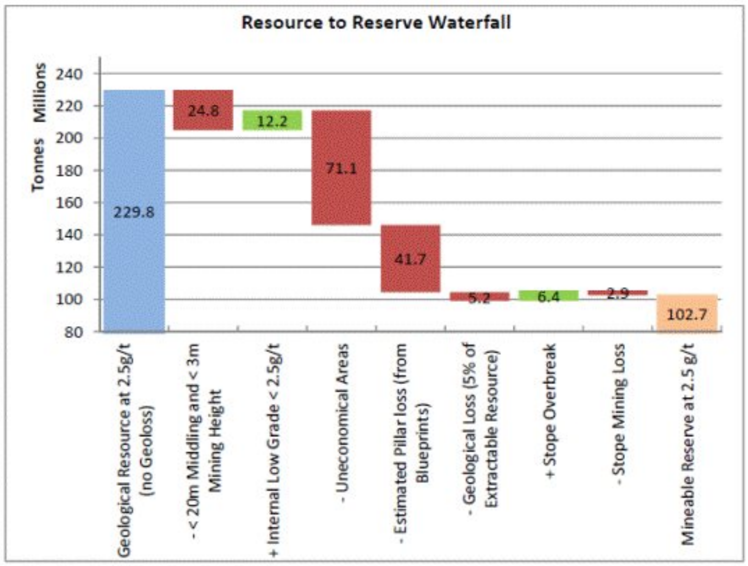
Table 15-12: Basic Mining Equation for F-Zone at 2.5 g/t 4E Cut-off

Description	Tonnes	4E (g/t)	Content (kg)	Content (Moz)
Geological Resource at 2.5g/t (no Geoloss)	196 552 632	3.49	685 969	22.05
- < 20m Middling and < 3m Mining Height	21 010 501	3.40	71 471	2.30
+ Internal Low Grade < 2.5g/t	11 351 806	2.60	29 566	0.95
- Uneconomical Areas	65 414 828	2.82	184 301	5.93
- Estimated Pillar loss (from Blueprints)	33 866 880	3.97	134 535	4.33
- Geological Loss (5% of Extractable Resource)	4 380 250	3.71	16 260	0.52
+ Stope Overbreak	5 422 626	3.31	17 937	0.58
- Stope Mining Loss	2 489 893	3.57	8 897	0.29
Mineable Reserve at 2.5g/t	**86 164 711**	**3.69**	**318 007**	**10.22**

Table 15-13: Basic Mining Equation for the Mine Total at 2.5 g/t 4E Cut-off

Description	Tonnes	4E (g/t)	Content (kg)	Content (Moz)
Geological Resource at 2.5g/t (no Geoloss)	229 752 632	3.55	814 785	26.20
- < 20m Middling and < 3m Mining Height	24 829 364	3.44	85 326	2.74
+ Internal Low Grade < 2.5g/t	12 214 002	2.59	31 593	1.02
- Uneconomical Areas	71 121 916	2.81	199 728	6.42
- Estimated Pillar loss (from Blueprints)	41 671 226	4.06	169 344	5.44
- Geological Loss (5% of Extractable Resource)	5 216 845	3.76	19 598	0.63
+ Stope Overbreak	6 424 153	3.32	21 298	0.68
- Stope Mining Loss	2 884 692	3.67	10 578	0.34
Mineable Reserve at 2.5g/t	**102 666 744**	**3.73**	**383 103**	**12.32**

The waterfall chart illustrates the progression of tonnes from Resource to Reserve.



15.3 Mineral Reserve Statement

The Probable Mineral Reserve at the 2.5 g/t 4E cut-off grade has been tabulated in Table 15-14.

Table 15-14: Probable Mineral Reserve at 2.5 g/t 4E Cut-off (effective date 17 October 2016)

Zone	Mt	Moz	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)
T Zone	16.50	2.09	1.14	1.93	0.83	0.04	3.94
F South	10.32	1.26	1.14	2.42	0.19	0.04	3.78
F Central	36.75	4.24	1.08	2.30	0.18	0.04	3.59
F Boundary	16.08	1.94	1.12	2.40	0.19	0.04	3.75
F North	23.02	2.79	1.13	2.42	0.19	0.04	3.78
Total	**102.67**	**12.32**	**1.11**	**2.29**	**0.29**	**0.04**	**3.73**

16. Mining Methods

16.1 Introduction

Three mining methods are being considered.

Mining zones included in the current Waterberg mine plans occur at depths ranging from approximately 170m to approximately 350m below surface.

Access to the mine will be via multiple decline shafts. Mining will be performed using highly productive mechanized methods.

The initial conversion to mineral reserves was undertaken at 3.0g/t 4E stope cut-off grade for both for the T and the F-Zone reefs, which considered costs, smelter discounts, concentrator recoveries from the previous and ongoing engineering work completed on the property by the Company and its independent engineers. Spot and three-year trailing average prices and exchange rates are considered for the cut-off considerations. The final reserve cut-off for the life of mine was completed on a 2.5 g/t cut-off.

Three mining methods are being considered, namely:

- Blind Longitudinal Retreat (BLR)
- Sub-level Open Stoping (SLOS) — Transverse
- Sub-level Open Stoping (SLOS) — Longitudinal

From the mineral resource as estimated in this report, each stope has been fully diluted, comprising of a planned dilution and additional dilution for all aspects of the mining process.

The effective date for the Mineral Reserve Estimate contained in this report is 17 October 2016. This has been compiled based on the Mineral Resource Estimate dated 17 October 2016.

Only Indicated Resources have been used for determination of the Probable Mineral Reserve.

16.2 Geotechnical Investigation for Surface Infrastructure

The primary aim of the PFS-level geotechnical investigation was to evaluate the geotechnical conditions present, assess the general suitability of the site for the planned development and recommend further work for the proposed development.

The following work programme was carried out:

- Site visit and evaluation of the geotechnical conditions present, assess the general suitability of the site and to make recommendations for foundations and site works for the proposed infrastructure elements.
- Provide preliminary design recommendations for the proposed portals and to comment on geotechnical factors that would have an impact on the overall stability of the sidewalls and highwalls to enable economic design and construction of the proposed access portals.
- To identify relevant ground-related features and determine the variability of ground conditions and the effect of such variability on the proposed designs.

The following methodology was adopted:

- Review of available geological records and site plans.

- Undertaking a geotechnical site inspection of geological boreholes drilled near the suggested portals positions, investigate soil/rock strengths/capacities and identify potential problem soils on site.
- Conducting soil and rock laboratory tests to establish preliminary geotechnical and design parameters of the soils and rock units impacting the infrastructure elements.
- Identification of relevant ground-related features and their influence on the proposed portals design.
- Undertaking of a geotechnical site investigation including Tractor-Loader-Backhoe (TLB) excavated trial pits to profile soils, investigate soil strengths/capacities, and identify potential problem soils on site and core inspection of drilled boreholes.
- Undertaking of laboratory tests, in situ testing and Dynamic Cone Penetrometer (DCP) tests to establish geotechnical and design parameters of the soils and rocks.
- Identification of relevant ground-related features and their influence on the proposed development.

16.2.1 Site Geology

The geological map of Pietersburg (sheet no. 2328, scale 1:250 000, published in 1985, Copyright Council for Geoscience) shows the area to be covered by Quaternary transported material (Q) composed by soil, sand, alluvium, calcrete and scree. The geological map also show the site to be underlain by medium-grained, yellowish, laminated sandstone from the Makgabeng Formation (Mma) and by coarse-grained dark-colored sandstone, minor conglomerate, arkose, tuff and mudstone from the Setlaole Formation (Mse). The Makgabeng and Setlaole Formations belong to the Waterberg Group. An excerpt of the geological map, with the site and the major geological groups is given in Figure 16-1. Table 16-1 shows the legend of the geological materials present in the studied area.

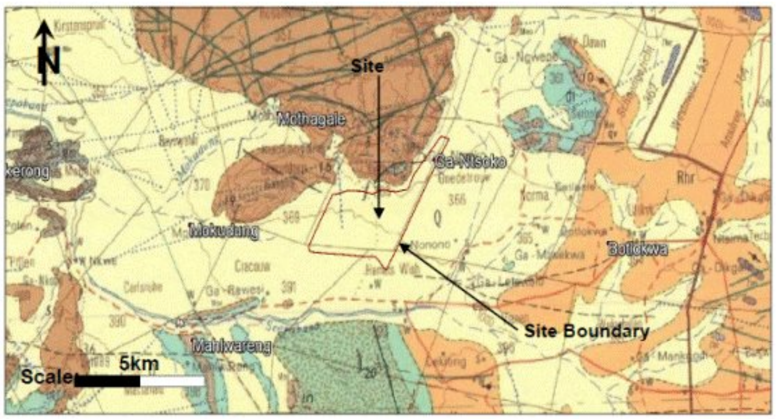


Figure 16-1: Geological Situation of the Proposed Site

Table 16-1: Legend of the Geological Materials Present in the Studied Area

Age	Symbol	Legend	Geological Time	Formation, Group, Suite and Complex	Geological Description
Youngest	Q		Quaternary	N/A	Soil, sand, alluvium; calcrete; scree
	di		Mokolian	N/A	Diabase; dyke
	Mm		Mokolian	Mogalakwena Formation (Waterberg Group)	Coarse-grained purplish brown sandstone; conglomerate and boulder conglomerate
	Mma		Mokolian	Makgabeng Formation (Waterberg Group)	Medium-grained, yellowish, laminated sandstone
	Mse		Mokolian	Setlaole Formation (Waterberg Group)	Coarse-grained dark-coloured sandstone; minor conglomerate, arkose, tuff and mudstone
	Vmo		Mokolian	Rustenburg Suite (Bushveld Complex)	Magnetite gabbro, gabbro, anorthosite, olivine diorite; magnetite layer
	Vm		Vaalian	Rustenburg Suite (Bushveld Complex)	Gabbro, norite, anorthosite, pyroxenite, harzburgite, troctolite
	Rhr		Randian	Hout River Gneiss	Leucocratic migmatite and gneiss, grey and pink hornblende-biotite gneiss, grey biotite gneiss; minor muscovite-bearing granite, pegmatite and gneiss
	Zbp		Swazian	Bandelierkop Complex	Metapelite
	Zbq		Swazian	Bandelierkop Complex	Magnetite quartzite, metaquartzite
	Zbm		Swazian	Bandelierkop Complex	Amphibolite, mafic granulite
Oldest	Zbu		Swazian	Bandelierkop Complex	Peridodite, dunite, metapyroxenite, hornblendite

Material covering the site area (yellow: Q) — Material underlying the site area (orange: Mma, Mse)

16.2.2 Geotechnical Investigation

A geotechnical investigation was undertaken to investigate founding conditions for the surface infrastructure elements of the current project layout. The expected loads communicated by the consulting company "DRA" are between 100 to 250 kPa for the light/medium structures and over 250 kPa for the heavy proposed structures. The investigation comprised a site walkover, fieldwork and laboratory testing. Fieldwork on site was undertaken between the 1st and 4th of June 2016 and comprised the following;

- Trial pits excavated with a TLB and profiled according to standard practice.
- DCP testing undertaken close to the test pit.
- Present preliminary design recommendations for the proposed portals
- Soil samples recovered from representative materials on site
- boreholes inspected were close to the proposed surface crusher's area.
- Rock core samples recovered from the borehole was submitted for laboratory point load testing.

16.2.3 Trial Pitting

Trial pits were excavated on the site. Figure 16-2 shows the location of the trial pits. The trial pits were excavated near the proposed structures on the site and will provide an overall assessment of the in-situ conditions.

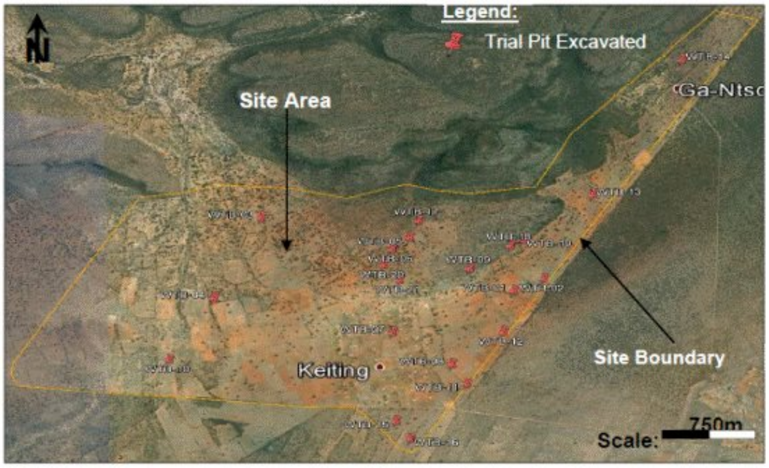


Figure 16-2: Position of the 21 Trial Pits on Satellite Image

The different lithologies observed in the trial pits are shown in Table 16-2.

Table 16-2: Lithologies Encountered on Site

Lithologies Encountered	Geotechnical Description
1	Very loose to loose, Sand with some roots, dry to slightly moist, brownish/reddish/yellowish orange, intact, Transported Material (Aeolian), Kalahari Sand (Quaternary Deposit (Q): Soil, Sand Alluvium, Calcrete).
2	Loose to medium dense, Silty Sand with some roots, dry to slightly moist, light brown, intact, Transported Material (Aeolian), Kalahari Sand (Quaternary Deposit (Q): Soil, Sand Alluvium, Calcrete).
3	Dense to very dense, Sandy Gravel with some cobbles and ferruginised pebbles, dry, light brown, intact, Transported Material, Alluvium?
4	Strongly cemented to very strongly cemented Calcrete Pan, dry, whitish beige, intact, pedocrete.
5	Loose to medium dense, Sand with boulders (min: 250mm, max: 600mm, ave: 350mm) and cobbles (min: 50mm, max: 150mm, ave: 100mm), reddish brown, intact, Transported Material (Colluvium, Scree).
6	Dense to very dense, Sand ferruginised in some areas (consolidated sand), dry to slightly moist, yellowish beige mottled orange, intact, Transported Material, Kalahari Sand (Quaternary Deposit (Q): Soil, Sand Alluvium, Calcrete).

Additional observations made were as follows:

- Stability of Trenches: The sidewalls of the trial pits were relatively stable during and after excavation. All test pits were logged from surface.

- Ground water seepage: No groundwater seepage occurred in the test pits during this investigation. However, a shallow permanent water table is usually expected at the interface between the bedrock and the transported aeolian material.

16.2.4 Portal investigations

A geotechnical investigation was undertaken to investigate the soil/rock characteristics to present preliminary design recommendations for the proposed portals of the current project. The investigation comprised a site walkover, fieldwork and laboratory testing.

Table 16-3: Summary of the Proposed Portals

Area Name	Coordinates (WGS 84 - Decimal Degrees) Latitude	Coordinates (WGS 84 - Decimal Degrees) Longitude	Elevation (m)
Portal T-Zone	S23.389923°	E28.878771°	1009
Portal F-Central Zone	S23.384506°	E28.888392°	1030
Portal F-North Zone	S23.362908°	E28.903461 °	1085

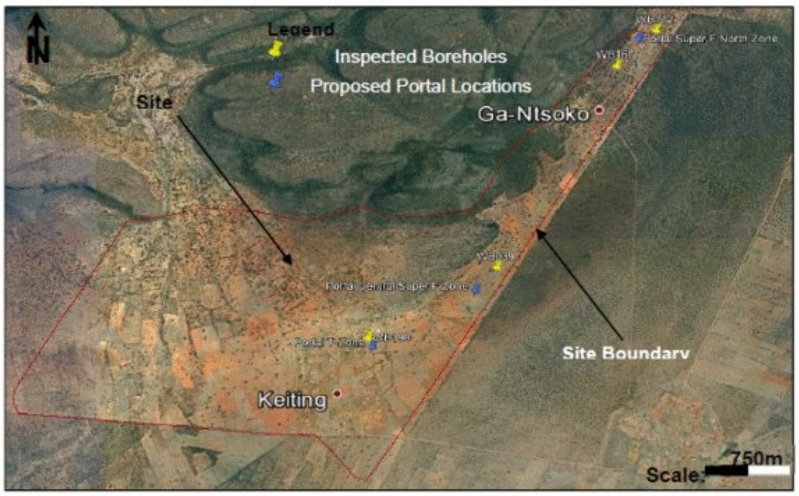


Figure 16-3: Position of the Four Inspected Boreholes on Drawing Layout

Table 16-4: General Lithologies encountered in the Inspected Boreholes for the portal area

Lithologies Encountered	Geotechnical Description
1	Very loose to loose, Sand with some roots, dry to slightly moist, brownish/reddish/yellowish orange, intact, Transported Material (Aeolian), Kalahari Sand (Quaternary Deposit (Q): Soil, Sand Alluvium, Calcrete).
2	Soft to medium hard rock Sandstone (pebbly in some areas), moderately to slightly weathered, bluish grey to reddish beige, moderately to highly fractured, clean joint in general (thick gauges in some areas), medium rough to rough, fine to medium grained.

16.2.5 Rock Mass Classification

Rock mass classification was determined from the data captured on site during the boreholes inspection and the laboratory results. The different rock mass classifications are summarized in Table 16-5. The rock mass quality is relatively poor overall with low RQD values and the presence of two predominant joint sets (80-90 degrees and 0-5 degrees). However, some areas in the rock mass could be expected to classify as a fair rock quality.

Table 16-5: Summary of the Different Rock Mass Classification

RMR (Bieniawski,1989)	MRMR	GSI	Q (Barton, 1995)
43	21	45	49

16.2.6 Rotary Core Drilling

Boreholes were inspected during the investigation to identify the appropriate founding depth for the surface proposed crushers. The coordinates of the proposed surface crushers and the two inspected boreholes are summarized in Table 16-6 and Table 16-7 respectively.

Table 16-6: Coordinates and Elevation of the Proposed Surface Structures

	Coordinates (WGS 84 - Decimal Degrees)		
Area Name	Latitude	Longitude	Elevation (m)
Primary Crusher (Surface)	S23.393144°	E28.881176°	1006
Secondary Crusher (Surface)	S23.392745°	E 28.881443°	1030

Table 16-7: Coordinates and Elevation of the Proposed Surface Structures

	Coordinates (WGS 84 - Decimal degrees)				Drilling Dates	
BHID	Latitude	Longitude	Elevation (m)	Max Depth Drilled (m)	Start Drilling Date	Stop Drilling Date
WB009	S23.39321°	E28.88029°	999.46	874.71	13-Feb-2012	30-Mar-2012
WB130	S23.39244°	E28.88185°	1001.99	812.18	3-Jul-2014	26-Jul-2014

The recommended founding level identified in the borehole is 4.21m depth from natural ground level on medium hard rock pebbly sandstone, moderately to slightly weathered, bluish grey, moderately fractured, clean joint, medium rough, fine to medium grained. The Corebox no.1 of the borehole is shown in Figure 16-4.

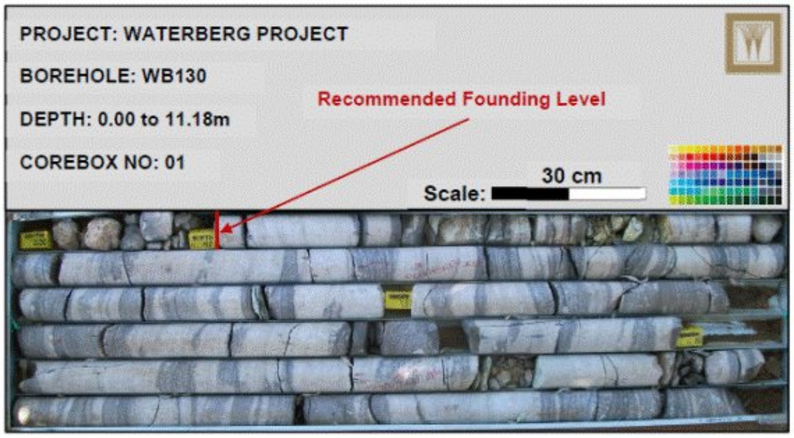


Figure 16-4: Corebox No. 1 of Borehole

16.2.7 Laboratory Testing

Laboratory tests were conducted to confirm the on-site investigation and establish engineering parameters for the soils. Tests were undertaken by SANAS accredited laboratories “SoilLab (Pty) Ltd”, “GeoLab (Pty) Ltd” and “RockLab (Pty) Ltd”. The results of this test were use in conjunction with the site observations and DCP testing to define land use potential and foundation recommendations.

16.2.8 Geotechnical Land Use Plan

According to the trial pits/rotary core drilling investigation and the laboratory test results, the site is classified as a “H1/S2/C2/R” site in the NHBRC Classification (slightly expansive, compressible and most probably collapsible soil horizons), with an expected range of total soil movements more than 20mm. The assumed differential movement is 50%. Figure 16-5 below shows the geotechnical land use plan for the proposed structures.

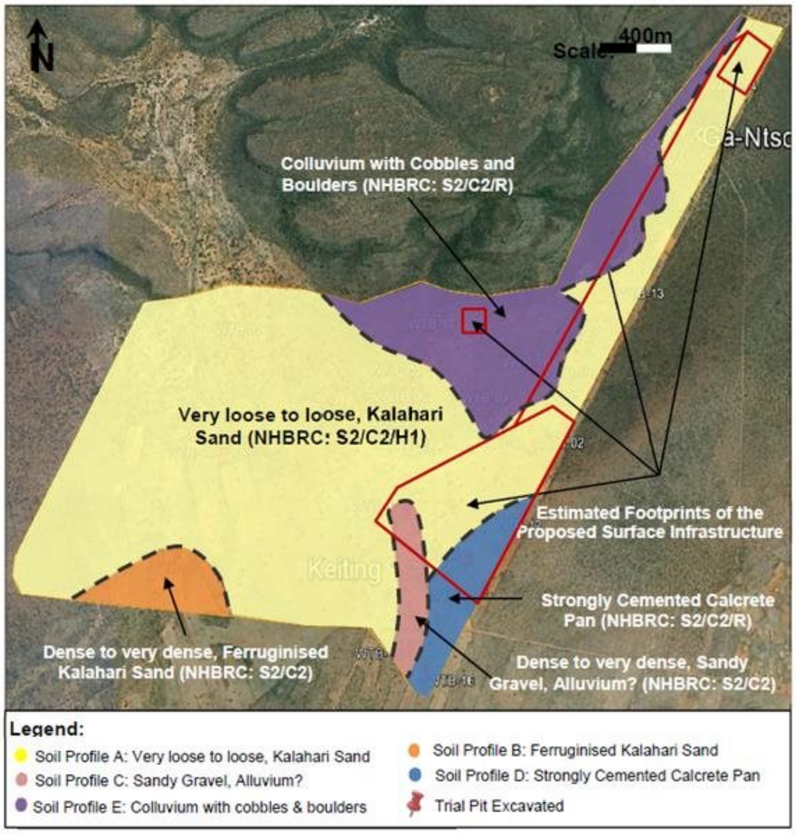


Figure 16-5: Geotechnical Land Use Plan according to the NHBRC Classification

16.2.9 Ground Conditions

The site is covered by five identified soil profiles (Kalahari sand, ferruginised Kalahari sand, colluvium, alluvium and strongly cemented calcrete) across the proposed site.

Refusal of the TLB machine occurred on colluvium soil, alluvium soil and strongly cemented calcrete pan during the investigation. Slow progress was also observed within the dense to very dense ferruginised Kalahari sand.

The DCP test results confirm that the transported material layer found from 0.5m below ground level has an allowable bearing capacity of at least 50kPa.

Settlement analysis was performed from the single oedometer test results. Assuming that the thickness of the compressible layer is 4m, the width and depth of the foundations are 2m and 1m respectively, the predicted settlement is 262.3mm for a load of 200kPa.

Laboratory tests have shown that the soil profile A (Kalahari sand) to occasionally have a low potential of expansiveness. The potential heave that could be expected in localized area with the insitu material is 13.6mm according to Van der Merwe’s method. In general, ground conditions are considered to be favourable.

16.2.9.1.1 Geotechnical Constraints

Based on the geological and ground profile encountered, the following geotechnical constraints should be considered in the development of the site:

- Profile A exhibits low potential expansiveness. With a layer thickness of approximately 2m the total expected heave was calculated at 14mm for the profile.
- Poor compressible and collapsible conditions are expected in the transported soil horizons.
- Increased seepage is expected during/after periods of heavy or continuous rain.
- Soft to medium hard rock and boulders excavation might be expected which may hamper installation of services or foundation.
- The soil chemistry indicates the soils are moderately aggressive and adequate protection of steel structures and re-enforcement in concrete will be required.

16.2.9.2 Foundations

16.2.9.2.1 Light Structures* (100 – 150kPa)

Remove the soil to a depth of 1.6m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod AASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 150kPa.

16.2.9.2.2 Medium Structures* (150 – 250kPa)

Remove the soil to a depth of 3m below surface or up to the bedrock. The excavation must then be back filled with G6 materials in in 0.200m thick layers; compacted to 93% mod AASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 250kPa.

16.2.9.2.3 Heavy Structures* (250 - 500kPa)

Remove the soil to a depth of 4m below surface or up to the bedrock. The excavation must then be back filled with G5 materials in in 0.200m thick layers; compacted to 93% mod AASHTO, wetted at -1 to +2% optimal moisture content. Conventional pad foundations can then be placed at minimal depth (min of 1m deep) with bearing pressures limited to 500kPa.

Notes*: Soil raft foundation with good site drainage is recommended. 93% compaction is a reasonable expectation. Anything above that might not be achievable during construction. Soil mattresses will have to be found on dense sand (>100kPa) as a minimum.

16.2.9.2.4 Primary and Secondary Surface Crushers

Spread foundations founded on the bedrock are considered feasible. Allowable bearing capacity of at least 5MPa, which is generally suitable for a crusher structure, was confirmed with the point load test results. The recommended founding level was identified at 4.21m depth below natural ground level in the borehole WB130. Good founding material (medium hard rock sandstone) will have to be validated by a competent person during construction.

16.2.9.3 Groundwater

No permanent or perched water levels were encountered during the investigation. However, a shallow perched water table is usually expected at the interface between the bedrock and the transported aeolian material. A DCP membrane and polyolefin sheeting are recommended below the foundations to protect them from the effects of rising damp.

16.2.9.4 Materials and Roads

The material encountered on the site, in the test pits WTB06, WTB12 and WTB19 between 0.2 and 1.4m below ground level test as G7 according to the COLTO classification and is thus generally suitable for use in engineered layer work applications. Further testing would be necessary if proposed for use.

16.2.9.5 Services and Excavatability

Transported materials encountered across the site would classify as "soft" according to the SABS 1200 D Earthworks classification, or as "Soft class 2" (materials which can be readily excavated with the aid of a pick) according to the Department of Works, (Watermeyer, 1997).

Soft to medium hard rock sandstone and strongly cemented calcrete pan can be expected at shallow depth below ground level. Some variation can be expected over the site. Blasting may be required to maintain the lines and levels of services and foundations depending on the design depths.

16.2.9.6 Stability of Trenches

The sidewalls of the trial pits were relatively stable during the investigations. Trial pits excavated during the geotechnical investigation give an optimistic indication of the stability of long trench excavations. It remains the responsibility of the contractor and engineer on site to ensure that excavations are stable. Lateral support; shoring, and or battening at excavations deeper than 1.0m is envisaged.

16.2.10 Portal Design

16.2.10.1 Ground Conditions

Laboratory tests have shown that the Kalahari sand has an apparent friction angle of 25.6 degrees and an apparent cohesion of 9 kPa. The rock mass quality is relatively poor overall (40% in average) with the presence of two predominant joint sets (80-90 degrees and 0-5 degrees). Some areas in the rock mass could be expected to classify as a fair rock quality. It is very probable that the cores have been disturbed heavily during the transport from site to core yard and artificial breaks might occur at this time during the site works.

In general, ground conditions are considered favourable for the proposed portals.

16.2.10.2 Preliminary Portals Design

Access to the underground workings will be by a twin decline system, consisting of a conveyor belt decline and a men and materials decline. These declines are to be developed at a dip of 9° from the highwall of the box cut.

Portal designs were created based on professional experience in similar ground environment and geotechnical information gathered from the inspection of four boreholes drilled near the proposed portals location.

The suggested preliminary portals designs presented in Figure 16-6, Figure 16-7, Figure 16-8 and Figure 16-9 will have to be supported and approved with the finite element and limit equilibrium methods during the Bankable Feasibility Study (BFS) to reach an acceptable Factor of Safety (FoS) determined for the project. The proposed portals designs were conducted in a manner consistent with the level of care and skill ordinarily exercised by members of the geotechnical profession practicing under similar conditions in the locality of the project.

16.2.10.2.1 Portals T-Zone and Central F North

The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 37° inclination into soil material. The high wall is 20 m high up from the footwall position. The overall slope angles are 41° and 50° for the sidewalls and highwall respectively in the preliminary portal design. The top two benches have a height of 4m. The remaining benches are 6m height. The catch berms have a width of 3m across the highwall and sidewalls.

16.2.10.2.2 Portal F North Zone

The box cut will consist of a bottom sidewall with an inclination of 51° into rock and a top sidewall of 36° inclination into soil material. The high wall is 35 m high up from the footwall position. The overall slope angles are 38° and 44° for the sidewalls and highwall respectively in the preliminary portal design. The first bench has a height of 5 m. The remaining benches are 6 m height. The catch berms have a width of 3 m across the highwall and sidewalls.

16.2.10.2.3 Support and Drainage Recommendations of the Portals

- 75mm of mesh and shotcrete or fibre shotcrete on the sidewalls and highwalls slopes
- In rock material, 6 m long 20mm rock bolts full column grouted with spider faceplates 100mm square and 200mm long 10mm spider legs. The anchor spacing should be 2m in a diamond pattern.
- In soil material, the three top rows of support on the highwall and sidewalls should be 3 m long soil nails full column grouted Self Drilling Anchors in 100mm holes on a 1.4m diamond pattern. Six meters long, Self-Drilling Anchors should be implemented in the remaining support rows until rock material is reached.
- Support should be implemented at 25° from horizontal.
- A 1.5m high berm wall of compacted soil should be built around the excavations.
- An appropriate drainage channel at the invert should be built to divert storm water from entering the decline.

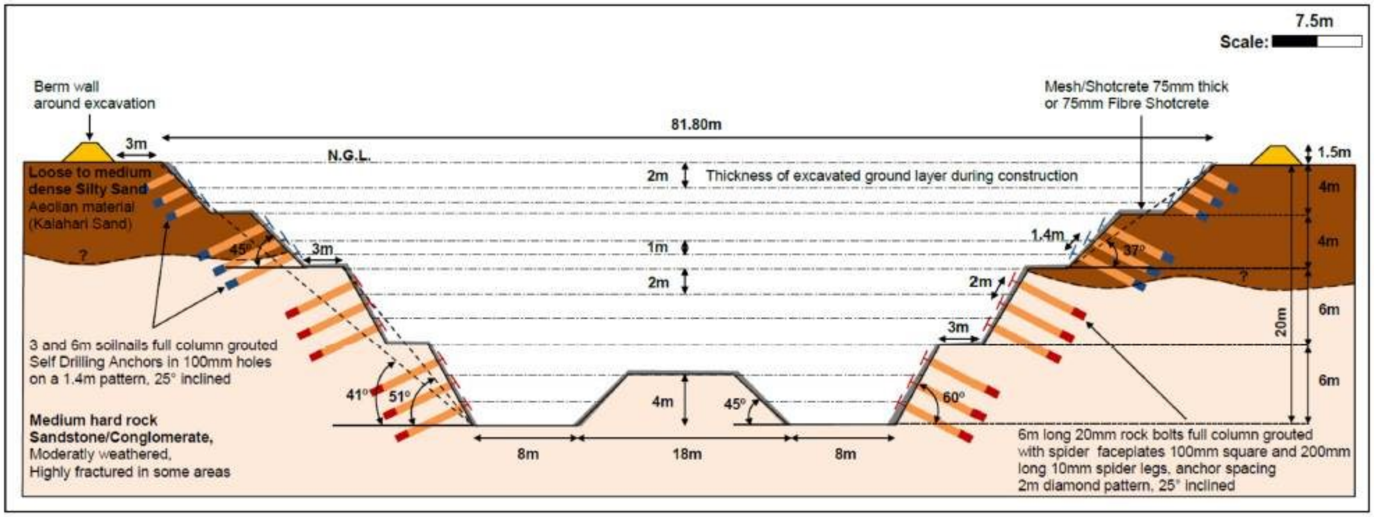


Figure 16-6: Cross Section of the T-Zone and F-Central Zone Portals

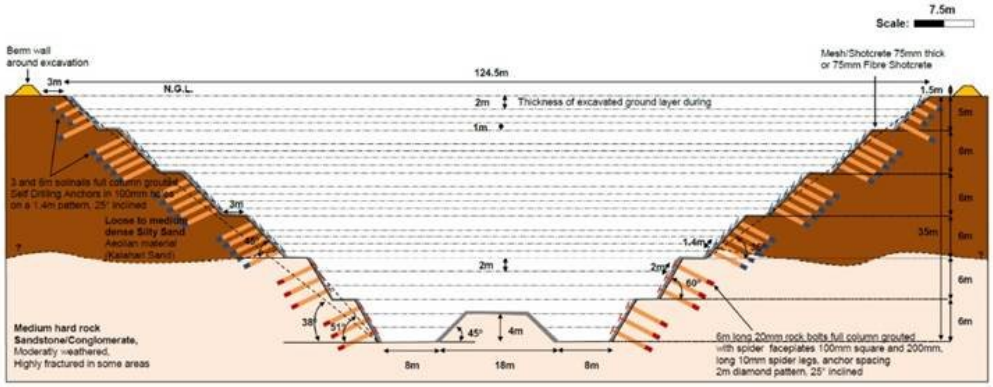


Figure 16-7: Cross Section of the Northern Portal

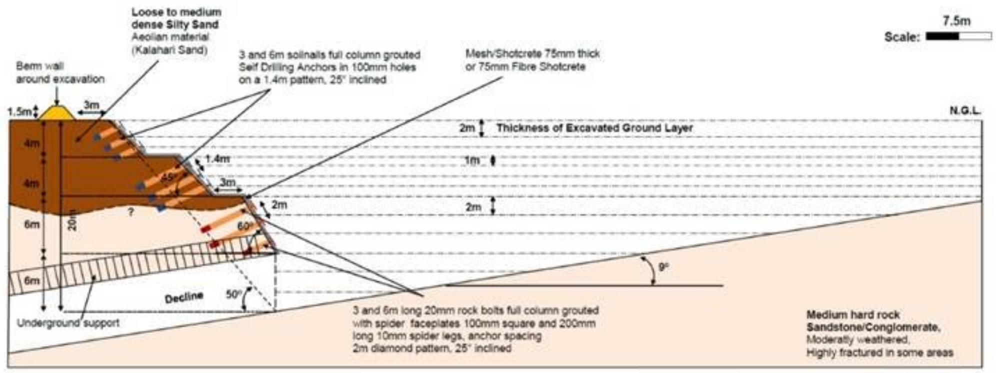


Figure 16-8: Cross Section showing the Entrance of the T-Zone and F-Central Zone Portals

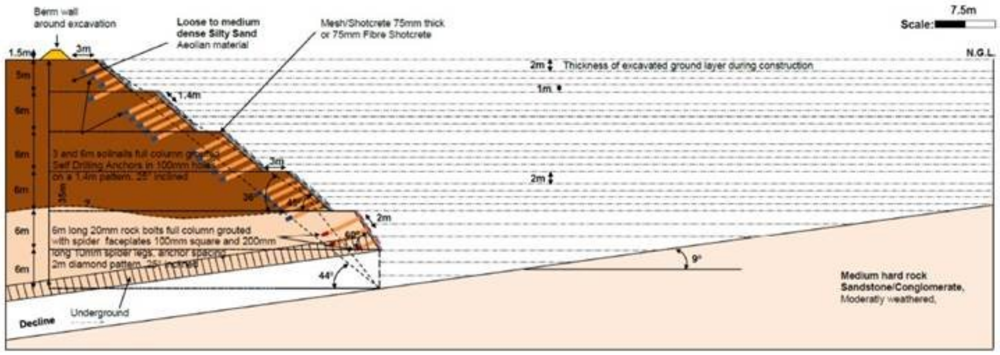


Figure 16-9: Cross Section showing the Entrance of the F-North Zone Portal

16.3 Rock Mechanics

This section is a summary of the PFS-level mining geotechnical investigation by WPRSA for the Waterberg Project in the Limpopo Province of South Africa. A summary dealing with rock mechanics topics pertinent to mine design is provided in this section.

16.3.1 Geotechnical Investigations

16.3.1.1 Geotechnical Database and Site Investigation

16.3.1.1.1 Review and Check Logging of Existing Data

The client provided an existing geotechnical database compiled by Open House Management Solutions (OHMS) and comprising 20 boreholes and 9,512m of geotechnically-logged core. A reviewed of the database by Middindi Consulting (Pty) Ltd was also made available. Investigations by WPRSA in quarter 4 of 2015 comprised a thorough review of the existing database as well as a detailed onsite verification process. Later in 2016 the database was expanded through a new drilling and geotechnical logging initiative (refer to Section 16.3.1.1.2).

In reviewing the database, WPRSA logged 343m of selected core from four different boreholes and conducted spot checks on an additional six boreholes during the on-site investigation. A list of the re-logged boreholes is presented in Table 16-8.

This on-site investigation (Shangase and Thompson (2015)) resulted in reviewing 67% of the available geotechnical logging and minor adjustments to the database. The revised WPRSA database includes the following:

- Changes in observation/interpretation of results;
- Adjustments across the database to the interpretation of joint properties based on detailed site observations; and
- Inclusion of laboratory strength test results.

Inspection of the existing and revised geotechnical data revealed that the rock mass quality was atypical of the lithology of the Bushveld Igneous Complex (BIC) encountered in other platinum and chrome mines. In particular, the following were observed:

- The fracture frequency is higher. Frequencies of approximately one to two fractures per meter, with zones of dense fracturing in between are typical of the BIC. The database for the Waterberg site indicates more than double this fracture frequency with three to six fractures per meter occurring commonly;
- The variation in joint intensity and spacing over short distances appeared to be wider than commonly encountered on other mines in the BIC;
- Joint surfaces tend to be stepped and slicken-sided. This is an uncommon combination. Typically, slicken-sided joint surfaces are accompanied by straight or undulating joint profiles.

Table 16-8: List of Boreholes Revisited during the On-site Review of the Geotechnical Database

BH ID	Detailed logging m	Spot checked m	Comments
WB016		536.35	WB016 was laid out, old borehole, badly weathered/degraded
WB017		501.43	WB017 was laid out, old borehole, badly weathered/degraded
WB119		784.53	WB119 Check logged — AVT data hole. OHMS database and observations agree
WB123		479.78	WB123 Check logged — AVT data hole. OHMS database and observations agree
WB139	155.64	916.4	WB139 Reassessment on site — focus on HW and FW of T- and F-Zones
WB150	109.14 (40%)	163.70	WB150 Reassessment on site — focus was uncut F-Zone core
WB164		901.68	WB164 Geology borehole
WB165	39.12	1,176.72	WB165 Check logged — database and observations differ
WB182		480	WB182 Uncut T-Zone
WB182D1	38.79	70.12	WB182 Deflection1 Uncut T-Zone
Total	**342.69**	**6,010.71**	

The site-verification process led to the following conclusions:

- The OHMS interpretation of the rock mass quality was, in general, confirmed through the WPRSA observations; however, interpretations of the joint sets and joint surface conditions led to lower ratings as compared to the OHMS analyses.

- Check logging revealed a reasonable correlation between the OHMS analyses and the WPRSA observations in terms of fracture frequency (RQD and joint counts). This observation supported the higher fracture-frequency estimates and led to improved confidence in the data.

- The check logging revealed that more detail could have been recorded. For example:

  - Discontinuity types (i.e. joints, faults, shears or crushed zones) were not differentiated;

  - There was a general lack of descriptive supporting comments to database records;

  - Many highly fractured/crushed zones had been lost through joint count smoothing and the determination of RQD over large intervals. The classification of such areas as separate geotechnical domains would have been useful for mine design.

16.3.1.1.2 New Geotechnical Data

Immediately prior to the PFS, additional geotechnical data was obtained by Advisian through core logging of 23 recently-drilled boreholes as indicated in Table 16-9. This work is detailed in Louw and Shangase (2016).

Of these 23 holes, four were logged geotechnically over the full borehole length from surface. These included two dedicated geotechnical boreholes, namely WE078 and WE079, from which core samples were collected in anticipation of possible additional laboratory test work later. The remaining boreholes were logged over an interval of 20m above to 20m below the F-reef mineralised zone.

Analysis confirmed the rock mass properties in general and that the previously unexplored portions of the F-reef do not display significant deviations. The updated geotechnical database now comprises data of nearly 12,190m.

The complete geotechnical database (including laboratory test results) was used to determine rock properties and classify the rock mass. Figure 16-10 shows the locations of all the geotechnically-logged boreholes. From the information gathered through the geotechnical investigations, data analyses and interpretations, Datamine Ltd. built a Three-Dimensional (3D) rock mass model (block model and wireframes). The rock mass model enabled geotechnical data to be overlain onto mine designs and geological information to provide visualisation and assist in localisation of rock mass parameters.

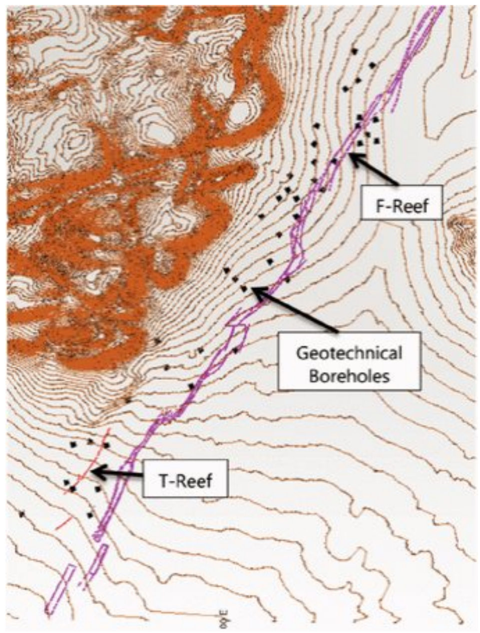


Figure 16-10: Surface Contour Map showing the Geotechnical Boreholes Used for the Waterberg Project in Relation to the Sub-Crops of the F- and T-Reefs

Table 16-9: Boreholes Geotechnically Logged by WPRSA

Planned	Meters logged				
BHID	**HW**	**Mineralisation**	**FW**	**Total metres**	**BH length**
WE001	40.8	0	32.1	72.94	850.5
WE003	21.7	44.5	0	66.22	666.5
WE017	19	0	16.2	35.17	948.17
WE019	39.5	0	16.7	56.13	535.66
WE020				299	966.97
WE022	21.4	0	12.2	33.61	709.22
WE023	19.8	0	32.1	51.89	542.15
WE024	23.2	0	36.6	59.77	728.62
WE029	29.7	0	27.8	57.49	566.75
WE032	21.2	0	2.04	23.25	586.04
WE033	24.1	0	1.61	25.74	783.84
WE034				94	763.32
WE052	23.6	0	0	23.56	379.57
WE054	20	0	0	20.04	427
WE058	26	0	4.63	30.63	824.63
WE061	52.9	0	18.5	71.36	502.49
WE016	17.1	0	0	17.07	883.8
WE066	36.9	0	0	36.89	760.27
WE073	21.2	0	30.1	51.26	358.47
WE078				390.94	390.94
WE079				430.36	430.36
WE080				373.56	373.56
WE083				388.57	388.57
Total (m)				**2,709.5**	

16.3.1.1.3 Laboratory Testing

The engineering properties for the different rock types encountered at the Waterberg site, has been assessed through laboratory testing. Laboratory tests carried out for the Waterberg Project are summarised in Table 16-10.

Table 16-10: Laboratory Tests Conducted for the Waterberg Project

Test	No of Samples
Density (t/m ^{3})	193
UCS with determined deformation modulus and Poisson's ratio (UCM) (MPa)	75
Ultimate Tensile Strength (UTS) (MPa)	30
Triaxial Compressive Strength (TCS) (MPa)	88

16.3.1.2 Geotechnical Domains and Mineralised Zones

Figure 16-11 shows the major geological units of the Waterberg Project. These units have also been adopted as the geotechnical domains for this phase of study. The Waterberg formation has a gentle dip of about 5º to the west, and unconformably overlies the BIC lithology. The Bushveld stratigraphy dips generally at 34º to 38º towards the west. However, some blocks may be tilted at different angles depending on structural and/or tectonic controls. Generally, the Bushveld package strikes southwest to northeast.

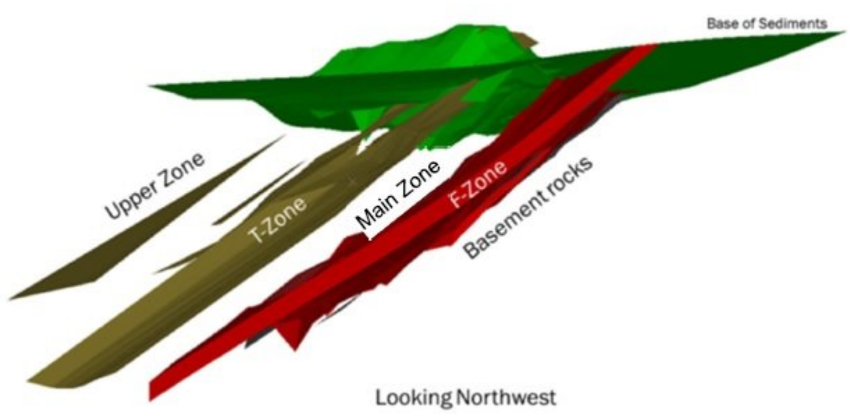


Figure 16-11: 3D View Showing Major Geological Units at the Waterberg Project

Mineralization of economic interest is localised in two zones. These are the T-reef and the much more extensive F-reef.

The T-reef is fully contained within the Bushveld Main Zone and occurs close to the contact of the overlying Upper Zone. Although the T - Zone consists of numerous mineralized layers, of which two, the T1 and the T2, show economic potential. These are composed mainly of anorthosite, pegmatoidal gabbros, pyroxenite, troctolite, harzburgite, gabbronorite and norite.

The F-reef is hosted in a cyclic unit of olivine-rich lithologies towards the base of the Main Zone at the bottom of the BIC. The F-reef, which is divided into the FH and FP layers, consists of alternating units of harzburgite, troctolite and pyroxenite. The FH layer has significantly higher volumes of olivine in contrast with the lower-lying FP layer, which is predominately pyroxenite.

The FH layer is further subdivided into six cyclic units identified by their geochemical signature, especially chrome. The base of these units can be identified by a pyroxenite layer.

16.3.1.3 Structural Geology

Wireframes of the structural geology were not available for this phase of the study. The following is however evident based on observations:

- Numerous dolerite and granodiorite sills and dykes criss-cross the Waterberg Sediments and range in thickness from less than 1m to more than 90m,
- A large number of shear zones have been identified through mapping and geological logging. These altered weakness zones present themselves throughout the main zone above the F reef and appear to me omnipresent (Figure 16-12) and are generally less than 3 to 5 cm in thickness when observed in drilled core. No clear understanding of the lateral extent of (or the interconnectedness of) the identified zones could however be formulated with the data on hand.

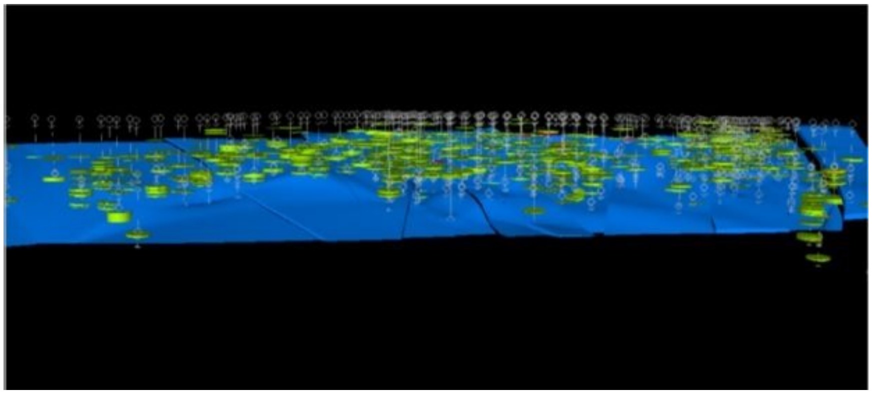


Figure 16-12: View, Looking Northeast, of the F-Zone Wireframe (Blue) and Boreholes Highlighting Weakness/Thrust Zones (Green discs)

A structural geological model will be required for future study phases as the location and quality of geological structures are critical to assessing stability and rock mass behaviour in all aspects of mine design. In addition, the spatial distribution of weak zones requires improved understanding to ensure safe, optimal extraction and to minimise potential for dilution.

16.3.1.4 Characterization of Rock Strength

Table 16-11 summarises the results of selected properties from laboratory tests for different stratigraphic domains.

The laboratory tri-axial results were used in the RockLab program by RocScience to estimate failure criteria. These together with the elastic properties were used for all the subsequent numerical models. The parameters are listed in Table 16-12. The UCS is somewhat higher than appears from averages in Table 16-11 because it is based on all the tri-axial sample tests. It does not discriminate based on domain, which for this level of study is not warranted, mainly because there are not enough samples per domain for meaningful statistical analysis, as shown in Table 16-3.

Table 16-11: List of Selected Laboratory Values for the Waterberg Project

Geotechnical Domain	Density (t/m^3)	Average UCS (MPa)	Average Tangent Elastic Modulus (GPa)	Average Poisson's Ratio
Sediments (SEDS)	2.7	123	34	0.12
Sill intrusive (SILL)	2.9	N/A	N/A	N/A
T-Reef Hanging Wall (T_HW)	2.9	144	90	0.23
T-Reef Mineralised Zone (T_MIN)	3.0	152	92	0.21
T-Reef Footwall (T_FW)	3.0	163	102	0.26
F-Reef Hanging Wall (F_HW)	3.0	163	102	0.26
F-Reef Mineralised Zone (F_MIN)	3.0	193	96	0.28
F-Reef Footwall (F-FW)	2.8	212	72	0.24

Table 16-12: Derived and Estimated Rock Mass Parameters for Numerical Modelling

Parameter	Value
Intact uniaxial compressive strength (MPa)	197.2
GSI*	40
mi (from tri-axial testing)	10.496
Disturbance factor	0.8
Intact modulus (GPa)	90
Hoek-Brown m_b	0.295
Hoek-Brown s	0.0001
Hoek-Brown a	0.511
Mohr Coulomb fit**cohesion (MPa)	5
Mohr Coulomb fit **phi	17.38
Rockmass tensile strength (MPa)	0.075
Rockmass deformation modulus (MPa)	4592.12

* *GSI estimate is low because of the uncertainty observed about localised olivine-rich and serpentinised zones of weakness. These uncertainties can be reduced and the models can be re-calibrated to yield improved results when more is learnt about the nature of the rock mass.*

** *The Failure Envelope Range in RocLab was set to "general" to represent brittle rock. Selecting different ranges will result in different cohesion and friction angle values. The Mohr Coulomb failure criterion was not used for the modelling in this study.*

16.3.1.5 Classification of Rock Mass Quality

The derivation of the rock mass classifications is contained in Appendix 16.1 along with the logs of all fieldwork conducted. The available geotechnical database is comprehensive and of good quality for this level of study; however, the data is insufficient to allow for fine geotechnical domaining and sub-domaining. The geotechnical domains were, therefore, classified according to the lithologies listed in Table 16-13.

Table 16-13: Geotechnical Domains for the Waterberg Project

Geotechnical Domain	Description
SEDS	Waterberg formation sediments
SILL	Refers to the various sill intrusions present both within the sediments and along its bottom contact
T_HW	Hanging wall of the T-Reef
T_MIN	T-Reef mineralised zone
T_FW	Footwall of the T-Reef
F_HW	Hanging wall of the F-Reef
F_MIN	F-Reef mineralised zone
F_FW	Footwall of the F-Reef

Rock mass quality has been assessed for each of these domains based on well-known classification systems:

- The Rock Quality Designation (RQD),
- The Rock Mass Rating (RMR) system by Bieniawski (1974)
- The Q-system by Barton et al (1974) and most recently described in NGI (2015), and
- The Mining Rock Mass Rating (MRMR) system by Laubscher (1990).

Literature on these classification systems is abundantly available and they will not be described in this report. The results are presented in the graphs in Figure 16-13 and design values are summarised in Table 16-14.

From the graphs in Figure 16-13, the following can be deduced:

- Series labels in each graph represent the normal distribution 25%, 50% (median) and 75% percentile values.
- The rock mass quality across most domains can be described as fair to good for design purposes. The exceptions are the sediments and the sills which are of somewhat poorer quality in places;
- Variation of rock mass quality between domains is in a narrow band;
- Deviation from the median is relatively small in all cases implying that the median is a suitable value to use for design purposes; and
- The statistical analyses provide a good mean of assessing the overall expected rock mass conditions; however, they do not consider the spatial distribution of rock mass quality. To assist with this, a 3D geotechnical block model was constructed.

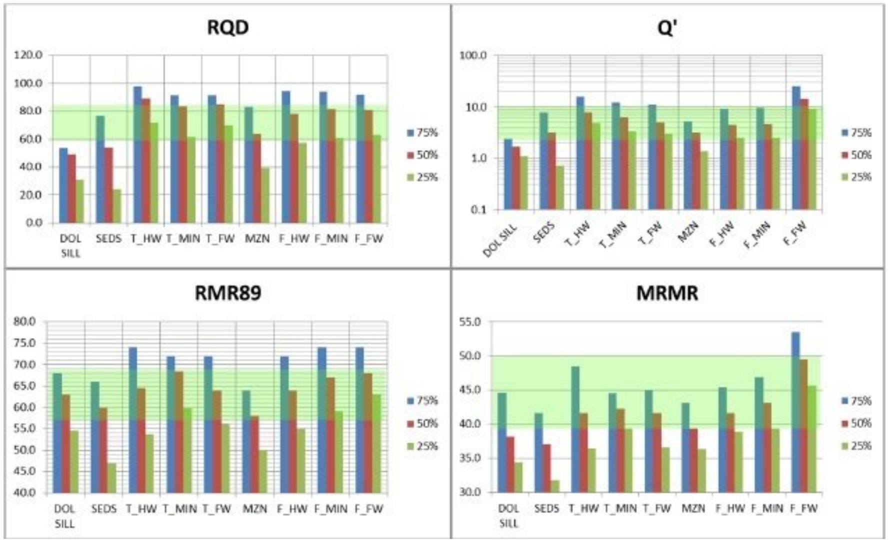


Figure 16-13: Rock Mass Classification for the Waterberg Project

Table 16-14: Design Rock Mass Parameters

Rock Mass Parameter	Design Range	Rock Mass Class
Q’	4 to 10	Fair
RMR89	55 to 70	Fair to Good
RMR90	50 to 65	Fair to Good
MRMR90	40 to 50	Fair

16.3.1.6 3D Geotechnical block model

The 3D geotechnical block model was built using the data from the geotechnical database. The model includes the Waterberg sediments and the sill that occurs in it as well as the mineralised zones of the T-Reef and F-Reef together with 20m of hanging wall and footwall for each. In this context, the T-Reef and F-Reef comprise the whole package and not just the economic horizon of each.

The 3D model is shown in Figure 16-14. The figure shows a section view, looking eastwards and it shows contours of the Q’ values for the Waterberg Project. From this section, the following observations can be made:

- The sediments are classed in the “fair” category for most part, the exceptions being a poor quality region in the middle and a good quality region in the south of the project area;
- The sill is generally of good to extremely good quality in the south; however, the quality deteriorates to poor in the central parts and very poor in the far north;
- The T-Reef hanging wall and mineralised zones are uniformly of fair quality, while the footwall ranges from fair to extremely good; and
- The F-Reef ranges from fair to very good. The central portion and a portion to the north have a poor hanging wall.

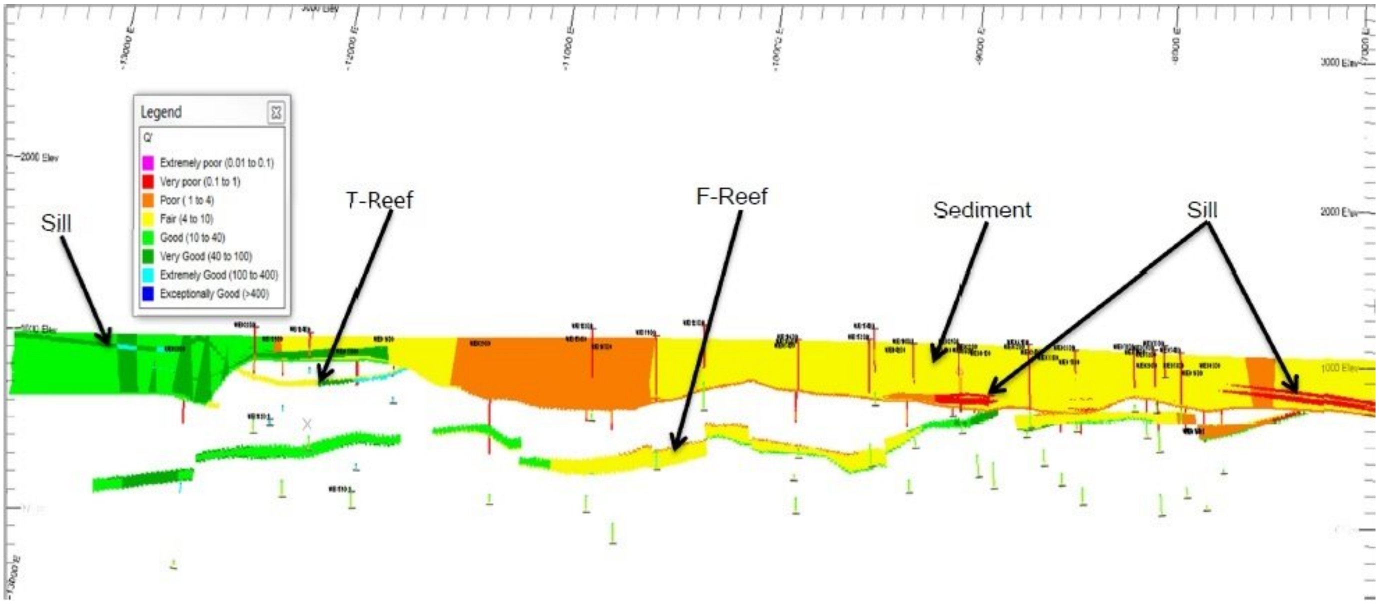


Figure 16-14: Section Looking Easterly of Q' Values

The 3D geotechnical block model is adequate and useful for this level of study; however, the potential power of such a model is best appreciated when it is combined with the geological structure and perhaps a geohydrological model. Additional data is required to expand it to incorporate geological structures and other weak zones such as the olivine-rich shear zones. The Forward Work Plan (FWP) includes recommendations to facilitate expansion of the block model. This will allow for more elaborate assessments during the next and subsequent phases of study.

16.3.1.7 Stress Regime

The stress regime for the Waterberg Project has not been measured but based on historic data the stress conditions in many Bushveld Igneous Complex (BIC) operations are characterised by a high ratio of horizontal-to-vertical ground stresses (K ratio > 1) at shallow depth reducing to a K-ratio of 1 at approximately 500m depth. Refer to Figure 16-15 after Jager & Ryder (1999). It is however clear that the scatter is very wide and that it is not possible to select typical values with confidence. It is nevertheless reasonable, within the given constraints, to assume that the stress gradient is equal to 0.03MPa per meter (based on the average density of the intact rock) and that the K-ratio = 1, for the expected depth range of the Waterberg complex, which is 700m below surface and less. Model sensitivity to varying K-ratio must however be kept in mind.

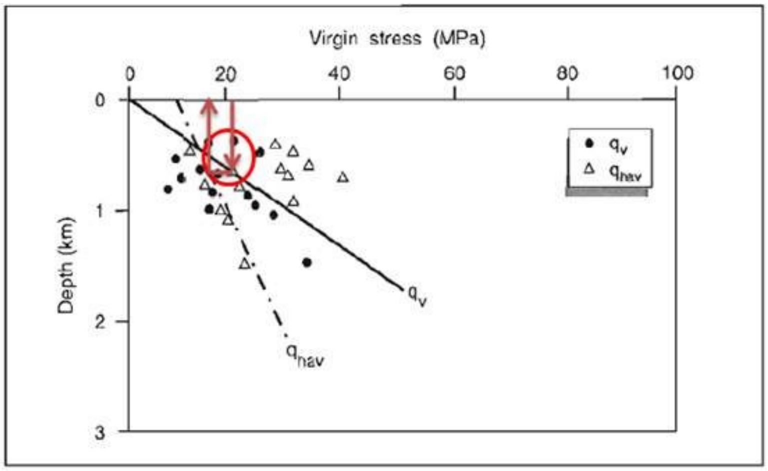


Figure 16-15: Vertical and Horizontal Stress Variation with Depth for Western Bushveld Platinum Mines after Jager & Ryder (1999)

16.3.2 Mining Methods

Although there are technically two orebodies, the T-Reef and the F-Reef, they are distinguished based on location, thickness and dip. There are no substantial differences from a rock mechanics perspective except for the basement gneiss. These underlie the F-reef and have a higher rock mass quality. The parameters in this report are, therefore, applicable to the mining methods irrespective of the orebody and are dealt with in terms of mining districts.

16.3.2.1 Stepped Room and Pillar Mining Method

For the Waterberg Project, the initial designs provided for significant proportions of the orebody to be mined with the Stepped Room and Pillar (SRP) mining method. As more information regarding the orebodies became available, the need for more efficient mining methods became apparent. Previously, SRP would have been employed where the target area of the orebody was up to 6.0m thick, but current thinking is that a Blind Longitudinal Retreat (BLR) mining method will be employed. It is conceivable that there will still be areas where SRP could be the preferred mining method; however, this will probably be an operational decision and is not considered any further in the PFS. The reasons for sacrificing SRP in favour of BLR are as follows:

- SRP requires relatively large pillars to maintain stability while advancing;
- Because of the relatively steep dip and confined space, supporting the dip boards (ventilation holings) with mechanical bolters would be difficult. To circumvent the necessity of personnel and machinery beyond unsupported ground, the ventilation holings would have to be developed on the retreat together with pillar reclamation; and
- The similarities between SRP with secondary retreat (to increase extraction) and the BLR method (see section 16.3.2.2) would negate any advantages of SRP; in addition, mining would be simplified by employing one instead of two mining methods.

16.3.2.2 Blind Longitudinal Retreat Mining Method

The Blind Longitudinal Retreat Mining (BLR) method is an adaptation of the longitudinal sub-level open stoping method. Ring blasting is used to reclaim reef in the hanging wall but no sub-levels are deployed. Ring blasting must be conducted on retreat and no rib pillars are left. Instead, pillars of suitable dimensions and spacing are left against and between sacrificial secondary roadways to serve as protection pillars as well as regional stability pillars.

For BLR on the T-Reef, it is planned to develop triple declines on reef. The distances between the three declines will vary from 6.0m above 300m depth, to 12m where the depth exceeds 700m. It is planned that sets of sacrificial, secondary roadways will be developed on apparent strike from triple on-reef declines. These secondary roadways will be developed on average 50m apart (skin to skin) assuming an orebody dip of 35°. From these roadways, extraction drives will be mined on strike, 6.0m apart, and the ore will be extracted with ring drilling, retreating towards the secondary roadways.

The mining configuration is illustrated in Figure 16-16.

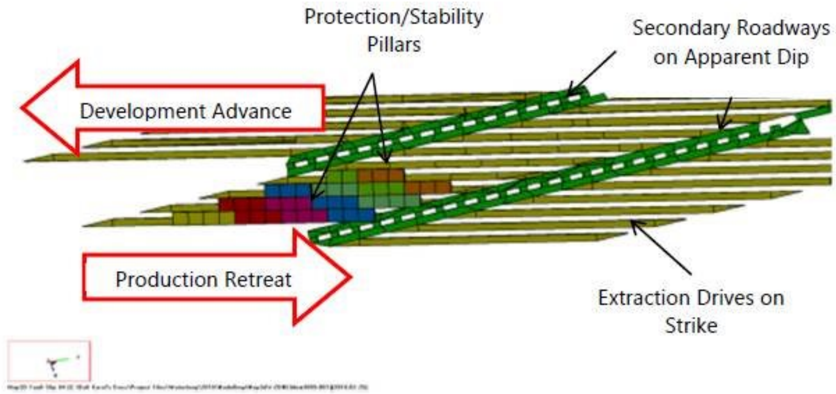


Figure 16-16: Mining Layout for BLR on Plan View

16.3.2.3 Sub-level Open Stoping

Sub-level Open Stoping (SLOS) is a well-known, widely practiced mining method and only the essential elements will be briefly discussed. The mining method is suited to steep dips (40° to vertical). Stopes are separated in the vertical sense by horizontal sill pillars and in the horizontal sense by vertical rib pillars. This nomenclature is illustrated in Figure 16-17.

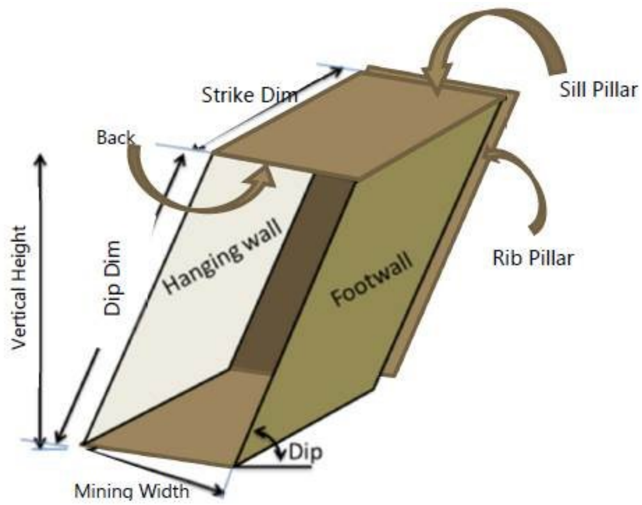


Figure 16-17: Illustration of Sub-level Open Stoping Nomenclature

Access to the orebody is via crosscuts developed from the footwall drive into and perpendicular to the orebody, half way between the rib pillars.

There are two basic configurations for this mining method. These are the longitudinal and transverse open stoping methods, both of which are discussed below.

16.3.2.3.1 Longitudinal Sub-level Open Stoping

When the orebody is relatively narrow, on-reef development takes place on strike. There is an ore drive more or less along the middle of the orebody developed from a crosscut half way between the rib pillars. The crosscuts connect the ore drive with footwall drives, which serve to transport the ore. Extraction of the ore is effected through long blast hole rings retreating along the ore drives from the rib pillars back towards the crosscuts.

16.3.2.3.2 Transverse Sub-level Open Stoping

This mining method is suited to relatively wide orebodies. Instead of an ore drive along strike, the crosscuts are developed from the footwall drive through the footwall of the orebody until the hanging wall is reached. The ring blasting then takes place back towards the footwall. The extremities of the stope are the rib pillars and the sill pillars similar to the longitudinal configurations. For the Waterberg, transverse sub-level open stoping will apply to areas where the orebody thickness exceeds 15m. This occurs predominantly in the F-reef horizon but also to a limited extent on the T-Reef horizon.

16.3.3 Analysis and Design

16.3.3.1 Surface Subsistence due to Mining

Whenever mining occurs relatively close to surface, it could lead to subsidence of the ground surface. Ways to ameliorate the risk, if present, include the leaving of ore pillars *in-situ* ; the placement of mining fill, or to place displacement-sensitive surface structures outside of an exclusion zone.

At the Waterberg, the intent is not to leave substantial pillars of to place extensive fill but rather to allow for later stope collapses if they do not extend upward far enough to affect surface infrastructure. Numerical modelling has been undertaken to assess the likely occurrence and extent of surface subsidence.

16.3.3.1.1 Methodology

The maximum strike extent of the orebody on the T-Zone is of the order of 500m. The mining method likely to be employed necessitates the leaving of protection pillars for the roadways. The roadways are, on average, about 50m apart on plan. The shallowest part of the mineable ore is about 140m below surface.

A series of simple Map3D models were run to investigate the risk of surface subsidence. For one of these models, three horizontal stopes with spans of 100m each were simulated with 8.0m-wide rib pillars between the stopes. Note that 100m is double the expected 50m inter-drive span for the shallower BLR areas. The models are, thus, conservative.

The displacement in elastic models is directly proportional to the Young’s modulus. The average Young’s modulus for the rock mass in the Waterberg Project area is about 80GPa. This was derived from laboratory tests and was used as input to the models for the far field deformation calculations.

A Displacement Discontinuity (DD) grid was modelled to simulate surface for monitoring surface displacements directly. This DD grid is 150m above the stope. Two vertical grids were modelled to measure displacements between surface and the stopes. The modelled geometry is shown in Figure 16-18.

An alternate approach to assessing surface subsidence considers the tensile zone that exists above a mined stope. The horizontal stress state above the stope is assessed to determine if one (or both) of the major horizontal components is negative (tensile).

As the ratio of depth to mining span ratio (H/L) reaches a critical lower value, the modelled tensile zone is expected to break through to surface and tributary area loading conditions are deemed to apply. It is under these conditions that surface subsidence is likely.

16.3.3.2 Stope design and pillar spacing

16.3.3.2.1 Hydraulic Radius and Design Charts

Hydraulic radius (HR) is used to guide mine design in terms of stable stope span. HR is defined as the area of a stoped face divided by its perimeter. The result, having the SI unit meter, is a single number representing both the size and the shape of an exposed stope face. Many different combinations of length and width of rectangular areas are possible for a given hydraulic radius. The Critical Hydraulic Radius (CHR) defines the boundary between a potentially stable and a potentially unstable surface wall for a given rock mass quality.

Empirical charts based on rock mass quality and HR has been established and is extensively used in industry to guide stope design. The maximum unsupported stable span of the stope hanging wall and footwall was determined empirically using the two most widely accepted methods, namely Laubscher's caving chart, as described in Brown (2000) and otherwise known as the MRMR method, and the Matthews-Potvin N' stability number method, as described in Potvin (1988).

16.3.3.2.2 Designing within the Transition Zone

It was recognised early in the study that, provided it is safe, indefinite stope stability should not be a requirement. Instead, the rock walls can be allowed to fail once the ore has been extracted and the spans can be designed to a hydraulic radius that lies within the Transition Zone where there is a higher probability of failure but not a great likelihood of continuous caving to surface. For the Waterberg Project, the stope spans were deliberately designed to lie within the Transition Zone in order to maximise extraction whilst reducing the chances of stress lockup in the back and possible pillar bursting.

16.3.3.2.3 Design Charts as applied to the BLR Design

The BLR mining method is described in section 16.3.2.2. The BLR assumes a skin-to-skin spacing of 50m between secondary roadways. The longest distance along strike will be about 300m. This is shown in the diagram in Figure 16-18. Depending on whether hydraulic radius is measured on plan or on reef plane, these dimensions result in an open stope hydraulic radius of between 17m and 22m.

For indefinite stability of the stope walls, the CHR must be between 9.0m and 11m. Since each stope will be mined from span centre back towards the secondary roadway, the maximum hydraulic radius of 17 - 22m will only be achieved when the last few blasts of the final retreat stope are taken.

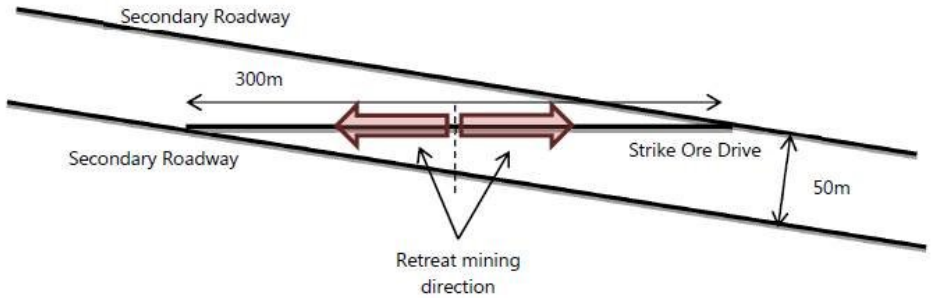


Figure 16-18: Diagrammatic Representation of BLR Layout on Plan View

16.3.3.2.4 Design Charts as Applied to the SLOS Design

The considerations applicable to BLR also apply to SLOS, but whereas the BLR method relies on the protection/stability pillars along the secondary roadways for regional stability, the span for SLOS is controlled by the dip pillars and the sill pillars. In addition, a measure of control can be had by controlled drawing of broken ore to minimise sloughing.

The current design calls for an on dip skin-to-skin dimension of the stope hanging wall of 103m (based on a reef dip of 40°), with four or five sub-levels spaced between the sill pillars. The skin-to-skin strike distance between rib pillars is 80m. These dimensions result in a hydraulic radius of between 20m and 23m, again, depending on whether hydraulic radius is measured on plan or on reef plane. Given rock mass quality MRMR=42 and N'=8.5-9.2, the required CHR for indefinite stability of the stope walls is between 8.0m and 11m. A CHR above 20m, therefore, has a high probability of collapse.

16.3.3.2.5 Conclusions on the Analysis of Stable Span

It must be understood that designing in the Transition Zone carries risk that is considerably higher than designs aimed for the stable zone because of high uncertainty around many variables. The analyses, however, show that there is a high probability of success for both mining methods understanding that precautionary measures will be required. These include:

- The mining will require careful planning with particular emphasis on stope sequencing. Strict control will be required as adherence to the planned sequence will be imperative;
- Ground movement monitoring systems and appropriate data analysis;
- Loading should be done with remote controlled machines; and
- Going back into mined areas must be prohibited and effectively controlled.

These precautions should ensure safe, high extraction mining. More work is required in assessing a more representative stand-up time estimate, especially in the case of the SLOS design stopes.

16.3.3.3 Pillar Dimensions

There are few reliable empirical methods for determining the dimensions of pillars in open stopes. The well-known pillar formulae that are available are applicable primarily to flat dipping, narrow and tabular orebodies. Most practitioners use numerical models to estimate the strength of pillars and predict when they will fail.

This was done for the Waterberg Project. Map3D and Phase 2 were used to analyse various aspects of the mine design. Although the focus was mainly on pillar design, the empirical stability analyses of unsupported stope spans (section 16.3.3.2) were also tested.

16.3.3.3.1 Modelling Assessment of BLR Mining

The models were situated at 600m below surface. It was assumed that the virgin stresses were equal in all directions (k=1) and increased with depth at a rate of 0.03MPa/m.

The input parameters for the Map3D models, based on laboratory results and other geotechnical data as contained in the geotechnical database, are listed in Section 16.3.1.4.

The outcome of the geotechnical analysis is that, as the design stands, the protection pillars will fail fairly quickly after having been cut if they are not confined with artificial support immediately. If adequately controlled, however, it will still be possible to access the broken ore from the stope when the stope face is very close to the secondary roadway.

Figure 16-19 shows the mining configuration at an intermediate stage of BLR mining. It indicates the area that can be expected to have collapsed at this stage of mining. The model shows that collapse will be extensive, as expected, but that there should be sufficient opportunity to remove blasted ore from the drives that are still active (areas to the right of the solid white line). These results generally support the findings of the empirical analyses. It is assumed that remote controlled machinery will be used as this limits exposure of personal to risk.

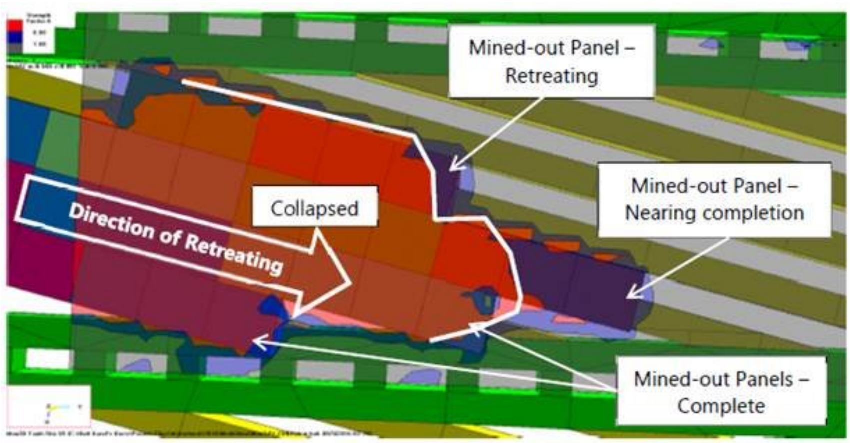


Figure 16-19: Extent of Hanging Wall Failure Predicted by the Model

16.3.3.3.2 Modelling Assessment of SLOS Mining

In SLOS, the function of pillars is to divide mining into manageable districts in which safe mining can take place. Two types of pillars define the boundaries of an open stope. These are sill pillars, which separate sub-levels in a vertical sense, and rib pillars, which separate stopes in the horizontal sense.

For this study, the dimensions of stopes and pillars are assumed the same for both longitudinal and transverse SLOS stope layouts and therefore will not be discussed separately. It is noted that this assumption only holds true for transverse layouts where the width of the targeted reef is not excessive compared to a typical maximum 15m-wide longitudinal layout. Based on the current mine layouts the maximum reef thickness is limited to approximately 25m.

Should transverse layouts target wider mining areas approaching 50m to 80m, then it is expected that sill pillar dimensions must increase and further assessment will be required.

Practical mining considerations require that ore drives should not be more than about 15m apart (footwall to footwall) to ensure effective drilling. The optimised mining layout requires four ore drives per sub-level. These constraints result in a 66m vertical distance between sill pillars. At 40° inclination, this comes to about 103m of mining span on dip. This fixes the vertical dimension for the open stope and only the strike dimension between rib pillars can be varied to keep the open stope spans within the limits imposed by the empirical hydraulic radius criterion. An 80m strike distance between the rib pillars was modelled based on the empirical work.

Map3D was used to determine the optimum size of rib pillars for open stopes under these conditions and to validate the empirical calculations. A basic model configuration representing the longitudinal SLOS layout is shown in Figure 16-20. The mining was simulated in blocks of 10m and pillar failure was monitored. The required rib pillar dimension was then judged one step before the failure propagated through the core of the pillars. This dimension (rib pillar width) was found to be 20m.

A similar model was used to assess sill pillar. The required sill pillar width in order to prevent complete failure of the core was found to be 15m.

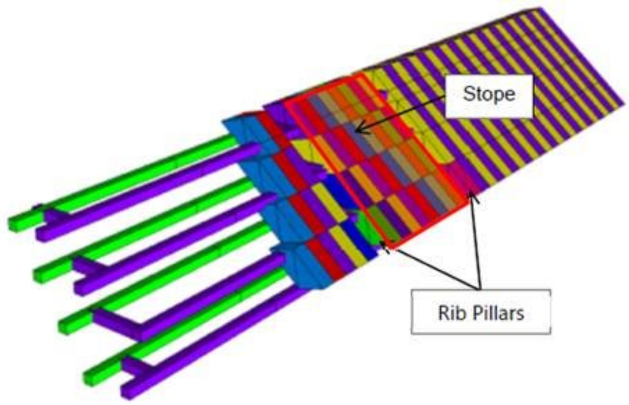


Figure 16-20: Example of Modelled Mining Configuration for Longitudinal Open Stoping

16.3.3.3.3 Findings on Pillar Dimensions

Map3D modelling of strength factor (overstressing) has been used to determine the following pillar dimensions tabulated in Table 16-15.

Table 16-15: Summary of Pillar Dimensions

Mining Method	Pillar Type and Dimensions	Comment
BLR	Drive protection pillars (minimum spacing of 8.0m skin-to-skin)	To be verified during the next stage of study.
SLOS Longitudinal	Rib pillars (20m), Sill pillars (15m)	Assumes maximum mining width of 15m. Relies on shallow dipping reef (40°).
SLOS Transverse	Rib pillars (20m)*, Sill pillars (15m)*	Assumes that the increase in mining width is not excessive. Further work is required during the next stage of study.

The conducted assessments are limited to the following assumptions:

- a shallow dipping reef (< 40°); and
- A maximum mining thickness of about 25m.

More detailed numerical assessment and verification work will be required during the next stage of study.

16.3.3.4 Raise Bore Stability

It is planned to effect ventilation shafts by means of raise boring and it is therefore necessary to investigate the likelihood of long term stability of raise bore holes in the Waterberg rock types. To this end, the geotechnical risk assessment techniques for raise bored shafts as proposed by McCraken and Stacey (1989) were used. The analysis is fully described in Louw (2016) and the conclusions of the analysis were as follows:

- For a 4.2m diameter ventilation shaft to be stable, it requires the QR rating of the rock mass to exceed 3.3;
- The percentage of instability in a raise bore can vary between 10% and 50%. Although these values may seem high it should be noted that such instabilities will occur over relatively short, in-contiguous zones;
- Dedicated boreholes at the raise bore positions were not available and the evaluated boreholes were limited to what was available. The evaluation was carried out to get a feel of likely ground conditions for raise boring. The variation in the percentage of instability highlights the requirement for a geotechnical drill hole at each ventilation shaft location to confirm the potential for instability;
- The collar area of holes (weathered zone) and the contact area between the sediments and igneous sequence are zones with potential for instability; and
- Once boreholes have been drilled at the locations of ventilation shafts, risk assessments will be required to evaluate the potential impact of instabilities and whether mitigation measures, such as ground support, will be required.

16.3.3.5 Main Triple On-reef Declines

Access to the ore will be by means of triple on-reef declines, 6.0m wide by 6.0m high, developed on apparent dip of the reef to accommodate machinery. The triple decline system is necessitated by ventilation requirements. A typical BLR configuration is shown in Figure 16-21.

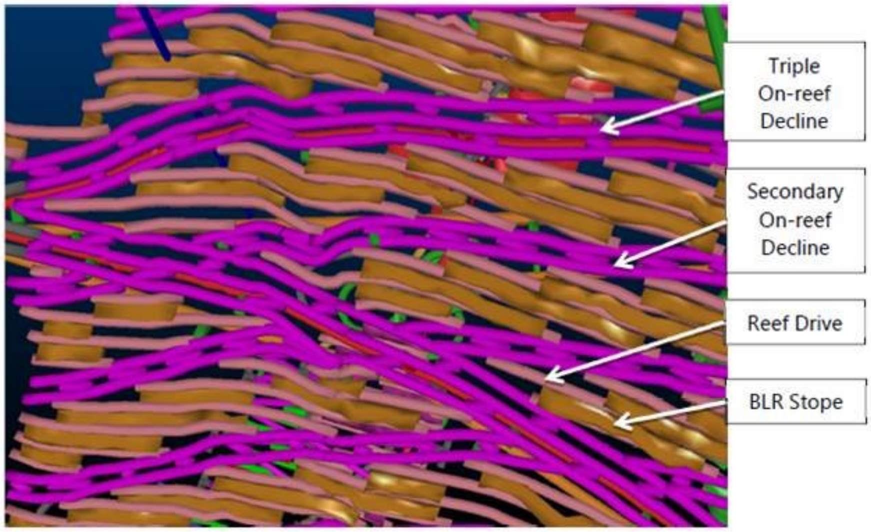


Figure 16-21: Typical BLR Mining Layout (not to scale)

The pillar configuration envisaged is similar to that for the secondary declines (refer to Section 16.3.3.3.1 where the skin-to-skin distance between declines is recommended as 8.0m). The distances between the three declines will vary from 6.0m above 300m depth to 12m where the depth exceeds 700m, to accommodate increasing stress conditions. This automatically results in larger pillars overall because there will be two pillars between the declines as opposed to one in the case of the twin secondary roadways.

16.3.3.6 Ground Control and Support Requirements

Under the Waterberg conditions, 2.4m-long bolts will be required, installed 1.5m apart on a square pattern. Bolts should be full column resin grouted. A diagram of the most common expected support regime is given in Figure 16-22. This support is adequate for all excavations up to a width of 8.0m and a height of 6.0m.

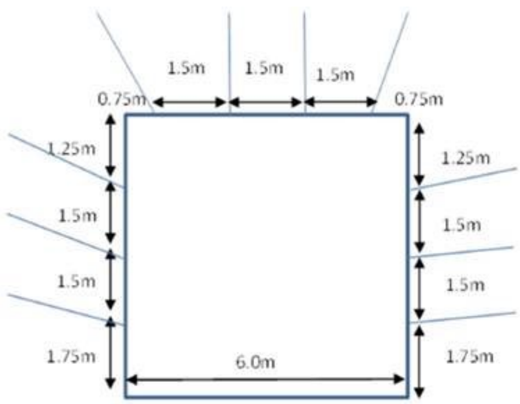


Figure 16-22: Diagrammatic Cross-sectional Representation of Support (not to scale)

In summary, the following is suggested about tunnels support:

- The most common support system recommended is shown in Figure 16-22. Bolts will be 2.4m long, 20mm-diameter mild steel or 14mm-diameter high-tensile steel resin bolts spaced 1.5m apart in rings 1.5m apart. In the main access declines and where adverse conditions are encountered, the spacing might have to be reduced to 1.0m by 1.0m;

- In stope pillars may require stabilisation by means of bolting, meshing and strapping before the pillars are allowed to yield. Strapping can consist of de-stranded hoist rope or lacing cable;

- Cable anchoring might be required in adverse ground, at tunnel intersections and in larger excavations (where spans exceed 8.0m or height exceeds 6m); and

- Surface support will be required at the portals and the initial parts of the declines. This will consist of mild steel mesh or shotcrete (fiber reinforced where required) of about 50mm to 100mm thick, or 8.0mm thick thin sprayed lining.

16.3.3.7 Tunnel Dimensions and Spacing

The geotechnical information acquired through this study is adequate to inform generic ground control guidelines. However, detailed layout and site-specific support design of Life of Mine (LoM) tunnels and large excavations will require geotechnical and geological studies through dedicated drilling and core logging very early or even prior to the feasibility study phase.

There will be weaker zones of local extent, which will require treatment and may result in minor schedule delays; at this level of study, these can be provided for through contingency planning. Based on the impression of the rock mass and its expected response to mining, this contingency should not be more than 15% if the perceived occurrence of olivine-rich shear zones is greater than expected.

Within the space of these perspectives, it can be assumed that the rock mass for off-reef excavations will not deviate from general industry experience in sandstone and bushveld lithologies; hence, empirical and general thumb rules have been applied to arrive at the criteria used for ground control requirement estimates.

16.3.4 Rock Mechanics Parameters for Mine Design

The full set of rock mechanics parameters for mine design, as detailed in Balt (2016), are presented in Table 16-16.

Table 16-16: Rock Mechanics Parameters for Mine Design

Item	Domain	Dol SIL	Seds	T_HW	T_Min	T_FW	MZN	F_HW	F_Min	F_FW
Rock Mass Rating Rock Strength	RQD	48.9	54.3	88.9	83.7	84.7	63.7	78.1	81.5	81.0
	Q'	2	3.2	7.8	6.3	5.0	3.2	4.5	4.6	14.1
	RMR89	51	49	56	56	55	52	55	57	64
	Laubscher's MRMR90	38	37	42	42	42	39	42	43	49
	N' for Stope Hanging Wall			8.5 – 9.5				8.5 – 9.5		
	N' for Stope Back				3.6 – 4.6				3.6 - 4.6	
	Stable CHR (m) MRMR-based (plan view)			10-11	10-11	10-11		10-11	10-11	10-11
	Stable CHR (m) N'-based (reef perp. view)			10-11	10-11	10-11		8-9	6-8	>11
	UCS (ave nearest 5MPa)		120	120	150	160	160	160	190	210
	DRMS (nearest 5MPa)			60	75	80		80	75	95
Modelling Parameters	Young's Modulus (GPa)	80	80	80	80	80	80	80	80	80
	Poisson's Ratio	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
	Hoek-Brown m	0.295	0.295	0.295	0.295	0.295	0.295	0.295	0.295	0.295
	Hoek-Brown s	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
BLR	Pillar Dimensions (m)				8.0m normal to the declines x12 along the declines				8.0m normal to the declines x12 along the declines	
	Pillar Spacing (m skin-to-skin)				±50			±50		
	Max Mining Height (m)				15			15		
	Minimum Distance between Reef Drives	8	8	8	8	8	8	8	8	8

Item	Domain	Dol SIL	Seds	T_HW	T_Min	T_FW	MZN	F_HW	F_Min	F_FW
SLOS (longitudinal)	Rib Pillar Dimensions (m)				20			20		
	Rib Pillar Spacing (m on strike)				80			80		
	Sill Pillar Dimensions (m vertical)				15			15		
	Sill Pillar Spacing (m on dip)				103			103		
	Standoff Distance for Footwall Drive (m)				20			20		
	Mining Direction				Underhand Towards Crosscut				Underhand Towards Crosscut	
SLOS (transverse)	Rib Pillar Dimensions (m)				20			20		
	Rib Pillar Spacing (m on strike)				80			80		
	Sill Pillar Dimensions (m vertical)				15			15		
	Sill Pillar Spacing (m on dip)				103			103		
	Standoff Distance for Footwall Drive (m)				20			20		
	Mining Direction				Underhand Towards Footwall				Underhand Towards Footwall	
Main On-reef Declines (triples)	Middling Depth up to 300m	6	6	6	6	6	6	6	6	6
	300<Depth<=400		8	8	8	8	8	8	8	8
	400<Depth<=500		9	9	9	9	9	9	9	9
	500<Depth<=700		10	10	10	10	10	10	10	10
	700<Depth					12	12	12	12	12

16.3.5 Rock Mechanics Related Risks and Forward Work Recommendations

A very detailed table of identified rock mechanics related risks associated with the present study is presented in Balt (2016) together with connected recommended forward work. This section merely summarizes the outcome from this risk assessment and forward work plan. Please refer to Balt (2016) for full detail.

16.3.5.1 Site Investigation Work

To better inform design during the next stage of study it is recommended that a geotechnical drilling and logging programme be designed and implemented before the commencement of a feasibility study:

- Additional drilling focus is necessary at the T reef;
- Orientated core drilling and Geotechnical logging of joint properties to allow for joint stability analysis;
- Dedicated drilling and geotechnical core logging aimed at:
  - The portal location;
  - The main triple decline route;
  - The raise bore locations; and
  - Underground shaft infrastructure and other large chambers.
- Structural geology model (3D) is to be compiled showing prominent faults, dykes and other major structures. A better interpretation of the observed olivine-rich shear zones is required. This would be the responsibility of a suitably qualified Geologist;
- Geohydrological studies should be conducted to assess the potential for water inflow;
- Stress measurements to be conducted with specific focus on determining relevant K ratios. Core can be selected and sent for Acoustic Emissions (AE) testing.

16.3.5.2 Analysis and Numerical Modelling

Basic assessments that are adequate for this level of study have been conducted but further numerical assessment in the following areas is required:

- The 3D block model should be updated with as much information as possible to cover at least all the areas where infrastructure might be located in order for appropriate design of these excavations and their ground support requirements;
- Structure related stability assessments (kinematic analysis at key LoM locations) and impact on support requirements. This relies on dedicated and directed orientated core drilling and geotechnical logging;
- Re assess the BLR and SLOS layouts using the updated geotechnical understanding;
- Case histories of design with the transitional zone. Focus on the issue of stand-up times and the risk of premature failure; and
- Additional assessment raise bore stability based on dedicated centerline boreholes.

16.3.5.3 Operational Systems

In stope ground, control strategies, including monitoring of ground movements and stope evacuation procedures, require investigation that is more intensive.

16.4 Proposed Mining Methods

The dip of the orebody varies from approximately 20° to over 50°. The minimum mining height considered as part of this study is 3m. This was due to the practical limitations of mechanized mobile equipment, especially when the associated dip is also considered. The maximum thickness of the target areas is well in excess of 25m.

Due to the complex nature of the orebody with varying thickness, varying dip and varying grades, a number of different mining methods have been utilized. These include the following:

- Blind Longitudinal Retreat (BLR)
- Sub-level Open Stoping (SLOS) — Transverse
- Sub-level Open Stoping (SLOS) — Longitudinal

16.4.1 Blind Longitudinal Retreat

Discussions and planning has evolved through different mining methods, mainly applicable to the T-reef. Initially the Stepped Room and Pillar (SRP) mining method was preferred for most of the T-reef, however as the project developed it became apparent that the orebody does not lend itself to this mining method in many places.

Due to limitations of the mobile mining equipment, the normal SRP method cannot accommodate parameters where the mining heights are well in excess of 6m. In addition, the dip of the orebody presented challenges with respect to affecting holings, as these could not be adequately supported. It was eventually considered to develop the holings on the retreat, however, that largely obviated the need for pillars and it was decided to investigate the possibility of mining most of the ore on the T-reef and in appropriate areas on the F-reef by means of a revised methodology.

Therefore, a conceptual mining method, termed the Blind Long Hole Retreat (BLR) mining method was developed. The development phase of the BLR mining method is largely based on the SRP mining method. A significant difference between the layouts of SRP and BLR mining methods is that for the BLR layout, a portion of the orebody is left intact to serve as regional pillars. The BLR mining method also contains aspects of the Sub-Level Open Stoping (SLOS) - Longitudinal mining method, especially the production phase.

In order to mine the ore, access between the main underground infrastructure is obtained by means of on-reef roadways and they are designed at an apparent dip. On-reef Extraction Drives (EDs) originating from the on-reef roadways are developed on strike as per the SRP mining method. Each mining block consists of four EDs and a regional pillar. The EDs can be sequenced to advance simultaneously and are developed to the extremity.

Once the EDs are mined to the extremity, preparation for the blind longitudinal retreat can commence. A free braking point is established at the extremity of each ED by creating a slot. A combination of drilling and blasting techniques are utilized for this. A schematic of the development layout including the initial slot is illustrated in Figure 16-23.

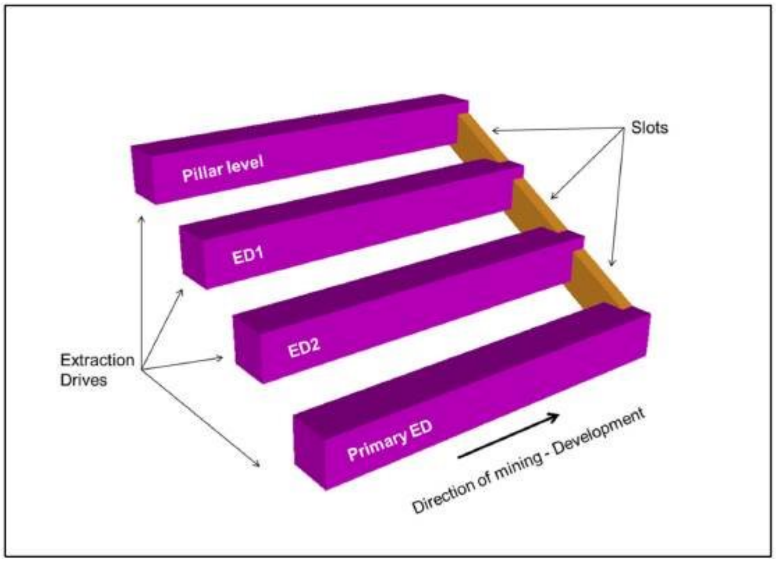


Figure 16-23: Development Layout

Production blasting takes place on retreat with rings drilled ±2 m apart, starting at the slot. Production commences at the top of the mining block, thereby creating the desired lead / lag sequence between EDs in a mining block. The lead / lag are maintained by ensuring that the retreat of each ED leads the one directly below it by two or three rings. A schematic of a BLR section in its production phase is shown Figure 16-24.

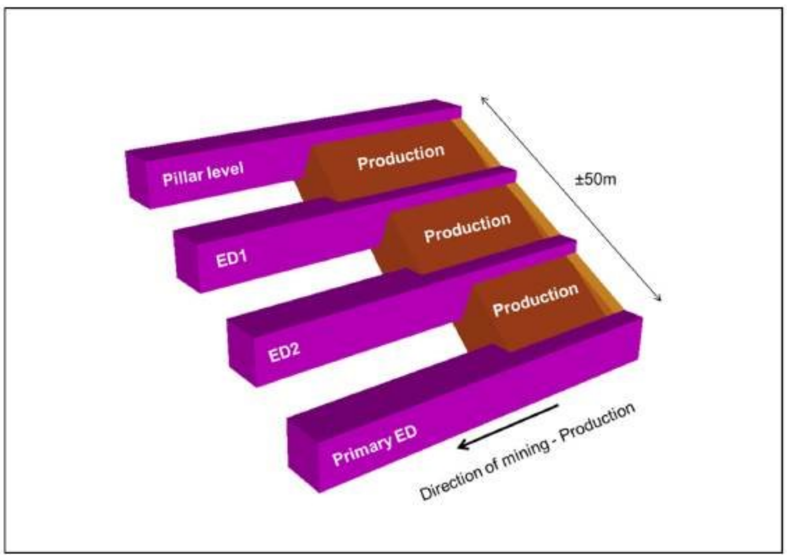


Figure 16-24: Production Phase

Because blasted production stopes will be unsupported, no person will be allowed access to these stopes between the EDs. For the same reason, loading of production ore is planned to be conducted by means of a LHD with a remote loading functionality.

In this study, the BLR mining method will be applied in areas where the mining target area has a thickness not exceeding 15m and a dip not exceeding 35° to the horizontal. Details pertaining to the layout, sequencing and scheduling are discussed in Section 16.6 of this document.

16.4.2 Sub-level Open Stoping (SLOS)

Sub-level Open Stoping (SLOS) is a mechanized mining method suitable for steeply dipping ore bodies of varying thickness. In this study, the SLOS mining method will be applied in areas where the mining target area has a dip exceeding 35 degrees.

A crown pillar is typically planned between the workings and surface. The strike pillar between two stopes is referred to as a sill pillar while the dip pillars separating two stopes are known as rib pillars. A stope typically consists of a pillar level and a number of other sub-levels. These sub-levels are developed from an access ramp situated in the footwall.

16.4.2.1 Sub-level Open Stoping — Longitudinal

In this study, the SLOS - Longitudinal mining method will be applied in areas where the mining target area has a thickness not exceeding 15m and a dip in excess of 35° to the horizontal. Details pertaining to the layout, sequencing and scheduling are discussed in Section 16.6 of this document. The size of the mining blocks is 80m (strike) and approximately 100m (dip) in size, separated by a 15m sill pillar on strike and a 20m rib pillar.

In order to mine the ore, on-reef EDs are developed on strike as per the BLR mining method. Each ED has an associated off-reef footwall drive, depending on the strike distance. Every 200m a lateral is planned between the two ends; this provides access to the on-reef EDs and enables production whilst development is still in progress.

Four or five (depending on the dip) on-reef EDs are developed per mining block, one per sub-level. Each EDs is developed for a distance of approximately 100m in each strike direction and the EDs on the various sub-levels can be sequenced to advance simultaneously.

Once the EDs are mined to the extremity, preparation for the longitudinal retreat can commence. A free braking point is established at the extremity of each ED by creating a slot. A combination of drilling and blasting techniques are utilized for this. A schematic of the development layout including the initial slot is illustrated in Figure 16-25

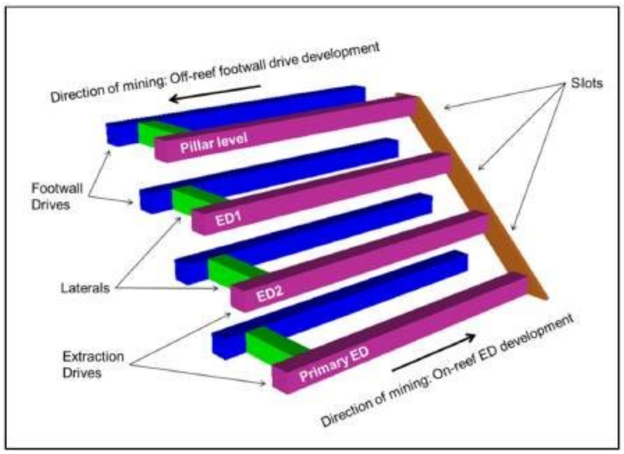


Figure 16-25: SLOS Longitudinal — Development Layout

Similar to the BLR mining method, production blasting takes place on retreat with rings drilled ±2 m apart, starting at the slot. Production commences at the top of the mining block, thereby creating the desired lead / lags sequence between EDs in a mining block. The lead / lag are maintained by ensuring that the retreat of each ED leads the one directly below it by two or three rings. A schematic of a SLOS — Longitudinal section in its production phase is illustrated in Figure 16-25.

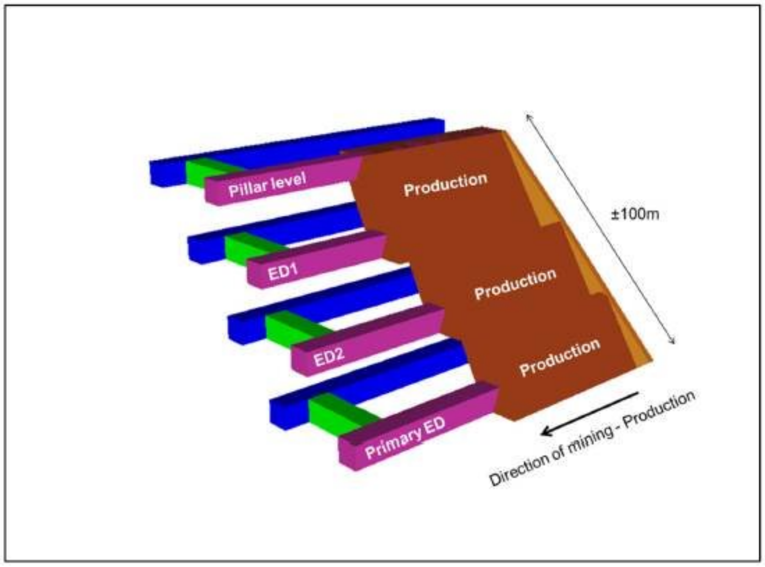


Figure 16-26: SLOS Longitudinal - Production Phase

As with the BLR mining areas, the blasted production areas will be unsupported and therefore no person will be allowed access to these stopes between the EDs. In addition, loading of production ore is planned to be conducted by means of a LHD with a remote loading functionality.

16.4.2.2 Sub-level Open Stoping — Transverse

In this study, the SLOS - Transverse mining method will be applied in areas where the mining target area has a thickness in excess of 15m and a dip in excess of 35° to the horizontal. Details pertaining to the layout, sequencing and scheduling are discussed in Section 16.6 of this document.

The size of the mining blocks is 80m (strike) and approximately 100m (dip) in size, separated by a 15m sill pillar on strike and a 20m rib pillar.

For this variation of the sub-level open stoping mining method, on-reef EDs are developed through the target areas as opposed to along the target area for the SLOS — Longitudinal mining method.

Four sub-levels are developed per mining block, which can be sequenced to advance simultaneously. Each sub-level has an off-reef footwall drive, the on-reef EDs originate from the respective footwall drives;

Four on-reef EDs are developed per sub-level for each mining block. The EDs can be sequenced to advance simultaneously and are developed to the extremity of the mining area. The development layout of two mining blocks utilizing the SLOS — Transverse mining method is shown in Figure 16-27

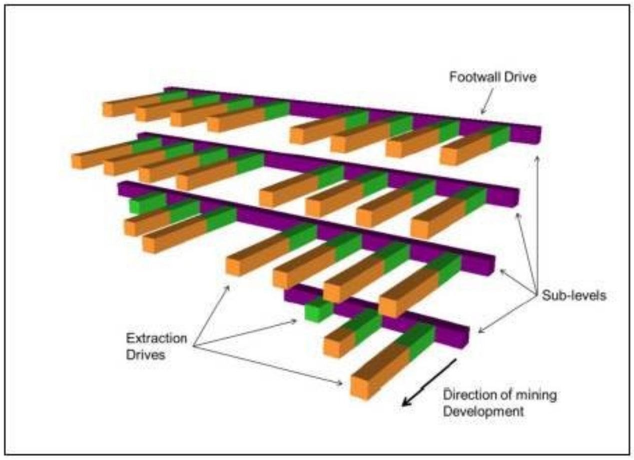


Figure 16-27: SLOS Transverse — Development Layout

The schematic shown in Figure 16-27 illustrate a point in time where the development is still in progress and not all the EDs have reached their extremities yet, specifically the lower sub-levels. The excavations shown in purple represent the off-reef footwall drives. The ends shown in green are the off-reef portion of the EDs while the on-reef portion of the EDs is illustrated in a brown color.

As soon as the EDs on a sub-level have been mined through the target mining area, preparation for the transverse retreat can commence. A free braking point is established at the extremity of each ED by creating a slot. A combination of drilling and blasting techniques are utilized for this. A schematic of the development layout where the slots have been established in one mining block is illustrated in Figure 16-28. It should be noted that for this specific scenario, development of the mining block on the left in the schematic is still in progress now.

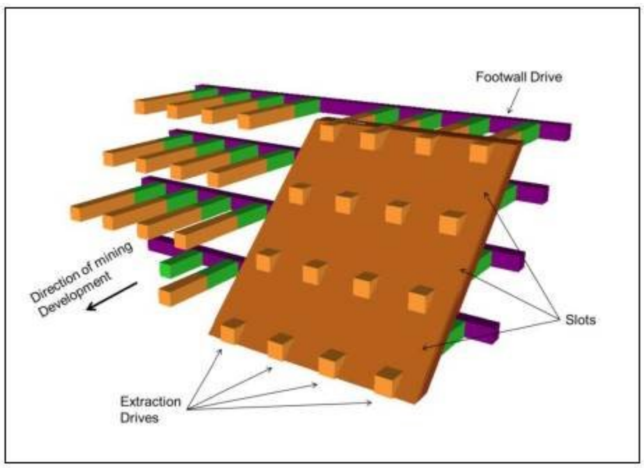


Figure 16-28: SLOS Transverse — Development Layout with Slot

With this mining method, production blasting also takes place on retreat with rings drilled ±2 m apart, starting at the slot. Production commences at the top of the mining block, thereby creating the desired lead / lags sequence between sub-levels in a mining block. The lead / lag are maintained by ensuring that the retreat of each sub-level leads the one directly below it by two or three rings.

In this case, however, the dip of the reef could affect the sequencing as a natural lead / lag is created by default. A schematic of a SLOS — Transverse mining area in its production phase is illustrated in Figure 16-29.

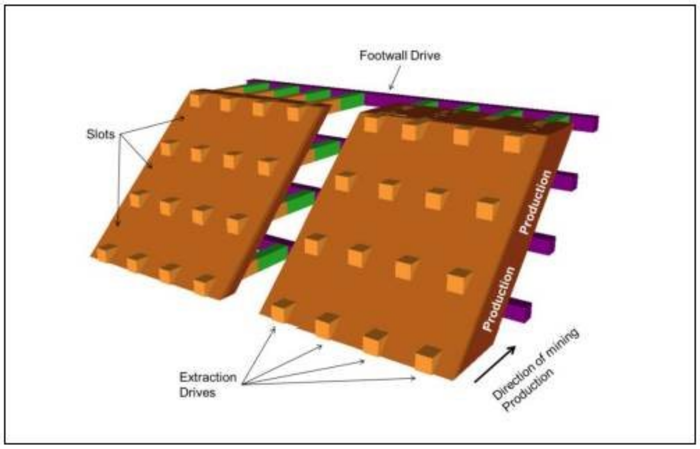


Figure 16-29: SLOS Transverse — Production Phase

In this study, the SLOS - Transverse mining method will be applied in areas where the mining target area has a thickness exceeding 15m and a dip in excess of 35° to the horizontal. Details pertaining to the layout, sequencing and scheduling are discussed in Section 16.6 of this document.

Similar to the other mining methods adopted for this study, the blasted production areas will be unsupported and therefore, loading of production ore is planned to be conducted by means of a LHD with a remote loading functionality. No person will be allowed access to these stopes between the EDs.

16.5 Required Mining Fleet and Machinery

The aim of this section is to list the prescribed mine equipment that is selected by size and type for the final mine plan layout. It should be noted that no supplier agreements are in place presently and the equipment selected is on a best suit basis. A full equipment tender process will be required when the Project proceeds to the next study phase.

16.5.1 Selected Mobile Equipment

The major mobile equipment for the mining activities will consist of drill rigs (development, slot establishment and production rings), support rigs (primary and secondary support), LHDs and dump trucks. Other equipment relevant to this section includes charging vehicles, scissor lifts and supervision vehicles. The equipment selected for the purpose of this study are listed Table 16-17.

Table 16-17: Selected Mobile Mining equipment

Area	Function	OEM	Model	Remarks
	Face drilling	Sandvik	DD321	Split feed
	Charging-up	AARD	UV80 Charging vehicle carrier	Carrier for charging unit
		BME	Lateral charger with lifting basket	Installed on carrier
Development	Loading	Sandvik	LH517	
	Hauling	Sandvik	TH540	
	Primary support	Sandvik	DS311	Resin bolts
	Secondary support	Sandvik	DS421	Cable bolting
	Slot drilling	Sandvik	DU421	Drilling of slot for all mining methods
	Production drilling	Sandvik	DL321	Drilling of production rings for all mining methods
Production	Charging- up	AARD	UV80 Charging vehicle carrier	Carrier for charging unit
	BME	Uphole charger	Installed on carrier	
	Loading	Sandvik	LH517	Remote controlled capabilities required
	Hauling	Sandvik	TH540	
	Material transport	AARD	UV80 Cassette carrier	Used to transport: General purpose cassette, Emulsion transfer cassette and lubrication cassette
	Underground construction	AARD	UV80 Scissor lift	Scissor lift
Services	Water truck	AARD	UV80 Water truck	Roadway maintenance
	Grader	AARD	Grader	Roadway maintenance
	Personnel transport	AARD	UV80 Men Dedicated Carrier	38 Seater
	Supervision	TBC	TBC	

16.5.2 Development Mobile Mining Equipment

The build-up and the utilization schedules of the mobile mining equipment, required for development, production and services such as underground mining construction and supervision, are discussed in this section.

16.5.2.1 Equipment fleet build-up

Initial capital development will be undertaken by contractors who will supply their own mobile mining equipment. When the first production ore is produced, mining will be conducted by the owner utilizing its own equipment.

The initial purchasing schedule and build-up of the fleet is shown in Table 16-18

Table 16-18: Initial Purchasing Schedule

Function	Peak	Yr 1-3	Yr 4	Yr 5	Yr 6	Yr 7
Development						
Face drill rig	**22**	0	9	10	3	
Primary support rig	**22**	0	9	10	3	
Charging vehicle	**11**	0	5	3	3	
LHD	**10**	0	4	5	1	
Dump truck	**22**	0	8	10	4	
Secondary support rig	**5**	0	4	0	1	
Production						
Slot drilling rig	**8**	0	3	3	2	
Production drill rig	**15**	0	5	5	2	3
Charging vehicle	**6**	0	4	0	1	1
LHD	**16**	0	3	6	3	4
Dump truck	**20**	0	5	10	5	
Services						
Underground mining construction	**34**	0	12	16	7	
Supervision	**24**	0	8	11	5	1

The equipment shown in Table 16-18 only reflects the fleet, which will be required for owner mining.

16.5.2.2 Equipment Utilization Schedule

The utilization schedule of the mobile mining fleet has been summarized in Table 16-19 Equipment required for the initial capital development by contractors is not reflected here.

Table 16-19: Utilisation schedule

Name	Totals	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19
Development																				
Face drill rig	DD321	0	0	0	9	19	22	17	16	14	16	15	17	13	15	14	13	11	3	1
Primary support rig	DS311	0	0	0	9	19	22	17	16	14	16	15	17	13	15	14	13	11	3	1
Charging vehicle	AARD & BME	0	0	0	5	8	11	7	7	6	7	7	7	7	8	8	7	6	3	4
LHD	LH517	0	0	0	4	9	10	8	8	7	9	8	8	6	8	7	7	6	2	1
Dump truck	TH540	0.0	0.0	0.0	8.4	19.3	23.7	18.9	17.5	15.6	19.4	17.9	19.6	14.3	17.7	16.5	14.2	12.2	3.3	0.5
Secondary support rig	DS421	0	0	0	4	4	5	5	5	5	5	5	5	5	5	5	5	4	3	4
Production																				
Slot drilling rig	DU421	0	0	0	3	6	8	5	6	7	7	6	7	5	5	6	5	4	2	0
Production drill rig	DL321	0	0	0	5	10	12	15	15	15	14	15	12	14	13	12	11	8	3	1
Charging vehicle	AARD & BME	0	0	0	4	4	5	6	6	6	5	5	5	6	5	5	5	5	3	1
LHD	LH517	0	0	0	3	9	12	16	16	16	14	15	13	15	13	14	13	7	3	1
Dump truck	TH540	0.0	0.0	0.0	4.3	13.5	17.7	22.6	23.3	24.0	20.8	22.2	19.1	22.6	19.6	20.5	19.6	9.9	4.7	0.1
Underground Mining Services																				
Underground mining construction	UV80 Scissors lift	0	0	0	12	27	34	32	32	30	30	30	30	27	28	28	25	17	6	1
Supervision	TBC	0	0	0	8	18	23	24	24	23	22	23	21	21	21	21	19	12	5	1
Totals																				
Drill rigs		0	0	0	30	58	69	59	58	55	58	56	58	50	53	51	47	38	14	7
LHD		0	0	0	7	18	22	24	24	23	23	23	21	21	21	21	20	13	5	2

Name	Totals	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19
Dump truck		0	0	0	13	33	42	42	41	40	41	41	39	37	38	37	34	23	9	1
Other (UV's, charging, supervision)		0	0	0	29	57	73	69	69	65	64	65	63	61	62	62	56	40	17	7
TOTAL UNITS		**0**	**0**	**0**	**79**	**166**	**206**	**194**	**192**	**183**	**186**	**185**	**181**	**169**	**174**	**171**	**157**	**114**	**45**	**17**

16.6 Mine Design and Scheduling Parameters

Mine design parameters are based on a number of factors including geotechnical recommendations, conceptual ventilation requirements and capabilities of mobile mining equipment. These factors are used to provide guidance in terms of size, inclination, turning radii as well as the positioning of underground excavations.

Scheduling parameters are based on rates from other projects with relevant similarities. The other main determinant of scheduling parameters is operating cycles.

16.6.1 Access Development

16.6.1.1 Design Parameters

During the initial stages of this PFS a trade-off study was conducted to compare a vertical shaft system to a set of twin declines as means of main underground access for PTM Waterberg. Due to initial capital costs and schedule to commissioning, the study recommended twin declines. Therefore, all mining areas have been accessed by means of twin declines.

The mine design parameters applicable to development in the PFS are summarized in Table 16-20. Design parameters related to Rock Mechanics are discussed in Section 16.3 of this report.

Table 16-20: Mine Design Parameters — Development

Item	Design parameter	
Access development and layout	Twin declines	
Management of development waste	Initially waste will be trucked to an appropriate site on surface; As soon as depleted underground production areas are abandoned, waste can be dumped in these locations.	
Development radii	≥30m	
Development gradients	Main access declines	9°
	Conveyor declines	9°
	Access spirals	9°
	On-reef roadways	9°
	Lateral development (Max)	9°
	Lateral development (Min)	0°
	Ventilation raises	>35°
Excavation sizes	Main access declines	6.0m(h) x 6.0m(w)
	Access development between declines and spirals	6.0m(h) x 6.0m(w)
	Primary conveyor declines	5.5m(h) x 5.5m(w)
	Spirals	6.0m(h) x 6.0m(w)
	On-reef roadways (BLR)	6.0m(h) x 6.0m(w)
	Sub-level footwall drives	6.0m(h) x 5.2m(w)

Item	Design parameter	
	On-reef EDs	6.0m(h) x 6.0m(w)
	Other development	Height: 5.5 - 6.0m Width: 5.2 - 6.0m
	Ventilation raises	3 — 5.1mØ
Planned overbreak	Height	5%
	Width	5%
Densities	In-situ (dry)	2.8t/m ^{3}
	Bulk (volume)	1.6t/m ^{3}
	Bulk (load)	1.8t/m ^{3}

Although the development gradient of lateral development is indicated as 0°, and these excavations have been designed accordingly in this study, a very gentle gradient of approximately 1:200m will be applied to these excavation types. This is to prevent the accumulation of water on the footwall and to enable the flow thereof by means of gravity.

16.6.1.2 Scheduling parameters

The scheduling parameters applicable to the access development in the PFS are summarized in Table 16-21.

Table 16-21: Scheduling Parameters — Development

Item	Scheduling parameter	Remarks
Main declines from surface and conveyor declines	100m system advance	Typically single end availability
All development between main declines and sub-level development, including spirals	130m per crew per month (maximum of 65m per single end)	This scenario assumes that there are typically two ends available
Sub-level and on-reef development	195m per crew per month (maximum of 65m per single end)	This scenario assumes that there are multiple ends available at any point in time
Raise boring	55m per month	Includes set-up, pilot and reaming activities
Drop raising	30m per month	Includes drilling and blasting
Silos and Settlers	12m per month	This includes raiseboring, sliping and installation of support

16.6.2 Production Areas

16.6.2.1 Design parameters

The mining method applied to a specific area is based on two criteria namely, mining height of the target area and dip of the reef. More detail is provided in Table 16-22.

Table 16-22: Selection Criteria for Mining Method

Mining method	Vertical mining height	Dip of the reef
BLR	3 - 15m	≤35°
SLOS - Longitudinal	3 - 15m	>35°
SLOS - Transverse	>15m	>35°

The average dips and mining heights for the various mining methods are summarized in Table 16-23. This is based on information obtained from the 3 g/t cut-off MSO wireframes.

Table 16-23: Average Dip and Mining Height per Mining Method

Mining method	Vertical mining height	Dip of the reef
BLR	5.5m	29.6°
SLOS — Longitudinal	11.3m	40.5°
SLOS — Transverse	26.8m	41.8°

It is important to note that for the purpose of this study, the mining height is measured vertically.

16.6.2.1.1 BLR

An analysis was conducted to find the optimal mining layout for the BLR mining method. Various criteria were considered as part of the analysis with extraction ratio and dilution having the highest weighting. The layout selected as an outcome of the analysis is discussed further.

The main design criteria applicable to the BLR mining method are listed in Table 16-24.

Table 16-24: Design Criteria for BLR Production Areas

Item	Parameter
Horizontal spacing between adjacent EDs	>6m
On-reef span between two sill (strike) pillars	±50m
Number of EDs per mining block	4

The on-reef span between two regional pillars is planned at approximately 50m. A minimum horizontal middling (skin to skin) of 6m is planned between two adjacent EDs. Both these parameters are dependent on the dip of the ore body in that specific area.

A typical BLR mining block indicating the target area utilization is illustrated in.

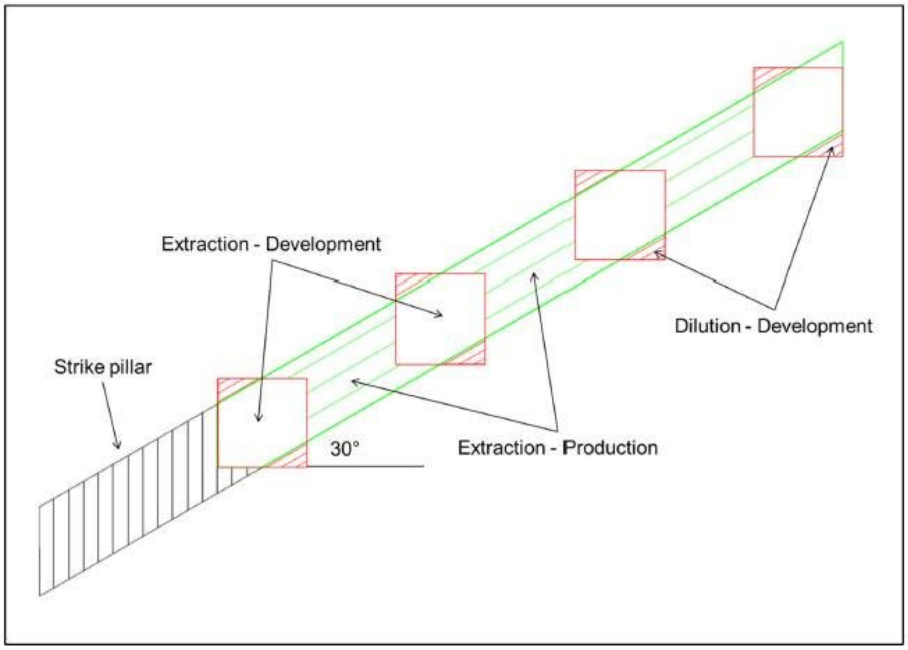


Figure 16-30: Target Area Utilisation

The example referred to in Figure 16-30 represents a production area with an inclination of 30° and a mining height of 6m. These parameters are representative of the averages for the areas where the BLR mining method has been planned.

The expected extraction and dilution percentages, for the BLR mining method, are shown in Table 16-25. The dilution referred to here only allows for the portion of ED development outside of the mining target area. Dilution encountered during the production phase has not been included in these calculations, as this is catered for in the modifying factors. The modifying factors applied are discussed in Section 15.1.8 of this document.

Table 16-25: Extraction and Dilution Ratios - BLR

Parameter	Quantity
Total area (including strike pillar)	324m ^{2}
Total target area	249m ^{2}
Pillar area	75m ^{2}
Development dilution	21m ^{2}
Extraction ratio	**77%**
Development dilution ratio	**8%**

These percentages have been calculated for the specific scenario illustrated in Figure 16-30. It is important to note that the extraction ratio can vary for this mining method and it would be more appropriate to refer to a range as opposed a specific ratio. The extraction ratio in a specific area is impacted by the presence of on-reef roadways, pillar shapes, strike distance of the mining block, dip of the reef and the mining height. The range referred to here will typically vary between 66 and 83%.

16.6.2.1.2 SLOS — Longitudinal

The main design criteria applicable to the SLOS — Longitudinal mining method are listed in Table 16-26.

Table 16-26: Design Criteria for SLOS Longitudinal Production Areas

Item	Parameter
Horizontal spacing between access spiral and target mining area	60m
Horizontal spacing between footwall drive and target mining area	20m
Interval for laterals between footwall drive and on-reef ED	200m
On-reef span between two sill (strike) pillars	± 100m
Strike distance per stope (between two rib pillars)	80m
Strike distance of sub-level required to include off-reef footwall drive	>150m
Vertical spacing (centre to centre) between sub-levels	15 – 20m

The on-reef span between two sill pillars is approximately 100m; the exact distance is dependent on the dip of the ore body in that specific area.

The strike length of the target area on that specific sub-level determines the requirement of an off-reef footwall drive. A footwall drive is planned where the strike length exceeds 150m. Every 200m a lateral is planned between the footwall drive and the on-reef ED; this provides access to the ED and enables production on that sub-level whilst development is still in progress. The footwall drive also serves as a loading area for of trucks and is required for the ventilation circuit. A plan view of a typical layout of a SLOS — Longitudinal sub-level with a footwall drive is illustrated in Figure 16-31.

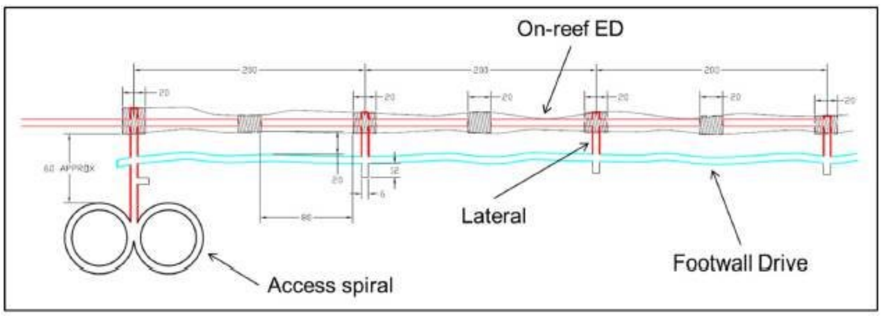


Figure 16-31: SLOS Longitudinal Sub-level with Footwall Drive

For scenarios where the strike distance of a sub-level does not exceed 150m, no off-reef is planned. A plan view of this layout is shown in Figure 16-32.

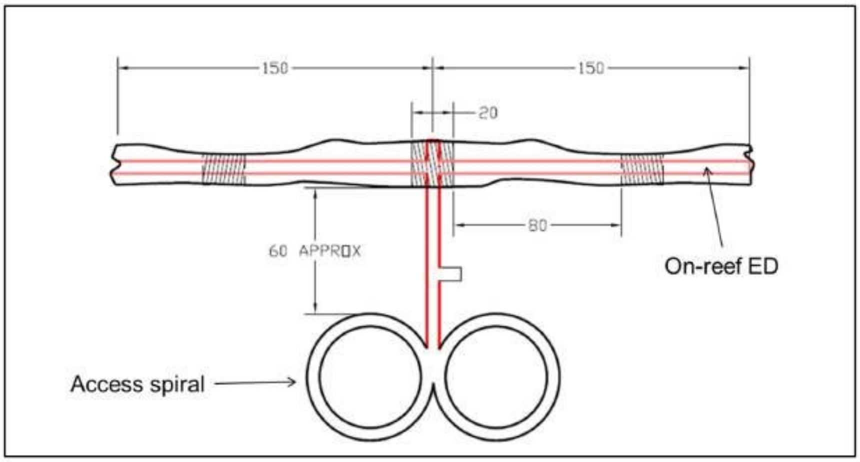


Figure 16-32: SLOS Longitudinal Sub-level without Footwall Drive

As mentioned in Table 16-26, the vertical spacing between two sub-levels varies between 15 and 20m. The two different scenarios are summarized in Table 16-27. The vertical spacing is defined as the interval between sub-levels and is measured from the footwall of the sub-level to the footwall of the sub-level immediately above it.

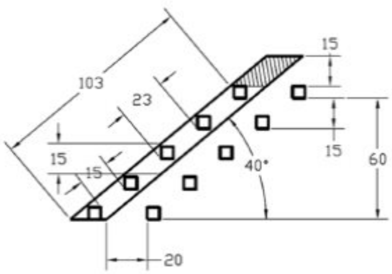
Table 16-27: Sub-level Spacing for SLOS Longitudinal

Dip of the Reef	Vertical Spacing	Number of Sub-levels per Stope
<40°	15m	5
$\geq$40°	20m	4

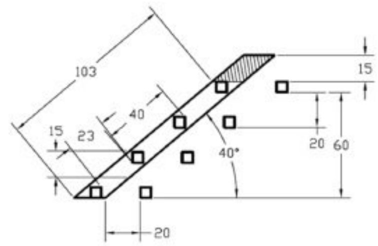
For both the scenarios mentioned in Table 16-27, the on-reef span between two sill pillars remains at approximately 100m. Therefore, for the purpose of this study, the total dimensions, in terms of dip and strike, of an individual stope are not affected by the number of sub-levels.

The impact is that the vertical spacing of 15m translates into five sub-levels per stope while the 20m spacing accommodates four sub-levels per stope. Schematics illustrating the two scenarios are shown in Table 16-28.

Table 16-28: SLOS Longitudinal Sub-level Layouts



15m vertical spacing



20m vertical spacing

The on-reef span between two consecutive sub-levels was the main determining factor in determining these scenarios. The on-reef span can have a significant impact on the following two aspects:

- Maximum length of shot holes, and;
- In-stope mining losses due to the angle of repose.

The criteria listed in Table 16-28 were arrived at by considering these two aspects.

16.6.2.1.2.1 Shot Hole Length

Drilling accuracy of production shot holes is vital regarding the extraction and dilution in the production areas. It is mainly determined by shot hole length, drilling equipment and operator skill levels. Maximum and average shot hole length is a direct function of the vertical spacing between sub-levels.

It is a generally accepted norm in the underground hard rock industry that shot holes can be drilled accurately up to a length of 25m and without excessive deviation. Therefore, 25m was used as a benchmark to compare drilling patterns used in this PFS.

Provisional drilling patterns were designed for the various mining methods. About the dip of the reef, four scenarios were considered for the SLOS Longitudinal mining method. A summary, with specific emphasis on the total drilled meters per ring, is shown in Table 16-29.

Table 16-29: Summary of Shot Hole Lengths — SLOS Longitudinal

Vertical spacing	Number of sub-levels per stope	Dip of the reef	Total drilled metres per ring	% of Total drilled metres >20m	% of Total drilled metres >25m
15m	5	35	325m	4	0
	4	35	452m	21	11
20m	4	40	322m	15	4
	4	45	268m	8	0

The fact that for a 15m sub-level interval, 4% and only 0% of the total drilled meters were more than 20m and 25m respectively, this indicated that there was scope to increase the vertical spacing of sub-levels for specific scenarios.

For the 20m sub-level spacing and a dip of 40°, the four percent of the total drilled meters being >25m was deemed as marginal and it was deemed as a manageable risk with regards to drilling accuracy.

Where a vertical spacing of 20m and a dip of 35° is applied, 11% of the total drilled meters were >25m in length which means that the risk of shot hole accuracy becomes meaningful. However, this percentage decreases as the dip of the ore body increases to an extent that it is zero where the dip reaches 45°.

16.6.2.1.2.2 Mining Losses

Rock Engineering design criteria do not require the installation of any ground support in the on-reef areas between the EDs. This is based on the assumption that no persons will require access to these areas at any point in time. This includes access to handle or move clean blasted rock from the affected areas. Blasted rock would therefore have to be handled and loaded exclusively from the EDs and no reliance could be placed on moving rock from the on-reef areas between the EDs.

It is therefore inevitable that not all blasted rock would be recovered from the on-reef areas between the EDs and those losses would be encountered as a result. The quantum of these losses is determined by the dip of the reef in relation to the angle of repose and the on-reef distance between two adjacent EDs.

The angle of repose of rock typically varies between 30° and 43° depending on the size and shape of the rock particles. For this study, the appropriate angle of repose for the typical blasted rock from underground has been determined to be 38°. Theoretically, in areas where the dip of the orebody is >38°, no mining losses should be incurred because of the angle of repose.

In a scenario where the dip of the orebody is 35°, the mining losses attributed to the angle of repose can increase by +10% when the vertical sub-level spacing is increased from 15m to 20m.

Mining losses are catered for in the modifying factors. The modifying factors applied are discussed in Section 15.1.8 of this document.

16.6.2.1.2.3 Target Area Utilisation

A typical SLOS — Longitudinal mining block, consisting of five EDs, indicating the target area utilization is illustrated in Figure 16-33.

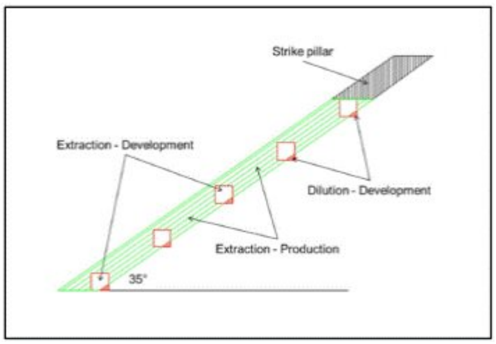


Figure 16-33: SLOS Longitudinal with 5 EDs — Target Utilisation

Figure 16-33 represents a production area with five EDs, an inclination of 35° and a mining height of approximately 10m. A SLOS Longitudinal mining block with four EDs and an inclination of 45° is illustrated in Figure 16-34. These parameters are representative of the averages for the areas where the SLOS Longitudinal mining method has been planned.

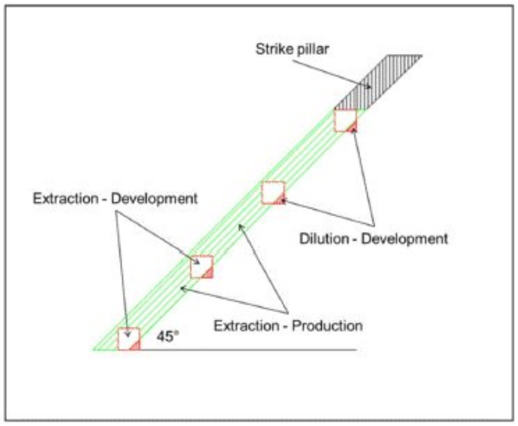


Figure 16-34: SLOS Longitudinal with 4 EDs — Trget Area Utilisation

The expected extraction and dilution percentages, for the two layouts of the SLOS Longitudinal mining method, are summarized in Table 16-30

Table 16-30: Extraction and Dilution Ratios — SLOS Longitudinal

	Quantity	
Parameter	**5 EDs**	**4 EDs**
Total area (including strike pillar)	14 120m ^{2}	11 490m ^{2}
Total target area	9 200m ^{2}	7 496m ^{2}
Pillar area	4 920m ^{2}	3 994m ^{2}
Development dilution	16m ^{2}	18m ^{2}
Extraction ratio	**65%**	**65%**
Development dilution ratio	**1.7%**	**2.9%**

These percentages have been calculated for the specific scenarios illustrated in Figure 16-33 and Figure 16-34 respectively and could vary depending on the dip of the reef and the mining height.

16.6.2.1.3 SLOS — Transverse

The main design criteria applicable to the SLOS — Transverse mining method are listed in Table 16-29.

Table 16-31: Design Criteria for SLOS Transverse Production Areas

Item	Parameter
Horizontal spacing between access spiral and target mining area	60m
Horizontal spacing between footwall drive and target mining area	20m
Spacing between consecutive EDs on sub-level	20m
On-reef span between two sill (strike) pillars	±100m
Strike distance per stope (between two rib pillars)	80m
Vertical spacing (centre to centre) between sub-levels	20m
Number of sub-levels per mining block	4
Number of EDs per sub-level	4

The on-reef span between two sill pillars is approximately 100m; the exact distance is dependent on the dip of the ore body in that specific area.

A footwall drive is planned for all areas where this mining method is to be applied as all EDs are established directly from the footwall drive. The footwall drive also serves as a loading area for of trucks and is required for the ventilation circuit. A plan view of a typical layout of a SLOS — Transverse sub-level as utilized in this mine design is illustrated in Figure 16-35.

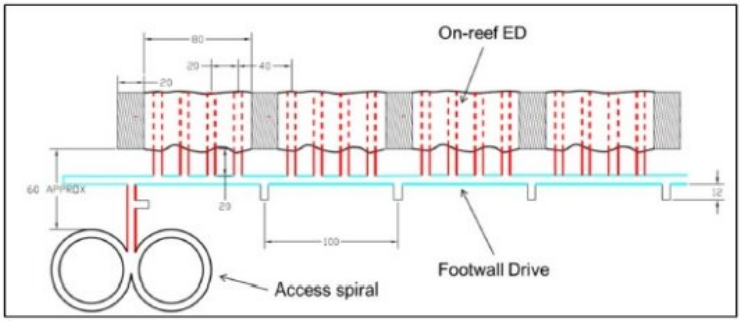


Figure 16-35: Plan View — SLOS Transverse Layout

The vertical spacing between two sub-levels is 20m, which translates four sub-levels per stope. A basic schematic illustrating the layout is shown in Figure 16-36.

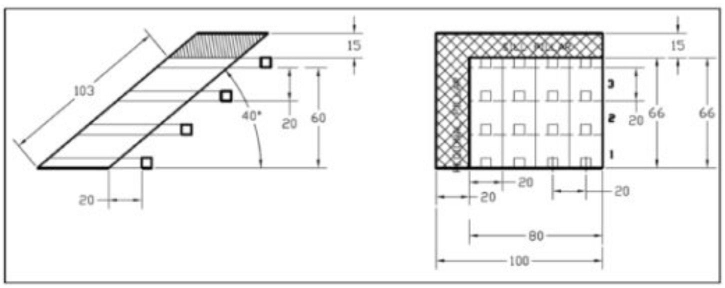


Figure 16-36: Section Views — SLOS Transverse Layout

Similar to the SLOS Longitudinal layout, the on-reef span between two consecutive sub-levels was the main determining factor in determining the vertical spacing between the sub-levels.se scenarios. The on-reef span can have a significant impact on the following two aspects:

- Maximum length of shot holes, and;
- In-stope mining losses due to the angle of repose.

The vertical spacing was mainly determined by the maximum shot hole length.

16.6.2.1.3.1 Shot Hole Length

Initially, five sub-levels were considered per mining block. Following the outcomes as shown in Table 16-32, a layout with four sub-levels was considered.

A provisional drilling pattern was also designed for the SLOS Transverse mining method. A summary, with specific emphasis on the total drilled meters per ring, is shown in Table 16-32.

Table 16-32: Summary of Shot Hole Lengths — SLOS Transverse

Vertical spacing	Number of sub-levels per stope	Dip of the reef	Total drilled metres per ring	% of Total drilled metres	
				>20m	>25m
20m	4	35	401m	4	0

The fact that for a 20m sub-level interval, 0% and only 4% of the total drilled meters were more than 20m and 25m respectively, it indicated that there is scope to increase the vertical spacing between sub-levels and should be investigated during the next study phase. For this reason, a five sub-level layout was not deemed necessary as far as drilling accuracy is concerned.

16.6.2.1.3.2 Mining Losses

The transverse orientation of the EDs in relation to the strike of the orebody results in this variant of the SLOS mining method being less sensitive to the losses attributed to the angle of repose of the blasted rock. The reason for this is that the production drilling patterns, and therefore the blasts, are not as exposed to the dip of the orebody when compared to that of the SLOS Longitudinal drilling patterns. As a result, the angle of repose is less critical than for this specific mining method.

Mining losses are catered for in the modifying factors. The modifying factors applied are discussed in Section 15.1.8 of this document.

16.6.2.1.3.3 Target Area Utilisation

A typical SLOS Transverse mining block indicating the target area utilization is illustrated in Figure 16-37.

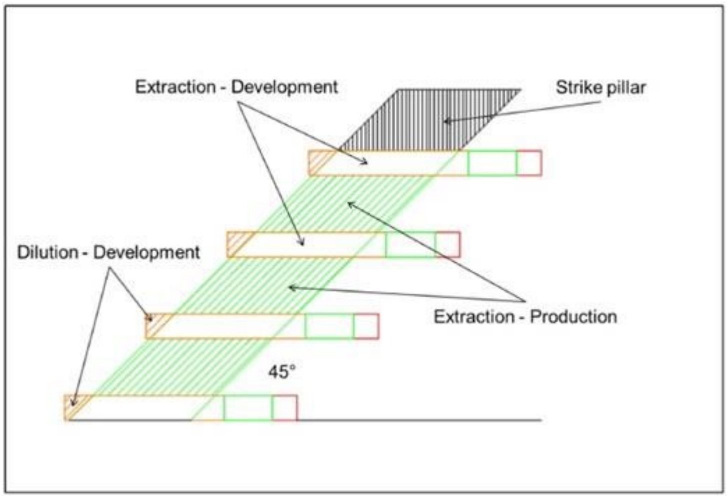


Figure 16-37: SLOS Transverse — Target Area Utilisation

The example referred to in Figure 16-37 represents a production area with an inclination of 45° and a mining height of 30m. These parameters are representative of the averages for the areas where the SLOS Transverse mining method has been planned.

The expected extraction and dilution percentages, for the SLOS Transverse mining method, are shown in Table 16-33. The dilution referred to here only allows for the portion of ED development outside of the mining target area. Dilution encountered during the production phase has not been included in these calculations, as this is catered for in the modifying factors. The modifying factors applied are discussed in Section 15 of this document.

Table 16-33: Extraction and Dilution Ratios — SLOS Transverse

Parameter	Quantity
Total area (including strike pillar)	11 490m ^{2}
Total target area	7 496m ^{2}
Pillar area	3 994m ^{2}
Development dilution	29m ^{2}
Extraction ratio	**65%**
Development dilution ratio	**1.5%**

These percentages have been calculated for the specific scenario illustrated in Figure 16-37 and could vary depending on the dip of the reef and the mining height.

It is important to note that the sub-level intervals used in this study, for both the SLOS Longitudinal and SLOS Transverse mining methods, have not been optimized comprehensively. A number of factors including extraction ratio, maximum drilling length, in-stope mining losses and the cost of development are to be considered in an optimization exercise. This optimization is a significant opportunity to increase grades and reduce waste development.

16.6.2.2 Sequencing and Scheduling Parameters

16.6.2.2.1 Initial slots

As discussed in Section 16.4 of this report, the on-reef EDs are developed to their extremity prior to the production commencing in that specific mining block. An initial slot is established at the extremity. It is from this position that production will commence on a retreat basis.

A delay of 2 weeks has been allowed for, in the mining schedule, between the completion of the on-reef development and the commencement of production in a specific mining block. During this time, drilling and blasting of the slot as well as loading of the associated blasted rock will be completed. Should secondary drilling and blasting be required, it is assumed that it will be completed during the 2-week period.

For the purpose of this study, the same sequencing and scheduling parameters for the slots have been applied for all three of the mining methods.

The scheduling parameters pertaining to the production areas for the different mining methods are discussed below.

16.6.2.2.2 Production stopes

Production in a mining block can commence when all the slots for that specific mining block have been established.

The production rates applied in the mining schedule were determined by calculating the time required for all the relevant mining activities during the production cycle. The scheduling parameters pertaining to the production rates for the different mining methods are summarized in Table 16-34.

Table 16-34: Scheduling Parameters - Production

Mining method	Mining height (m)	Scheduling parameter (tpm)
BLR	3	7 500
	6	10 000
	≥ 12	15 000
SLOS Longitudinal	3	5 000
	6	10 000
	≥ 9	15 000
SLOS Transverse	≥ 15	30 000

The production rate refers to the rate at which production ore tonnes can be generated, in other words the rate at which production rings are blasted. It is important to note that the production rates mentioned are per mining block and not per ED.

A monthly loading rate of 30 000 tonnes per LHD, which refers to the expected production ore tonnes loaded per month for each 17 tonne LHD in the fleet, has been applied. This rate is based on typical loading cycle times that are appropriate for the mining layouts applied in this study.

These LHDs will be dedicated to the loading of ore in production areas. It is also assumed that the LHDs will always have an area to load and that it would not have to wait for broken rock to be generated. For this reason, the 30 000 tonnes loaded in any month would not necessarily originate from the same mining block.

Further details regarding the operating cycles for the mining activities are discussed in Section 16.8 of this report.

16.7 Mine Designs

16.7.1 Portals

There were four different portal positions considered for this Study, namely D, G J and I.

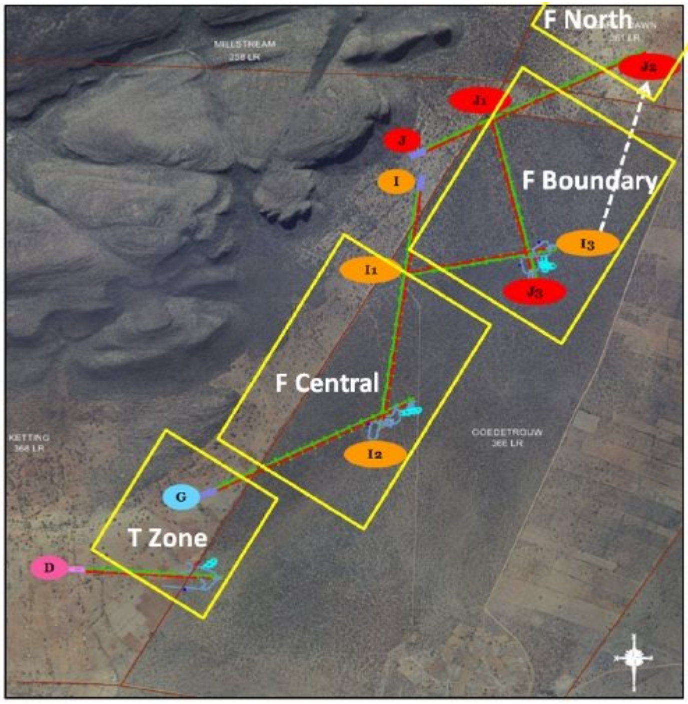


Figure 16-38: Portals and Resource Zones

Portal D and decline system services the T-Zone with a production rate of 150ktpa.

Portal G and decline system services the F-Central only, allowing for a production rate of 300ktpm.

Portal J and decline system servicing F-Boundary and F North and a production rate of 300ktpm.

Portal I servicing multiple decline systems, which access F Central and F Boundary ore bodies. It will require underground development to access the F North orebody.

The combination of portals was divided into three options:

- Option 1 — D, G and J
- Option 2 — D, I and underground development
- Option 3 — D, I and J

Based on the outcomes of a location analysis and considering the mining related parameters, it was recommended that Option 1 (portals D, G and J) be developed further.

16.7.2 Underground Design

The mine designs for this study were done in Studio 5D Planner.

16.7.2.1 Declines

Primary access to the ore body is via a 6.0mH x 6.0mW, 9° twin decline systems spaced 18m apart skin-to-skin, and connected by laterals every 100m. The decline systems will start from portals, which will be excavated to a high wall depth of 18m and sidewall distance of 30m at the same dip of 9°. Additional details regarding the portals can be found in Section 18.

The first decline will be utilized for all personnel and material transportation down the mine as well as rock haulage while the conveyor system is not established. The second decline will host the conveyor system, which will be responsible for all rock handling once the mine is in steady state. The muck bays are aligned with the connections between the declines and are 12m in length.

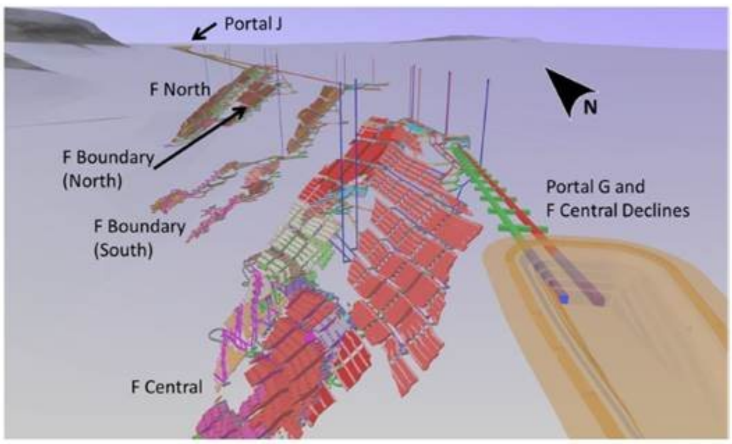


Figure 16-39: F-Centre Declines (not to scale)

The underground silo and vertical dam are positioned at the end of the main declines.

From the main declines, there is a breakaway towards the production area from which the spiral ramp will access the main levels and sublevels.

Workshops are established at the decline breakaway as close as possible to the production areas in order to reduce the travelling distance required to service the mining equipment, thus reducing idling time.

Figure 16-40 shows the transition between the main decline and the production areas for F Boundary (south block):

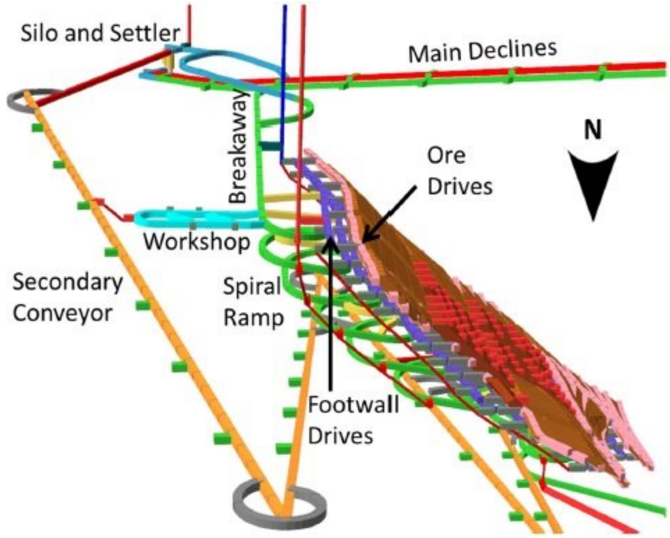


Figure 16-40: End of Main Decline Arrangement (not to scale)

From the underground silo position, a secondary conveyor decline is developed along the footwall of the ore body and will be connected to the production areas by means of tipping points. Each leg of the secondary conveyor will have one tipping access. The conveyor decline will be developed up to the tipping point, when construction of the tip will begin. A connection between the conveyor decline and the tipping access will allow the decline to continue with development while the tipping point is constructed. The tipping access will have a 15m maneuvering bay for trucks.

16.7.2.2 Main Access Levels and Spirals

The development in the BLR mining method is predominantly on-reef. The main accesses are placed at an apparent dip of 9° while the production sub-levels are horizontal. The largest continuous BLR mining area can be found in T-Zone, and can be easily identified by the pink on-reef haulages running across the ore body as per Figure 16-41.

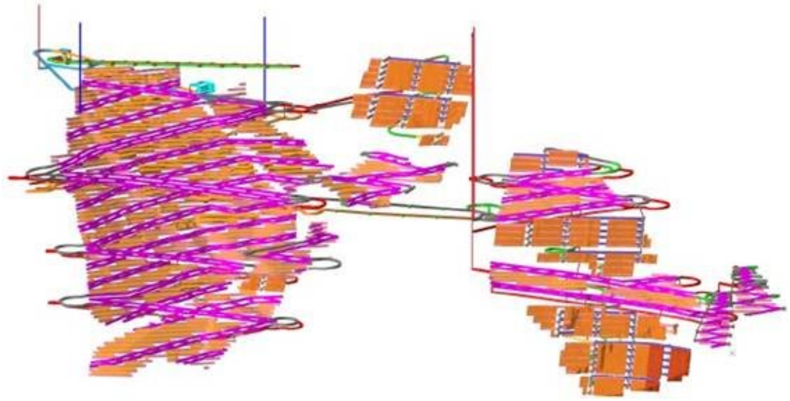


Figure 16-41: T-Zone showing a Large BLR Area to the Left (not to scale)

At the edges of the BLR mining area, there are spiral ramps that connect the main on-reef declines to the return airways (RAW) below. These are developed off-reef.

The two SLOS methods require spiral ramp development in the footwall waste to provide access between the sub-levels and the main declines. The turning radius for the spirals has been fixed at 30 meters to accommodate the mining equipment and the dip was set at 9°. Figure 16-42 shows a section of a SLOS — Transverse mining area.

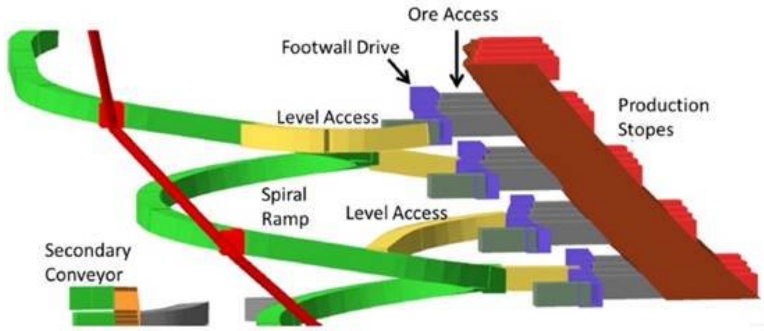


Figure 16-42: Section of Transverse SLOS (not to scale)

The conveyor tipping point can be seen in the bottom left corner of Figure 16-42. This particular production block will use this tipping point once it is installed and commissioned.

From the footwall, it is possible to observe the entire access infrastructure that services the production areas. This example was taken from the northern area of F Boundary.

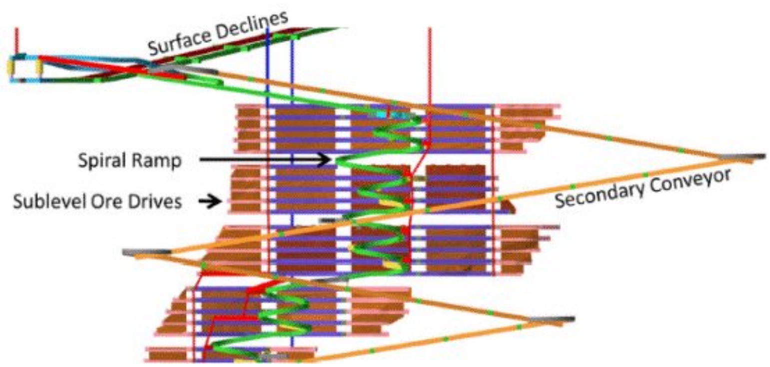


Figure 16-43: Footwall View of Infrastructure Development (not to scale)

16.7.2.3 Ventilation Development

The main access declines and access development including the spiral ramps form part of the main intake airways. The secondary conveyor declines are used as part of the RAW circuit.

Figure 16-44 shows the return air raises in the spirals for the SLOS methods (both Longitudinal and Transverse), but additional raises are required to supply air during peak production that are not in the image above. These raises are connected to the surface by raise bore holes that measure between 3 and 5.1 meters in diameter. The ventilation raises between sub-levels are connected by remuck bays that have been excluded from the following image to allow for better visualization of the system.

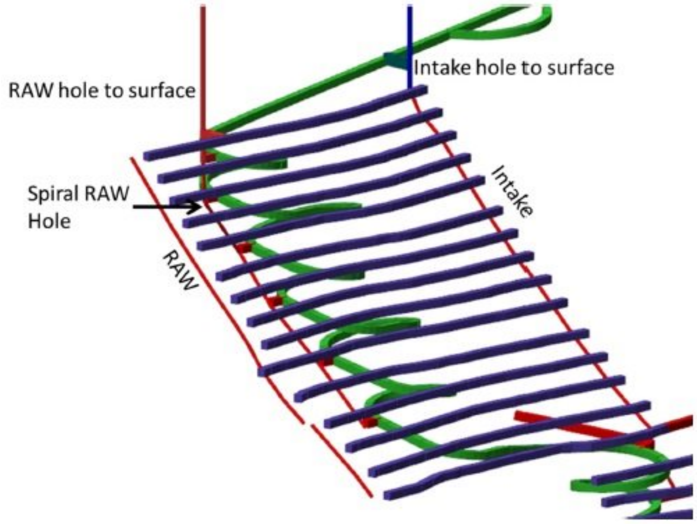


Figure 16-44: Example of Ventilation System for SLOS Method (Long and Transverse) (not to scale)

The ventilation system for the BLR method involves development underneath the mining areas. The RAW follows the main on-reef declines offset by 20 metres vertically and drop raises are used to create connections that will facilitate the movement of air between the working areas and the RAW system.

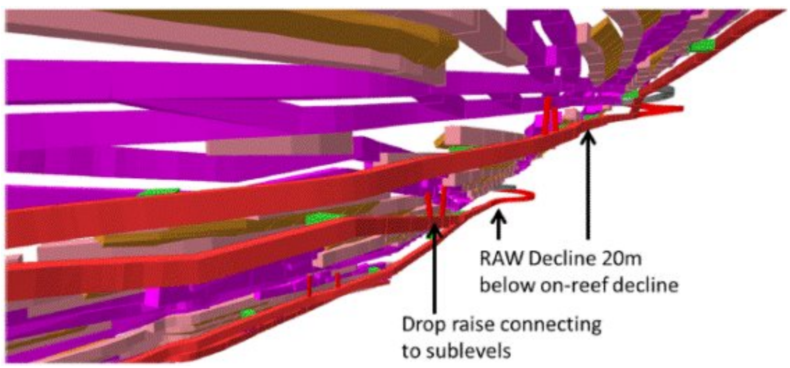


Figure 16-45: Typical Layout of BLR Mining Method (not to scale)

16.7.2.4 Production

The stoping design was generated from the MSO hanging wall and footwall surfaces. All three mining methods followed the same principle for stope design: 10 meter sections were created along the strike of the ore body and the stope outlines were designed to the extent of the MSO hanging wall and footwall. Figure 16-46 shows the stoping outlines with and without the finished design to better understand the process (taken from F Central).

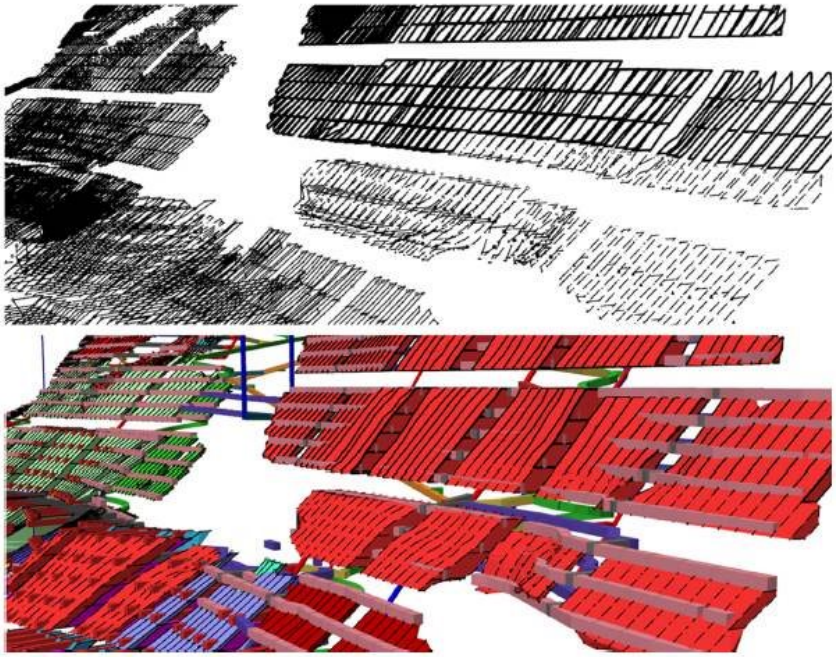


Figure 16-46: Stope Outlines (not to scale)

The geotechnical parameters were applied to all the stoping designs.

The BLR mining method requires pillars between the main on-reef declines and the also between the ore drives. The following figure represents a typical BLR layout:

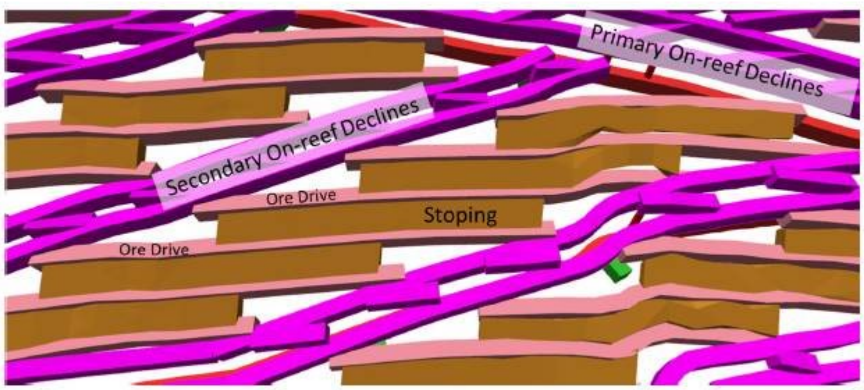


Figure 16-47: Typical Layout of BLR Mining Method (not to scale)

Both variations of SLOS will have the same parameters for panel size and spacing between sublevels. The difference will be the ore drives that will be either longitudinal or transverse to the ore body strike. The geotechnical pillars have been aligned for a continuous area to ensure greater efficiency. Figure 16-48 illustrates a typical area that has both Longitudinal SLOS (LSLOS) and Transverse SLOS (TSLOS, outlined in black).

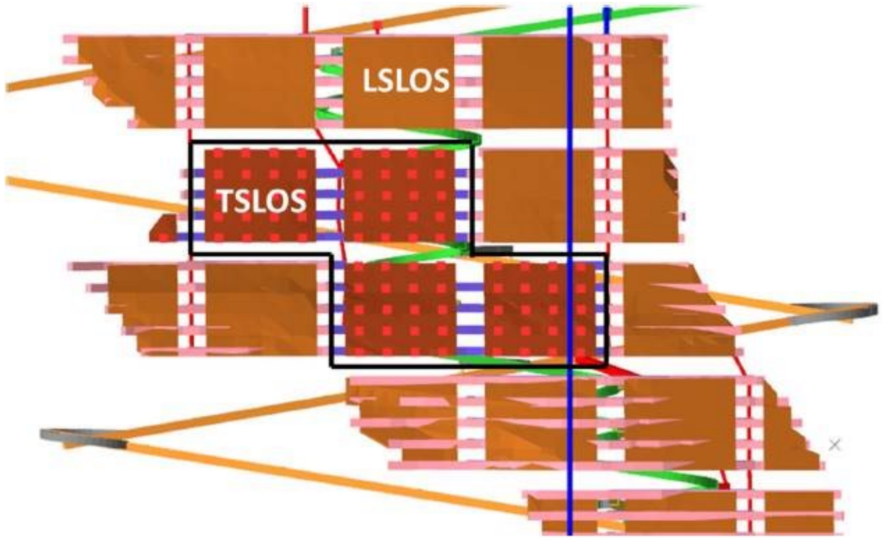


Figure 16-48: Mining Areas with Long SLOS
Longitudinal and Transverse SLOS (not to scale)

All the panels were initially designed with four stope sublevels as per the image above (five development sublevels), but an optimization study revealed that three sublevels could be used efficiently from a drilling perspective when the dip of the ore body is greater than 40°. To cater for this change, the top development sub-level of each panel with a dip greater than 40° was excluded from the schedule report; therefore the costs associated to that sublevel are not reported but the images will still show them.

The final designs for the different mining areas are shown in Figure 16-49 to Figure 16-53 looking from the west to the east.

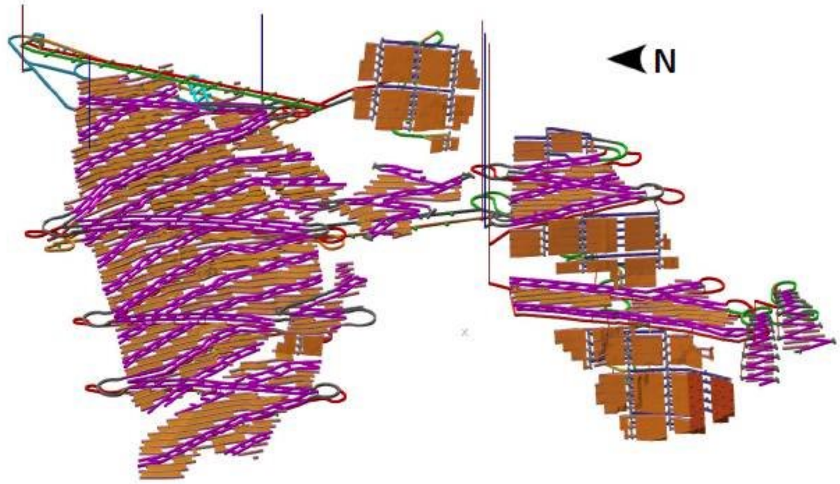


Figure 16-49: Isometric image of the T-Zone Mine Design (not to scale)

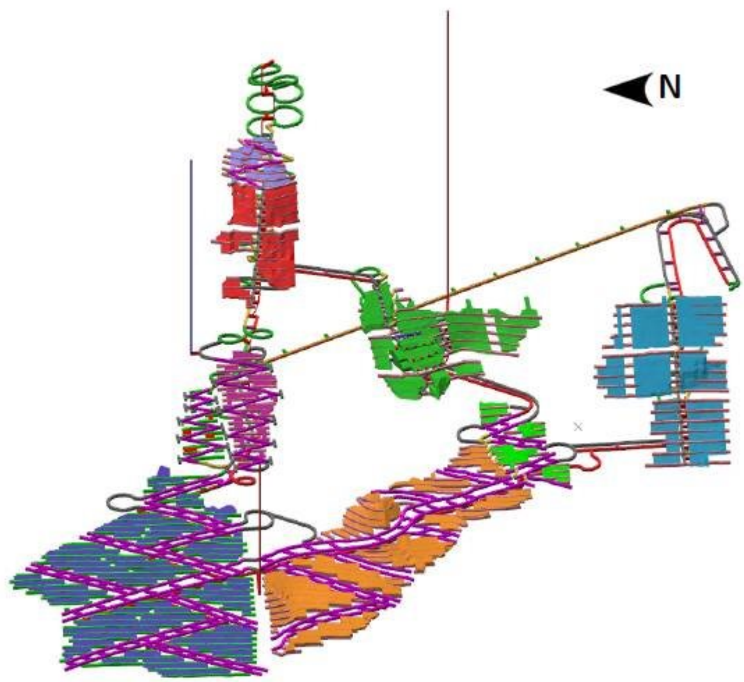


Figure 16-50: Isometric image of the F-South Mine Design (not to scale)

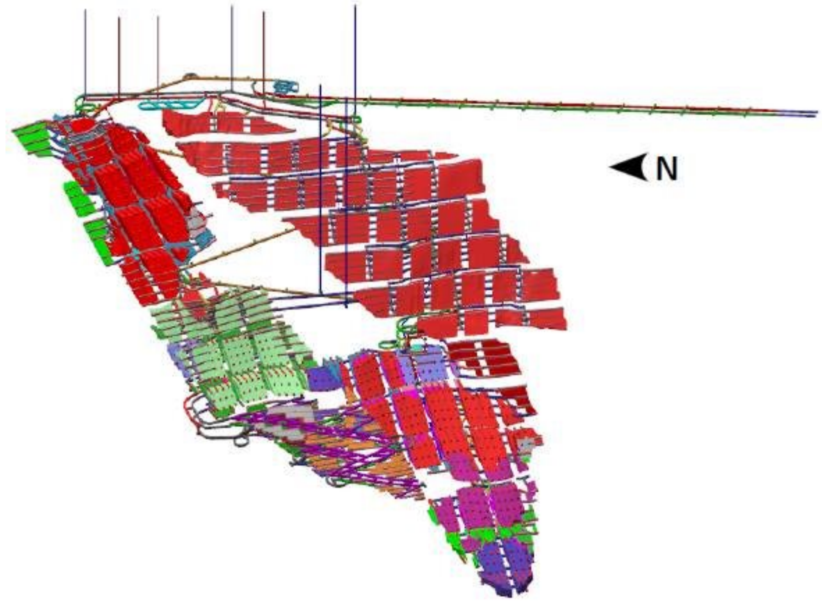


Figure 16-51: Isometric image of the F-Central Mine Design (not to scale)

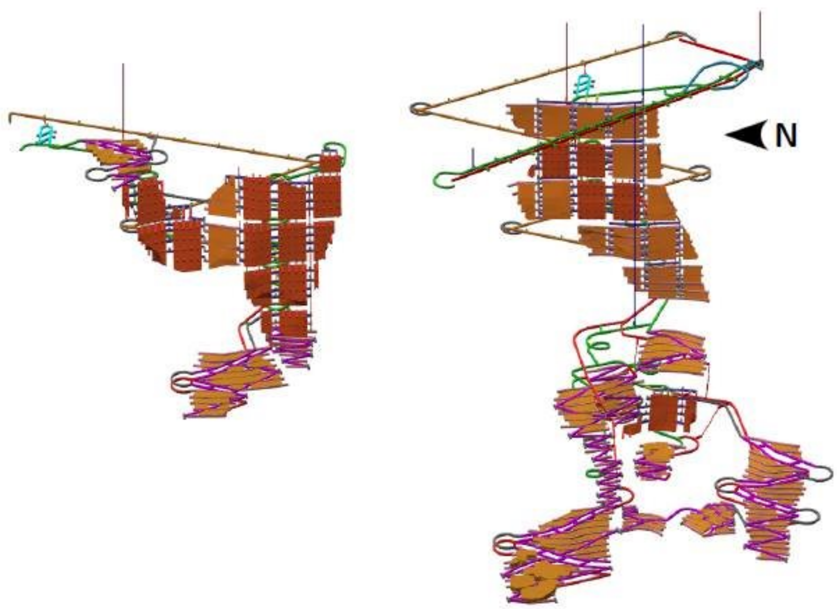


Figure 16-52: Isometric image of the F-Boundary Mine Design (not to scale)

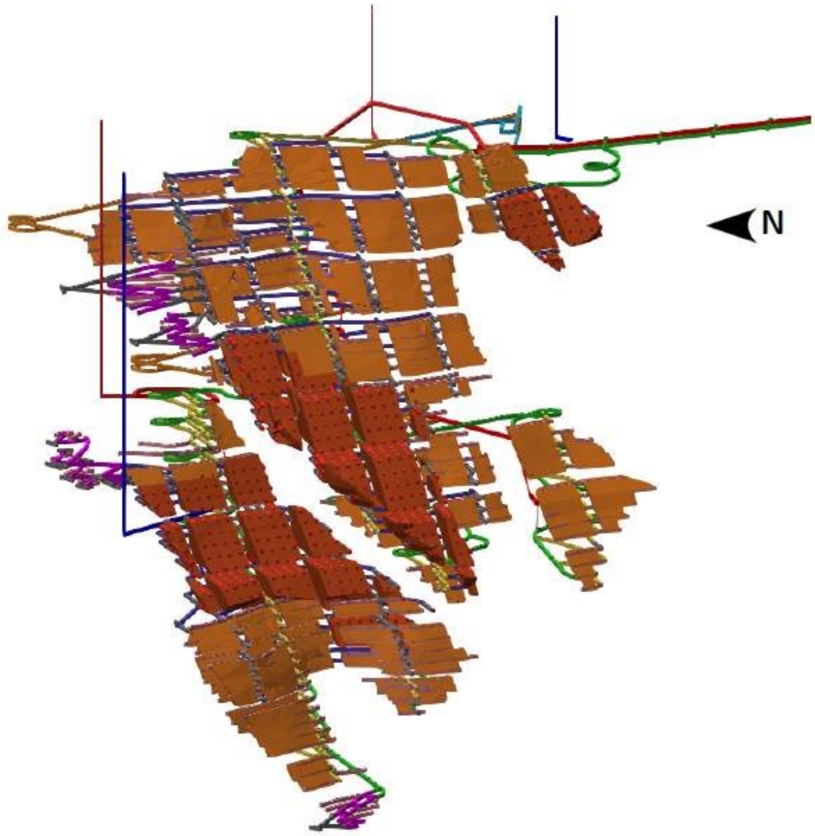


Figure 16-53: Isometric image of F-North Mine Design (not to scale)

16.8 Operating Cycles

The operating cycles were determined assuming that a Full Calendar Operations (FULCO) shift roster would be applied. The available working days and shifts per annum have been summarized in Table 16-35.

Table 16-35: Available Working Shifts

Item	Quantity
Non-working days	12
Other non-working days	0
Working days per year	353
Shifts per day	2
Average working shifts per month	58.9
Average working shifts per year	706.5

Although a FULCO working roster is applied, it was assumed that there would be 12 non-working days per year. This has become an accepted norm in the South African mining industry and is typically made up of ±8 days break over the Christmas / New Year period and a 4-day break over Easter.

It has become the norm in the South African mining industry that even operations working on FULCO do not mine for the full 365 calendar days in any year. Most mines stop operations during the Christmas break between Christmas and New Year. This equates to eight days of closure. Some mines break over the Easter weekend for four days, which again mean a closure of four days. For this study, it was assumed that twelve public holidays would apply.

As the nature of the main mining activities required to complete the mining cycle associated with development are substantially different to those associated with the production cycle, they are discussed separately.

16.8.1 Development

16.8.1.1 Effective Operating Hours

The underground mine development cycles are based on end dimensions of 6.0m (H) x 6.0m (W) and an effective advance of 3.5m per blast.

Two scenarios have been considered in the operating cycles of development. The main contributing factors to be considered for the scenarios are:

- Shift length;
- Travelling time required to get to the underground working places; and
- Ground support requirements.

The clocking point for underground shifts will typically be situated at the lamp room. Therefore, the shift commences when a person commences underground and the shift will end once a person has arrived back on surface at the lamp room.

Over the life of mine, the shift duration has been taken as 10.5 hours for underground mining employees. The shift duration has been arrived at considering the requirements of the Basic Conditions of Employment Act (BCEA) as applicable in the Republic of South Africa. Shift duration is discussed further in Section 16.11 of this report.

However, during the initial stages of the mine, while the main underground access is being developed, a 12-hour shift has been assumed. This has been done in an attempt to reduce the risk of not achieving the required development advance rates during the initial stages of the project.

To remain within the requirements of the BCEA while implementing extended shift durations, alternative shift arrangements, which includes the appointment of additional labor, will have to be implemented for the duration of this period. It is envisaged that the development of the main access declines will be conducted by mining contractors who will implement shift rosters that do adhere to relevant legislation.

As the project proceeds, the underground working places extend deeper below surface. It will therefore take longer for people to travel to their respective working places as time progresses. This additional travelling time is deemed as part of the shift and therefore the effective productive hours per shift reduces as the mine is developed further. There are also other activities undertaken in each shift, which contribute to the total non-productive time. The non-productive time for the two scenarios is listed in Table 16-36.

Table 16-36: Non-productive Shift Time for Development

Activity	Time: Main access development	Time: All other development
Travelling time to waiting place (minutes)	10	20
Start of shift safety meeting (minutes)	15	15
Pre-use inspection (minutes	15	15
Travel to working face (minutes)	0	10
Travel to workshop (minutes)	10	15
Re-fuel (minutes)	20	20
Wash and grease after the shift (minutes)	15	15
Operator unavailable and other (minutes)	20	20
Travelling time to surface (minutes)	10	20
Total shift time not productive per shift	**115 minutes**	**150 minutes**
Total shift time not productive per shift	**1.9 hours**	**2.5 hours**
Total shift time not productive per day	**3.8 hours**	**5.0 hours**
Total shift time not productive per month	**112.8 hours**	**147.2 hours**
Total shift time not productive per year	**1 354 hours**	**1 766 hours**

A mechanical availability of 90% has been applied for all mobile mining equipment except for LHDs where a mechanical availability of 85% has been applied. The effective available operating hours applicable, the underground mining activities, for a 12-hour shift as well as a 10.5-hour shift, have been summarized in Table 16-37.

Table 16-37: Available Operating Hours

Item		Time: 12 hour shift	Time: 10.5 hour shift
Working hours per shift		12	10.5
Working hours per working day		24	21
Working hours per month		706	618
Working hours per year		8 478	7 418
Available operating hours per shift	LHDs and trucks	8.3	6.4
	Other equipment	8.9	7.0
Available operating hours per day	LHDs and trucks	16.6	12.9
	Other equipment	17.8	13.9
Available operating hours per month	LHDs and trucks	488	378
	Other equipment	523	409

Item		Time 12 hour shift	Time 10.5 hour shift
Available operating hours per year	LHDs and trucks	5 852	4 539
	Other equipment	6 276	4 910

The available operating hours make provision for the effective face time as well as equipment availability.

16.8.1.2 Main Activities in Mining Cycle

The development cycle caters for two scenarios and this is based on ground support requirements. The cycle and the various activities that it comprises of are listed in Table 16-38.

Table 16-38: Main Activities in Development Cycle

Activity	Time 1.0m support pattern	Time 1.5m support pattern
Scaling / making safe (hours)	0.8	0.8
Primary support — Bolts (hours)	7.3	3.7
Secondary support — Shotcrete (hours)	0.3	0.3
Face drilling (hours)	4.3	4.3
Charging up (hours)	2.1	2.1
Clear shift and re-entry (hours)	0.3	0.3
Loading (hours)	3.1	3.1
Total time for cycle (excl. secondary support)	**17.9 hours**	**14.3 hours**
Total time for cycle (incl. secondary support)	**18.2 hours**	**14.6 hours**

16.8.2 Production

16.8.2.1 Effective Operating Hours

The main contributing factors to be considered for the effective operating hours are:

- Shift length; and
- Travelling time required to get to the underground working places.

The non-productive time related to production is listed in Table 16-36.

Table 16-39: Non-productive Shift Time for Production

Activity	Time
Travelling time to waiting place (minutes)	20
Start of shift safety meeting (minutes)	15
Pre-use inspection (minutes	15
Travel to working face (minutes)	10
Travel to workshop (minutes)	15

Activity	Time
Re-fuel (minutes)	20
Wash and grease after the shift (minutes)	15
Operator unavailable and other (minutes)	20
Travelling time to surface (minutes)	20
Total shift time not productive per shift	**150 minutes**
Total shift time not productive per shift	**2.5 hours**
Total shift time not productive per day	**5.0 hours**
Total shift time not productive per month	**147 hours**
Total shift time not productive per year	**1 766 hours**

A mechanical availability of 85% has been applied for all mobile mining equipment except for LHDs where a mechanical availability of 80% has been applied. The effective available operating hours applicable the production activities, for a 10.5-hour shift, have been summarized in Table 16-37.

Table 16-40: Available Operating Hours

Item		Time 10.5 hour shift
Working hours per shift		10.5
Working hours per working day		21
Working hours per month		618
Working hours per year		7 418
Available operating hours per shift	LHDs and trucks	5.9
	Other equipment	6.4
Available operating hours per day	LHDs and trucks	11.8
	Other equipment	12.9
Available operating hours per month	LHDs and trucks	347
	Other equipment	378
Available operating hours per year	LHDs and trucks	4 168
	Other equipment	4 539

The available operating hours make provision for the effective face time as well as equipment availability.

16.8.2.2 Main Activities in Mining Cycle

The underground mining production cycles are based on average mining heights per mining method and a ring spacing of 1.8m. The cycle and the various activities that it comprises of are listed in Table 16-38.

Table 16-41: Main Activities in Production Cycle

Activity	Time BLR	Time LSLOS	Time TSLOS
Production drilling (hours)	2.3	15.5	19.0
Charging up (hours)	1.1	4.2	4.7
Loading (hours)	4.3	37.0	45.0
Total time for cycle (excl. shift clearance and re-entry)	**7.8 hours**	**56.8 hours**	**68.6 hours**

16.9 Mining Schedules

16.9.1 Introduction

The designs detailed in Chapter 16.7 are given a mining sequence and are evaluated against the geological block models to obtain tonnage and grade. From this point, the design moves to Enhanced Production Scheduler (EPS) to be scheduled.

All areas were scheduled together so that the interaction between the different portals could be controlled holistically. T Zone, F North and F Central were prioritized. The north block of F Boundary (F4), which comes from the same portal as F North, acts as a filler to achieve the 7.2Mtpa. Once the north block of F Boundary is close to depletion, the south block of F Boundary (F3) is triggered. This arrangement was done to delay all capital related to the decline and infrastructure required for F Boundary south. F South seen in red below has the lowest priority and acts as filler for the entire project.

16.9.2 Production schedules

The final 3 g/t production schedule per mining area is illustrated in Figure 16-54.

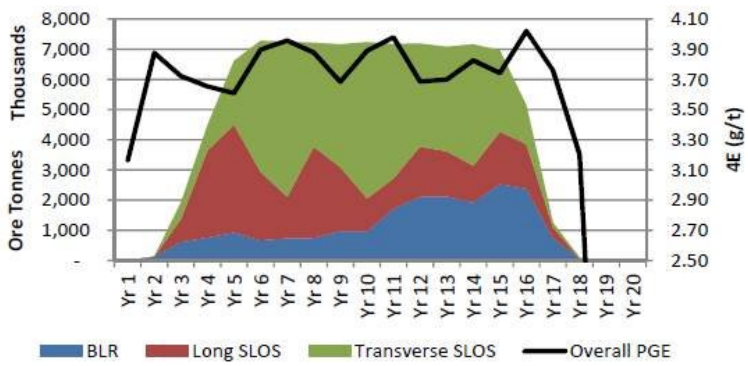


Figure 16-54: Production Schedule per Mining Area at 3g/t

The final production schedule indicating the contributions from each of the three mining methods is shown in Figure 16-59.

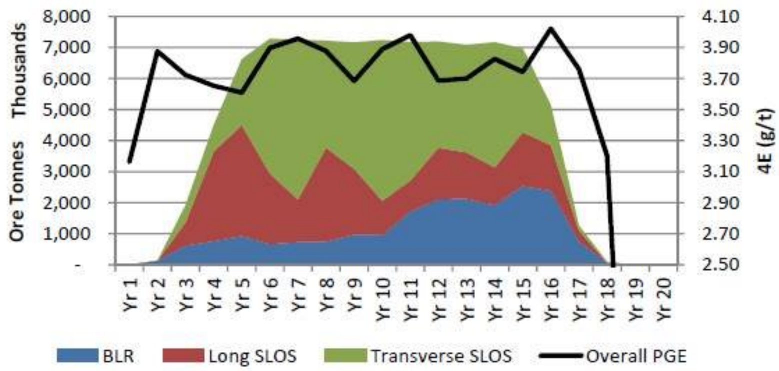


Figure 16-55: Production Schedule per Mining Method at 3g/t 4E cut-off

16.9.3 Development schedules

Development was scheduled to only be done only when it is needed. What drives the requirement for development is the ore production target. The development schedule as shown in Figure 16-56 has been split into ore and waste development meters. The proportion of ore development meters is significant in relation to the waste development meters. This is especially prevalent in the BLR mining method.

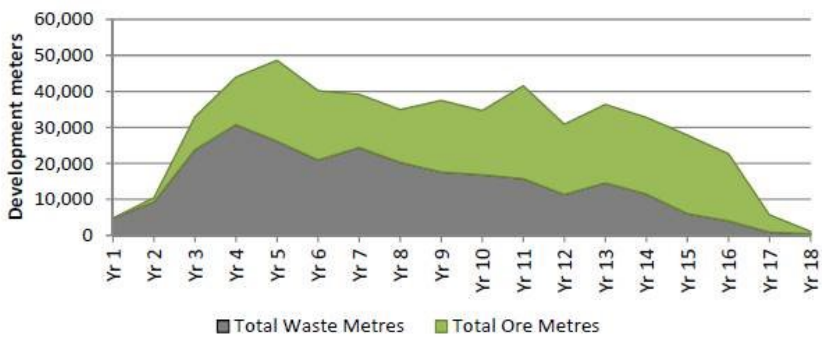


Figure 16-56: Development Schedule in Total Meters

16.9.4 Additional 2.5 g/t 4E cut-off ore

The inclusion of the additional minable material provided one extra year of steady state production at the end of the LOM. The combined schedule can be seen in Figure 16-57.

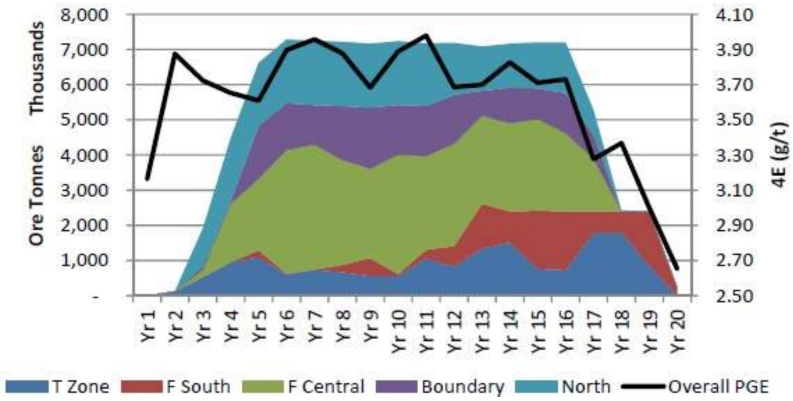


Figure 16-57: Production Schedule per Mining Area including the Additional 2.5 g/t 4E cut off ore

16.10 Ventilation and Refrigeration

The ventilation and cooling systems consider safety and health requirements in accordance with the Mine Health and Safety Act [MHSA, Act 29 of 1996] as well as complying with the PGM requirements and were undertaken by Bluhm Burton Engineering (BBE) under the guidance of Advisian.

Ventilation and cooling system designs are based on the production and development tonnage profiles and diesel fleet. The mining plan is based on steady state production of 600 000 ore tonnes per month, ventilation and cooling requirements for each mining area is phased-in accordingly over LoM.

Although diesel exhaust dilution is the main factor driving primary air requirements, air quantities were calculated from first principles and took into account heat, diesel exhaust dilution, face velocities, re-entry periods and quantities based on face area. Using the internationally accepted factor of 0.06 m^3/s/kW total rated power at the point of use, and based on the size and deployment of the diesel vehicles air quantities for each mining block were determined.

Diesel equipment will be a significant heat source accounting for almost 40% of mine heat, in comparison heat flow from rock will account for less than 10% [maximum Virgin Rock Temperature VRT 46.0°C]. The balance will come from auto-compression and other sources including electrical. In mechanized mines, to a depth of approximately 700 mbs this heat can usually be removed by ventilation used to dilute exhaust gasses. However, beyond this depth, depending on geothermal properties, heat flowing into the mine from rock and other sources combined with heat from the diesel equipment means that generally, air alone cannot adequately cool the mine and additional mechanical cooling is required. Interactive modelling of ventilation and heat flow [VUMA3D] confirmed that at depth T-Zone, F1 South, F2 Central, F4 Boundary North and F5 North additional cooling will be required to ensure that work-place wet-bulb temperatures do not exceed the work-place design limit which was 29.0°C.

16.10.1 Design Philosophy

- Intake ventilation systems combine the main decline system with strategically positioned downcast [fresh air] raise-bore-holes [RBHs]. The 'fresh air' vent RBHs will supply the bulk of the intake air and the declines and ramps will be the main intake 'headers' which will distribute air to the active sections.

- Main conveyor belt decline from surface to silos will be ventilated to return, to provide 24 hour access and in the event of a fire vent directly to return.

- Belt declines below the orebody will be used as return ventilation airways, in parallel with RBHs between the spiral ramp levels and drop vent raises between sub levels.

- Maximum wet bulb temperature in working areas is 29.0°C and this is the criteria for introducing cooling.

In general, each mining block will be ventilated as a separate district while at the same time utilizing as much common infrastructure as practically achievable. Fresh air will be introduced to mining blocks through a combination of the main North, Central and South access decline systems and strategically located fresh air raise bore holes [RBH]. Air returns through return RBHs equipped with fans. Where ore bodies overlap, returns will serve more than one block but in general, each mine will require dedicated return rises to surface. It should be noted that RBHs and primary ventilation infrastructure were phased-in to meet the production requirements as provided.

Ventilation, cooling, absorbed power and phase-in of estimated costs for each mining area is summarized in the tables below:

Mining Block	ktpm	MRBD (mbs)	Maximum Depth (mbs)	Air Quality (kg/s)	Cooling (MW)
T-Zone and F1 South [F1]	230	780	1 140	900	10.0
F Central [F2]	350	870	1 200	1200	15.0
Boundary South [F3]	145	950	1 100	600	5.0
Boundary North [F4]	140	620	700	600	
F North [F5]	200	815	1 100	700	5.0

Notes:

1) Air quantity and cooling requirements listed in each mining block are for representative points in time with steady state production at maximum depth.

2) Not all mining areas mine simultaneously as an example F3 Boundary South phases-in when F4 Boundary North depletes.

16.10.2 Absorbed Power for Ventilation and Cooling

Absorbed electrical power for ventilation and cooling will increase steadily from about 1.6 MW in 2019 to approximately 28.1 MW after 2027, at steady state production and with refrigeration. The main consumers are:

- Main surface fans : 12.7 MW
- Refrigeration system : 5.4 MW
- Secondary ventilation : 10.0 MW

Mining Block	Primary Vent (MW)	Secondary Vent (MW)	Refrigeration (MW)	Total (MW)
T-Zone [0]	0.7	0.8	0.5	2.0
F1-South [F1]	0.9	1.3	0.7	2.9
F-Central [F2]	4.1	3.5	2.8	10.4
Boundary South [F3]	2.7	2.0	0.0	4.7
Boundary North [F4]	1.0	0.0	0.0	1.0
F-North [F5]	3.3	2.4	1.4	7.1
Total	**12.7**	**10.0**	**5.4**	**28.1**

Note: These are not peak values for each mining block but are representative of mining in Year 2027, with all mining blocks in production. The peak absorbed power for each block is discussed in the relevant sections.

16.11 Labour

Labor discussed in this section includes mining labor from a Mine Overseer level and below. In this section, not all other labor compliments are discussed in this section.

16.11.1 Criteria

The two main components determining the mining labor requirements were shift arrangements and rates based on the mobile mining equipment fleet.

16.11.2 Shift arrangements

The labor was based on a FULCO, two-shift operation with shift duration of 10.5hours.

It is envisaged that a 2:1 cycle will be worked. A typical cycle would therefore consist of 8 days of work and 4 days off. This cycle requires three streams; one on day shift, one on night shift and the third stream will be on the off cycle. The 2:1 cycle proposed will adhere to the requirements of the Basic Conditions of Employment Act.

16.11.3 Rates

The mining labor reflected in this section only caters for mine employees and does not include people employed by contractors. As initial capital development will be done by contractors, mining labor employed by the mine will only be required from Year 4. Labor requirements listed in Figure 16-58 are for the total number of people at work per shift.

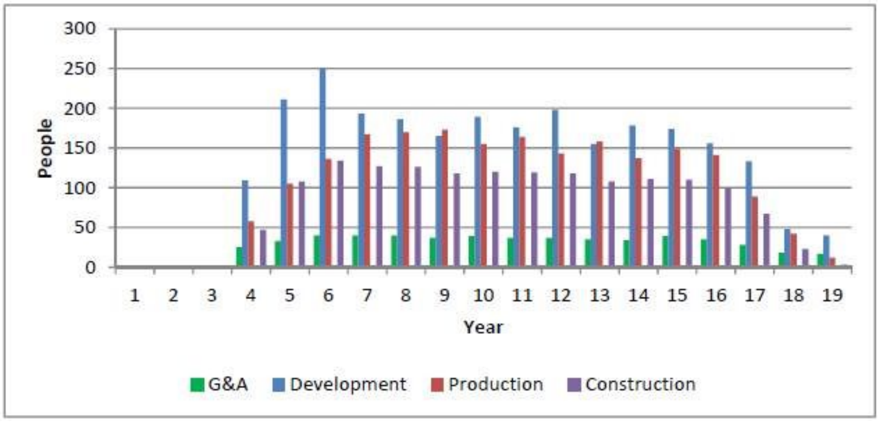


Figure 16-58: Total People at Work per Shift

Labor requirements listed in Figure 16-59 are for the total number of people for all streams combined.

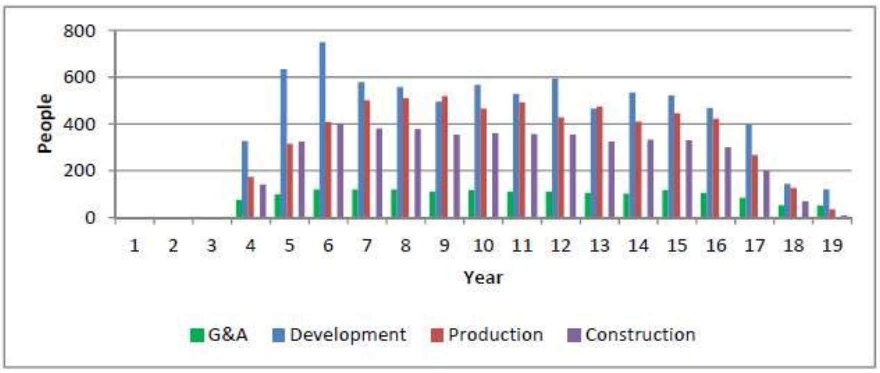


Figure 16-59: Total People at Work for All Streams

These figures do not include labor that is unavailable due to annual leave, sick leave, training and the like. An allowance has been made for this in the Operating costs.

16.12 Risks and Opportunities

16.12.1 Risks

The following mining related risks have been identified:

- Non-achievement of development advance rates resulting in schedule impacts.
- Under estimation of the unit costs for development.
- Design contingencies - The mine design of underground access infrastructure, other underground excavations and production areas should be prepared to higher level of confidence.
- Due to Step Room, Pillar, and Blind Longitudinal Retreat being mining methods of limited use, this may lead to the non-achievement of tonnages and on-reef advance rates resulting in impact on production and unit costs.

- The selected strategy of initially targeting higher-grade mining cuts has the potential of sterilizing lower grade resources.
- The design and scheduling of the additional tonnages at the tail of the production profile has been conducted at a lower level of confidence.
- Lack of local availability of appropriate skills for mining and engineering may lead to unavailability of skills to achieve development schedule and production targets resulting in cost and time impacts. This applies to the skills required for both the operation and maintenance of trackless mobile machinery.
- Fragmentation of blasted rock could have an impact on hauling cycle times, and therefore productivities, as well as the availability of the conveyor system.
- Congestion caused by the movement of underground traffic could result in longer cycle times, which in turn could have an impact on production volumes.

16.12.2 Opportunities

The following are potential opportunities, which could have an impact on the project:

- Underground access methodologies and designs. This includes the positioning and orientation of main declines.
- Potential for further optimization of development end sizes for certain excavation types.
- Optimize the layouts of the SLOS — Longitudinal and SLOS — Transverse mining methods. Production ring designs and in-stope cleaning for lower inclinations would be specific aspects to be considered.
- Potential to optimize underground rock handling with the view of reducing hauling distances.
- Further optimization of the ventilation and refrigeration system.
- Investigate the potential to increase effective face time for specific production activities such as loading and hauling.

17. Recovery Methods

The process design for the Waterberg Concentrator Plant has been developed using the metallurgical test work and assessments discussed in Section 13 of this report, as well as other desktop level studies completed by the project team. Based on the outcome of a ramp-up trade-off study conducted the selected option for the design is a two phased concentrator production.

This approach was used for flow sheet development and design. The second concentrator module is designed as a copy of the first module, with minor exceptions about shared infrastructure.

Phase 1 includes the construction of a 3.6 Mtpa concentrator module and associated infrastructure, to start production in Month 36. Phase 2 includes the construction of a second 3.6 Mtpa module to take the total production to 7.2 Mtpa in Month 53.

17.1 Process Design Criteria

The main elements from the Process Design Criteria are summarized in Table 17-1.

Table 17-1: Process Design Criteria Summary

Criteria Mining		Units	Nominal	Design
Phase 1 Ore Make-up	T-Zone	%	27%	0 - 100%
	F-South	%	0%	0 - 100%
	F-Central	%	31%	0 - 100%
	F-Boundary	%	2%	0 - 100%
	F-North	%	40%	0 - 100%
Phase 2 Ore Make-up	T-Zone	%	16%	0 - 100%
	F-South	%	11%	0 - 100%
	F-Central	%	36%	0 - 100%
	F-Boundary	%	16%	0 - 100%
	F-North	%	22%	0 - 100%
Life of Mine		years		20
Production Summary				
Annual ROM treatment rate	Phase 1	tpa		3 600 000
	Phase 2	tpa		7 200 000
No of modules	Phase 1	#		1 x 300 ktpm
	Phase 2	#		2 x 300 ktpm
Expected ROM moisture content		%m/m	5	3 - 6
Material Density	ROM blend	t/m^3		2.90
	ROM bulk density	t/m^3		1.74

Criteria Mining		Units	Nominal	Design
	Rougher concentrate	t/m^3		2.90
	Cleaner concentrate	t/m^3		3.20
ROM Size Distribution	F_{100}	mm	350	350 - 400
	F_{80}	mm	220	220 - 250
	F_{50}	mm	105	100 - 115
Target Grind	Primary mill P_{80}	µm	160	160
	Secondary mill P_{80}	µm	75	75
Crushing Circuit Operating Schedule				
Operating weeks per annum		weeks/y		52
Operating days per week		days/week		7
Operating days per annum		dpa		363
Operating hours per day		hpd		24
Crusher circuit utilisation		%		65%
Crusher annual run hours		hpa		5 663
No of modules	Phase 1	#		1 x 300 ktpm
	Phase 2	#		2 x 300 ktpm
Crushing module feed rate		dmtph		636
Milling Circuit Operating Schedule				
Operating days per annum		dpa		363
Operating hours per day		hpd		24
Mill running time		%		91.8%
Mill annual run hours		hpa		8 000
No of modules	Phase 1	#		1 x 300 ktpm
	Phase 2	#		2 x 300 ktpm
Milling module feed rate		dmtph		450
Mill Feed Head Grades				
3E + Au	T-Zone	Gpt	3.94	2.5 - 5.0
	F-South	gpt	3.78	2.5 - 5.0
	F-Central	gpt	3.59	2.5 - 5.0
	F-Boundary	gpt	3.75	2.5 - 5.0
	F-North	gpt	3.78	2.5 - 5.0
	ROM Phase 1	gpt	3.72	2.5 - 5.0

Criteria Mining		Units	Nominal	Design
	ROM Phase 2	gpt	3.73	2.5 - 5.0
Cu	T-Zone	%	0.16	0.15 – 0.25
	F-South	%	0.07	0.15 – 0.25
	F-Central	%	0.07	0.15 – 0.25
	F-Boundary	%	0.07	0.15 – 0.25
	F-North	%	0.07	0.15 – 0.25
	ROM Phase 1	%	0.09	0.15 – 0.25
	ROM Phase 2	%	0.08	0.15 – 0.25
Ni	T-Zone	%	0.08	0.08 – 0.15
	F-South	%	0.16	0.12 – 0.20
	F-Central	%	0.16	0.12 – 0.20
	F-Boundary	%	0.16	0.12 – 0.20
	F-North	%	0.16	0.12 – 0.20
	ROM Phase 1	%	0.14	0.12 – 0.20
	ROM Phase 2	%	0.15	0.12 – 0.20
Concentrate Grades				
Phase 1		g/t 3E+Au	Minimum 80	60 – 100
Phase 2		g/t 3E+Au	Minimum 80	60 – 100
Mass Pull to Final Products				
Phase 1		% of Mill feed	3.78	3.5 - 5.5
Phase 2		% of Mill feed	3.77	3.5 - 5.5

17.2 Process Description

The selected process design makes use the following key unit processes:

- Crushing and Screening
- Milling
- Flotation
- Tailings Disposal
- Concentrate Filtration and Dispatch
- Reagent Make Up and Dosing

Figure 17-1 below presents a high-level block flow diagram of the Waterberg Project concentrator plant and indicates how unit processes are added to the design to obtain the final throughput of 600ktpm.

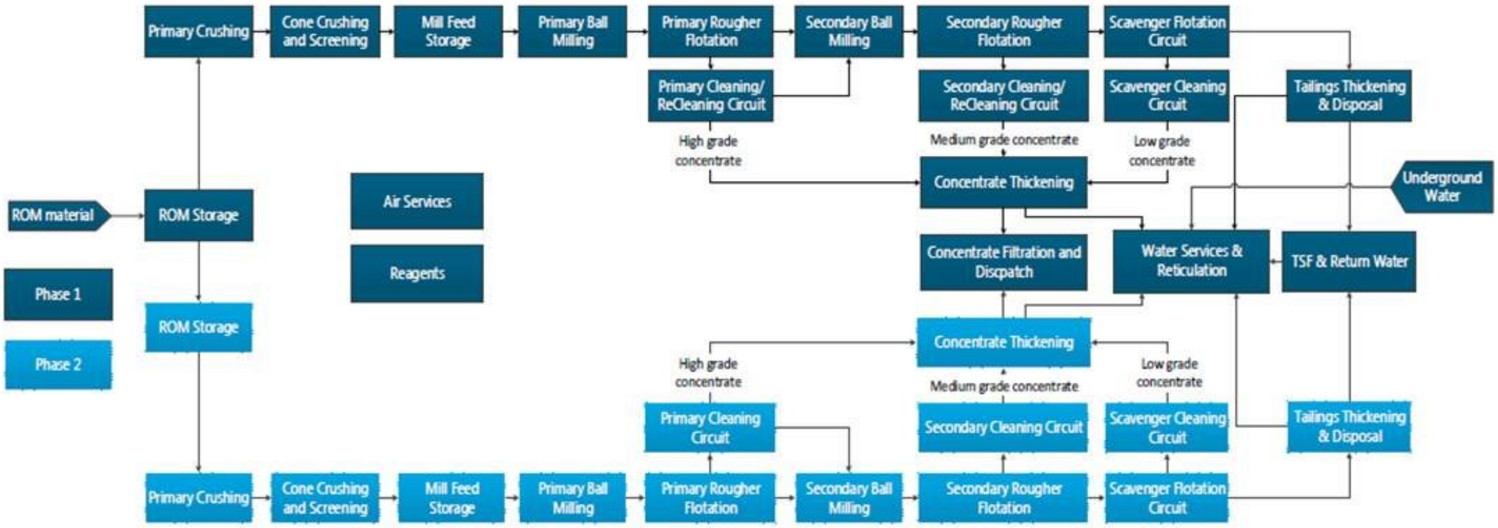


Figure 17-1: High Level Block Flow Diagram

17.2.1 ROM Ore Storage and Primary Crushing

Run-of-Mine (RoM) ore at a top size of 350mm, from underground, will be conveyed to either one of two 6500t RoM silos for storage prior to the crushing circuit. The RoM ore will be extracted from the silos, in pre-determined ratios, at a controlled rate using variable speed vibrating pan feeders and discharged onto the primary crushing feed conveyor, which will convey the blended ore to a vibrating grizzly screen. The screen oversize material will report to a single jaw crusher for size reduction to -255mm, while the undersize material will be combined with the primary crushed material onto the primary crusher product conveyor, which in turn will transfer the primary crushing circuit product to the screening circuit. Provision will be made for dust suppression at the primary crushing area.

Tramp metal will be removed prior to crushing by means of a tramp metal magnet situated at the conveyor head end.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.2 Screening and Cone Crushing Circuit

The primary crushing circuit product will be conveyed to either one of two, dual deck, coarse ore screens for classification into 3 size fractions:

- The coarse ore screen oversize product will be conveyed to either one of two secondary cone crushers for further size reduction.
- The coarse ore screen's middling product will report to the tertiary crusher feed conveyor which in turn will convey the material to either one of the two tertiary cone crushers.
- The coarse ore screen's undersize product will report directly to the mill silo feed conveyor.

The secondary cone crusher product will report to the secondary crusher product conveyor, which in turn will convey the material back to the coarse ore screening area.

The tertiary crushing product will be conveyed to either one of two, single deck, fine ore screens for classification into 2 size fraction:

- The fines ore screens o/size product will report to the tertiary crushing feed conveyor together with the middlings product from the coarse ore screens.
- The undersize product from the fine ore screens will report to the mill silo feed conveyor together with the undersize from the coarse ore screens.

This screening and crushing circuit will be designed to produce a -13mm product as feed to the mill feed silos.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.3 Mill Feed

The undersize products from the coarse and fine ore screening circuits will report to a dedicated 10 000 ton mill feed silo. The mill feed material will be extracted from the mill feed silos at a controlled rate, via dedicated duty/standby belt feeder arrangements.

Provision will be made for spillage/scats reloading as well as primary milling grinding media addition to the mill feed belt.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.4 Primary Milling and Classification

The primary milling circuit will consist of a 9 MW, 6.31m × 9.15m EGL, grate discharge ball mill, operating in closed circuit with a classification screen. A de-chipping and trash removal system will be provided.

The primary milled product will be pumped to a classification screen, after which the screen oversize product will be recycled back to the primary mill feed while the undersize product will gravitate to the primary rougher flotation circuit, via a sampling system. The Phase 2 installation will be identical to the Phase 1 installation.

17.2.5 Primary Rougher Flotation

The primary milling classification screen undersize product will gravitate to the primary rougher feed surge tank via a sampling system, from where it will be pumped as feed to the primary rougher flotation circuit.

The primary rougher flotation circuit will consist of a single bank of 6 x 50m^3 forced air tank cells in series, designed to produce a single concentrate product. The concentrate product will gravitate to the primary rougher concentrate sump from where it will be pumped to the primary cleaning circuit. The primary rougher tailings product will gravitate to the primary rougher tailings sump via a two-stage sampling system, from where it will be pumped to the secondary mill classification cyclone surge tank at the secondary milling circuit.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.6 Secondary Milling and Classification

The primary rougher tailings, as well as the primary cleaner tailings, will report to the primary rougher tailings surge tank from where it will be fed to the secondary milling circuit.

The secondary milling circuit will consist of a 9MW, 6.31m Ø × 9.45m EGL, overflow discharge, ball mill operating in reversed closed circuit configuration with a classification cyclone cluster. The secondary milling product will be pumped to the rougher tailings surge tank, where it will be combined with the primary rougher tailings and the primary cleaner tailings products. The primary rougher tailings surge tank will feed the secondary milling classification cyclone cluster. The cyclone underflow product will be recycled back to the secondary mill, while the overflow product will gravitate to the secondary rougher flotation feed surge tank via a sampling system.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.7 Secondary Rougher and Scavenger Flotation

The secondary milling classification cyclone overflow product will be pumped to the secondary rougher flotation circuit. The secondary rougher flotation circuit will consist of a single bank of 6 x 130m^3 forced air tank cells in series to produce a single concentrate product. The concentrate product will gravitate to the secondary rougher concentrate sump from where it will be pumped to the secondary cleaning circuit. The secondary rougher tailings product will gravitate to the secondary rougher tailings sump from where it will be pumped to the scavenger flotation bank.

The scavenger flotation circuit will consist of a single bank of 6 x 300m^3 forced air tank cells in series to produce a single concentrate product that will gravitate to the scavenger concentrate sump from where it will be pumped to the scavenger cleaning circuit. The scavenger tailings product will gravitate to the scavenger tailings sump via a two staged sampling system, from where it will be pumped to a final tailings thickener dedicated to each concentrator module.

Provision will be made for coagulant addition to the scavenger tailings sump, upstream of the flocculant dosage at the tailings thickener.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.8 Cleaner Flotation

The primary rougher concentrate product will be pumped to the primary cleaning circuit, where it will be combined with the primary recleaner tailings product. The primary cleaning circuit will consist of a single bank of 3 x 20m ^{3} forced air tank cells in series to produce a single concentrate, which will be pumped to the primary re-cleaning circuit.

The primary re-cleaning circuit will consist of a single bank of 2 x 10m ^{3} forced air tank cells in series to produce a final high grade concentrate, which will be pumped to the concentrate thickening circuit. The primary cleaning tailings product will be pumped to the secondary milling circuit for regrinding.

The secondary rougher concentrate product will be pumped to the secondary cleaning circuit, where it will combine with the secondary recleaner tailings product. The secondary cleaning circuit will consist of a single bank of 3 x 50m ^{3} forced air tank cells in series to produce a single concentrate, which will be pumped to the secondary re-cleaning circuit for upgrading. The secondary re-cleaning circuit will consist of a single bank of 3 x 20m ^{3} forced air tank cells in series to produce a final medium grade concentrate, which will be pumped to the concentrate thickening circuit. The secondary cleaning tailings product will gravitate to the scavenger cleaning circuit.

The scavenger flotation concentrate product will be pumped to the scavenger cleaning circuit, where it will combine with the secondary cleaner tailings product as well as the second scavenger cleaner concentrate product.

The scavenger cleaning circuit will consist of a single bank of 5 x 130m ^{3} forced air tank cells in series to produce two concentrate products. The first concentrate product will report to the secondary cleaner circuit for further upgrading, while the second scavenger concentrate product will report directly to the final concentrate circuit as a low grade concentrate.

The scavenger cleaning tailings product will gravitate to the scavenger cleaner tailings sump, from where it will be pumped to the scavenger tailings sump.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.9 Concentrate Thickening

The three concentrate products (high, medium and low grade) from each flotation module will report to the dedicated 35m diameter high rate concentrate thickeners. Each concentrate product will be sampled individually, prior to thickening. Provision will be made for trash removal via linear screen installations prior to thickening.

The thickened concentrate, at 55% solids w/w, from each thickener circuit will be pumped to either one of three concentrate filter feed surge tanks, while the concentrate thickener overflow streams will be re-used for spray water in the respective flotation modules. Any excess overflow from the concentrate thickeners will report to the process water circuit for re-use as process water.

Provision will be made for coagulant addition prior to flocculant addition for each thickener installation.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.10 Concentrate Filtration

The thickened concentrate from each concentrator module will report to either one of three concentrate filter feed surge tanks, from where it will be pumped to the final concentrate filters.

The concentrate will be dewatered to a product containing less than 12% moisture. The final product will be stored on the floor from where it will be loaded into trucks for final transportation to the smelters.

Provision will be made for final sampling of the final product prior to dispatch.

17.2.11 Tailings Handling and Disposal

The flotation circuit tailings will be pumped to dedicated, 40m diameter high rate thickeners for dewatering of the tailings slurry to a 60% (w/w) solid concentration. The thickened underflow will be pumped to dedicated final tailings tanks from where it will be pumped to the RDF via dedicated duty/standby pumping installations consisting of 4 centrifugal pumps in series (per train).

The tailings thickener overflow products will gravitate to the process water circuit.

The Phase 2 installation will be identical to the Phase 1 installation.

17.2.12 Water Services

Raw water make-up from centralized services (from bore hole and sewage water sources) will be stored in the plant raw water tank from where it will be distributed to the required points in the processing plant. The processing plant fire water system will be fed from the plant raw water tank. Raw water will be used as top-up to the process water circuit and the clean water system.

Potable water will be pumped from the centralized services to the processing plant potable water storage tanks from where it will be gravity fed to the potable water distribution system.

Plant process water will be stored in a common process water tank from where it will be distributed to each of the concentrator modules via dedicated pumping systems. The process water dam will be fed by the TSF return water, the tailings thickeners overflow products, excess concentrate thickeners overflow product, as well as plant run-off from the dedicated plant pollution control dam.

A clean water system will supply gland service water to the required areas. A duty/standby pumping system will be provided for the Phase 1 concentrator module, after which a second duty pump will be included as part of the Phase 2 expansion, utilizing a common standby pump between the two modules. The gland service water to the Phase 1 and Phase 2 final tailings pumping systems will be provided by a single pump system consisting of duty and standby multistage pumps.

A pollution control dam, equipped with a submersible pump, will be provided for plant run-off collection. Any storm water will be pumped to the process water circuit.

17.2.13 Air Services

Low pressure blower air to the flotation circuit will be supplied by a single dedicated multistage, centrifugal air blower per concentrator module. A common standby unit will be shared between the modules and will be installed as part of Phase 1.

Plant and instrument air will be supplied by rotary screw compressors. The majority of the compressed air will pass through an air filtration and drying system, before being used for instrument air. The remainder of the air will be available for use as plant air. Each concentrator module will be equipped with a dedicated duty compressor, and a common standby unit will be shared between the modules.

The drying air to each of the three final concentrate filters will be supplied by dedicated compressors and air receivers, while the pressing air to the final concentrate filters will be supplied by a common duty/standby compressor installation and a single air receiver.

17.3 Sampling and Ancillaries

17.3.1 Process Plant Sampling and Laboratory

Provision is made in the concentrator plant design for the inclusion of a sample preparation laboratory, to prepare daily samples prior to dispatch to the centralized, complex assay laboratory, where the required analysis will be conducted on each of the samples. The centralized complex assay laboratory will cater for mining grade-control, processing plant and environmental samples.

Provision has been made in the design for the necessary sampling points and equipment, as per Table 17-2.

Table 17-2: Processing Plant Sampling Summary

Sample Description	Sample Type	Analyses Required	Sampling Equipment Provided
Mill Feed Sample	Process control	Particle size distribution 4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Manual belt cut
Primary Rougher Feed	Metal accounting	Particle size distribution 6E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Primary Rougher Feed	Process control	Particle size distribution 4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Primary Rougher Tails	Process control	Particle size distribution 4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Secondary Rougher Feed	Process control	4E Fire-assay	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Scavenger Tailings	Process control	Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler

Sample Description	Sample Type	Analyses Required	Sampling Equipment Provided
Scavenger Cleaner Tailings	Process control	4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Final Tailings	Metal accounting	Particle size distribution 6E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Primary Re-Cleaner Concentrate	Process control	4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Timed vezin type sampler
Secondary Re-Cleaner Concentrate	Process control	4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Timed vezin type sampler
Scavenger Cleaner Concentrate	Process control	4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Timed vezin type sampler
Thickened Concentrate	Process control	4E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Primary rotary vezin type sampler in conjunction with a secondary rotary vezin type sampler
Reagent make-up checks	Process control	Various	Manual sampling required
Final Concentrate Product	Metal accounting	6E Fire-assay Cu, Ni, Fe, Mg, Si via ICP Sulphur via Leco	Auger type sampler

The primary rougher flotation feed, final tailings and final concentrate product assays will be used to compile the plant metallurgical balance.

The labor plan used to estimate the process plant operating costs includes operational staff on each shift to cater for sample collection and preparation.

A monthly allowance has been included in the operating cost to cater for sample preparation consumables.

17.3.2 Process Control

Provision has been made in the design for a fully integrated control system to allow for control of the concentrator modules from a centralized control room.

Each concentrator module will be equipped fitted with a high level of automation to allow for remote control of major processing equipment by a PLC and SCADA system. An integrated SCADA/HMI control system will be used for interfacing with the operational staff.

The labor plan used to estimate the process plant operating costs includes operational staff on each shift to operate the control room as well as dedicated control and instrumentation technicians.

No on-line analyzer has been included in the process plant design, but the equipment can be retrofitted in future if deemed necessary.

17.3.3 Weighbridge

A weighbridge, dedicated to the concentrator plant, is included in the design. This weighbridge will be used to control delivery and dispatch of the concentrate product as well as reagent and grinding media deliveries.

The concentrate shipment with 30 tonne trucks will require between 20 and 30 shipment transfers per day.

17.4 Utility Consumption

17.4.1 Power

Refer to Table 17-3 for a summary of the envisaged power consumption of the concentrator plant.

Table 17-3: Processing Plant Power Consumption

Item	Installed Power MW	Absorbed Power MW	Absorbed Power MVA
Module 1	45.56	30.60	35.95
Module 2	45.65	31.95	35.15
Total	**91.21**	**62.55**	**71.10**

The power consumption is calculated as 69.5 kW/t ore milled.

17.4.2 Water

The processing plant raw water requirement is based on the concentrator circuit mass balance, and takes into account the predicted water return from the RDF.

The raw water make-up requirement has been calculated as 0.32 t /t ore milled.

17.4.3 Materials

Refer to Table 17-4 for a summary of the envisaged material types to be used by the process plant and associated consumptions.

Table 17-4: Process Plant Materials Summary

Material	Material Use	Physical Form of Delivery	Dosing Strength	Consumption
SIBX	Collector	Liquid, via bulk tankers	10.00% w/v	100 t/module/month
Sendep 30E	Depressant	Solid, via bulk tankers	1.00% w/v	135 t/module/month
Senfroth 522	Frother	Liquid, via bulk tankers	25.00% w/v	55 t/module/month

Material	Material Use	Physical Form of Delivery	Dosing Strength	Consumption
Magnafloc 1957	Coagulant	Solid, via bulk tankers	2.00% w/v	60 t/module/month
Magnafloc 919	Flocculant	Solid, in bulk bags	0.50% w/v	8 t/module/month
76mm High chrome steel balls	Primary Mill Grinding media	—	N/A	90 t/module/month
32mm High chrome steel balls	Secondary Mill Grinding media	—	N/A	230 t/module/month

18. Project Infrastructure

18.1 Introduction

This section describes the infrastructure work that will be required for the Waterberg Project.

The project infrastructure includes regional and local infrastructure. A breakdown of the key infrastructure is provided below:

- Regional Infrastructure
  - Bulk water supply infrastructure to get bulk water onto site.
  - Infrastructure to get ground water from surrounding area to the site.
  - Electrical supply infrastructure to get supply onto site and into consumer sub-station.
  - Access roads to and from the mine.
  - Telecommunication and internet services (probably satellite).
- Local Infrastructure

  Local infrastructure relates to all infrastructure on the mine site (Ketting, Goedetrouw and Early Dawn farms) and is sub-divided into the following areas:
- Underground Infrastructure:

  The underground infrastructure will consist of following:
  - Ore and waste handling systems
  - Water supply and dewatering systems
  - Workshops and refueling facilities
  - Personnel and Material transportation
- Portal Infrastructure including portal MV substation, offices, change houses, waste handling/sorting facility, parking and transport area for three portal areas namely:
  - South Complex (Figure 18-1)

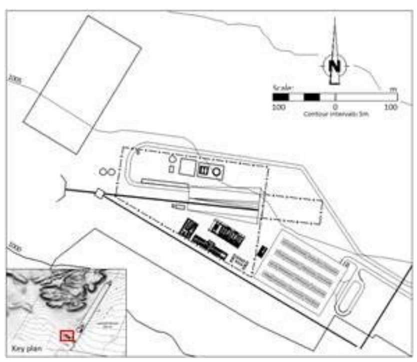


Figure 18-1: Surface Layout: South Complex (Source: Advisian)

- Central Complex (Figure 18-2)

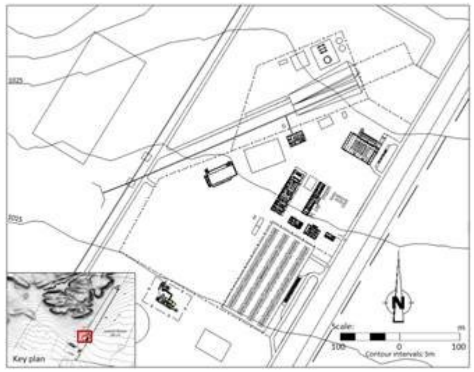


Figure 18-2: Surface Layout: Central Complex (Source: Advisian)

- Northern Complex (Figure 18-3)

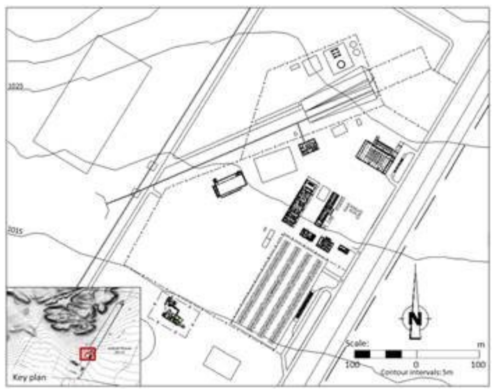


Figure 18-3: Surface Layout: North Complex (Source: Advisian)

- Ore processing
  - Processing plant.
  - Residue facility.
- Shared services
  - Return water infrastructure.
  - Storm water and pollution control infrastructure.
  - Waste rock dumps.
  - Main consumer sub-station.
  - Standby generator station.
  - Administrative offices.

- Induction centre.
- Control room and medical centre.
- Proto room.
- Fire water pumping system for south and central areas.
- Compressor station.
- Potable water treatment plant and storage tank.
- Sewage treatment.
- Bulk water distribution and buffer dam.
- Central workshop.
- Assay laboratory (grade control).
- Training centre (original construction camp).
- Weighbridge.
- Stores.
- Sewage transfer facility.
- Guardhouse and access control to area.
- Tyre/wheel exchange workshop.
- Potable water tank and infrastructure.

- Mine ventilation and refrigeration surface infrastructure

18.2 Site Layout and Access Roads

The Waterberg Project location is shown in Figure 18-5. The project site is located some 85km north of the town of Mokopane (formerly Potgietersrus) in Seshego and Mokerong, districts of the Limpopo Province. Although the bulk of the roads surrounding the site are provincial roads under the jurisdiction of the Roads Agency Limpopo (RAL), some of the minor roads are the responsibility of either the Capricorn District Municipality or the three relevant Local Municipalities.

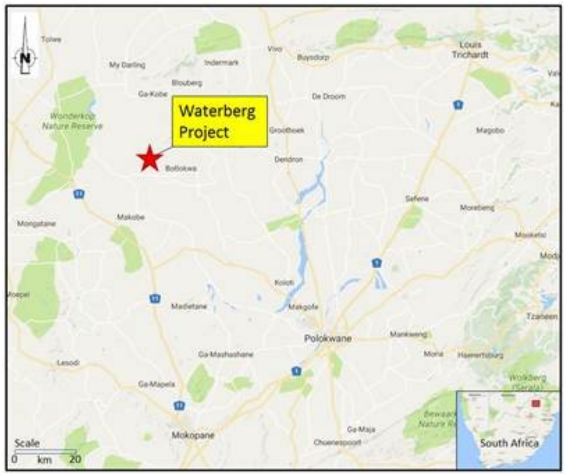


Figure 18-4: Location of Waterberg Project (Source: Google Maps)

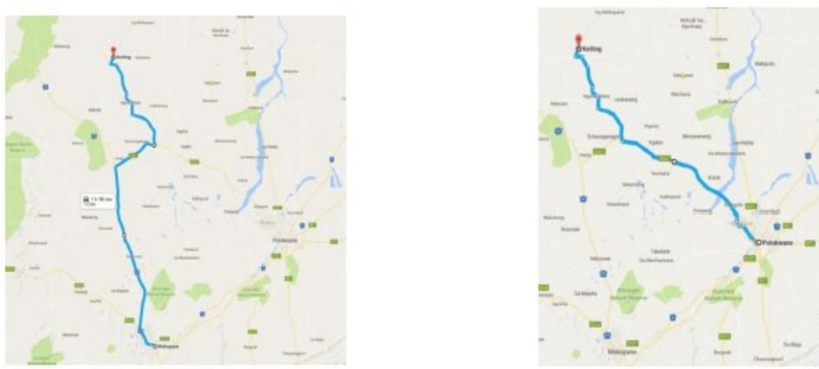


Figure 18-5: Access Routes to Waterberg from Makapane (112 km) and Polokwane (94 km)

The Waterberg Project is situated some 56km from the N11 national road that links Mokopane with the Groblers Bridge border post to Botswana. Access to the project area (Figure 18-5) from Mokopane (112km) and Polokwane (94km) includes about 32km of unpaved roads. It has been assumed in this study that this portion of the access route will be surfaced.

The balance of the route and the impact of the increased movement of vehicles over river crossings and through settlement areas will have to be assessed, during subsequent project development phases, to determine additional costs that may be incurred to upgrade and repair.

Transport of the concentrate has been assumed to be done by contract haul and a rate per tonne component has been included in the financial model.

18.3 Water General Infrastructure

South Africa is a country of relatively low rainfall and, in particular, the Limpopo province where the PTM Waterberg project is located. The project is located in the Mogalakwena River Catchment area, which is semi-arid, and runoff is limited. Extensive water resources development has however been undertaken in the catchment for urban (Mokopane and Mahwelereng) and irrigation development in the form of water storage in the Doorndraai Dam, and along the lower reaches of the Mogalakwena River through the construction of the Glen Alpine Dam mainly for irrigation. Sewage effluent from Mokopane and Mahwelereng is pumped to the Mogalakwena mine just north west of the town. The area will require additional water capacity to meet the growing demand from the mining, agricultural, and domestic sectors. The Government has committed to address the infrastructure and funding challenges in the interest of development in the region and has launched the Olifants River Water Resources Development Project (ORWRDP), which intends to bring water from the Olifants River system to the Mogalakwena River system.

Groundwater abstraction schemes in the area have also been developed mainly for domestic consumption at the rural villages. Water quality problems however exist due to high salts and nitrates in some areas. Borehole yields vary over the area with yields of up to 10l/s, along major structures in the Waterberg sediments and in the highly weathered and fractured Gneisses. However, due to the low rainfall, recharge to the aquifers is low with the average annual recharge estimated to be only about 12mm per annum. The potential for further groundwater development however exists.

Based on a hydrogeological desk study, the groundwater inflow into the proposed mine workings has been estimated at 3.3Ml/day. These inflows will result in an impact zone around the mining lease area of about 6kms.

During the Pre-Feasibility Study the water demand was estimated for the Waterberg operation to be 10.6 Ml/d. Options for this bulk water supply considered were:

- Glen Alpine Dam,
- Olifants River Water Resource Development Project (ORWRDP)
- Transfer of water from Lephalala River,
- Local and regional groundwater,
- Effluent from various Waste Water Treatment Works (WWTW) including:
  - Louis Trichardt / Makhado,
  - Bochum,
  - Dendron and
  - Seshego.

In the assessment of the above options it became clear that there are risks related to the capacity and timing of the various options and that a combination of options should most probably be considered.

The Glen Alpine dam option involves raising of the dam wall and installation of pumping and piping infrastructure to deliver bulk water supply to Waterberg site. As a result of an over allocation of water in the Mogolakwena catchment area and the seasonal varying dam level of the dam, this option would have to be combined with the acquisition of ground and surface water rights. Although this would mitigate the risk, it would still not guarantee a sustainable source of bulk water to the site.

The Olifants River Water Resource Development Project (ORWRDP) has been designed to deliver water for domestic use and mining in the Eastern Limb and Northern Limb of the Bushveld Igneous Complex (BIC) of South Africa. The ORWRDP consists of the new De Hoop Dam, the raising of the wall of the Flag Boshielo Dam, and related pipeline infrastructure, which will ultimately deliver water up to Sekuruwe. Sekuruwe is situated some 30kms to the north of Mokopane and 60kms south of PTM Waterberg Project. PTM would then be responsible for the development of the pipe and pumping infrastructure from Sekuruwe to site. This option poses huge risks to the project as the water availability has been questioned, Government funding is problematic and the forecast delivery of the infrastructure is not aligned with the project needs and could be more than 5 years late.

The option of effluent water from various Waste Water Treatment Works (WWTW) was identified as a more sustainable source and should be available as long as there is human consumption. Although a sewage effluent line from Makhado to the Waterberg site is in excess of 130km, other opportunities of water collection by this pipeline arises. Those opportunities are the collection of water from smaller sewage work ponds at Mogwadi and Senwabarwama. The pipe line also allows access to groundwater from various wellfield areas along the line route to supplement supply. The construction of this line could also be executed in stages to suit the process water demand.

18.3.1 Water Balance

Due to the scarcity of surface water in the Waterberg site area a mine specific water balance was prepared to indicate the water usage for the overall mine, including the process plant.

It is also understood that there will be minimal rainfall runoff on average in the mine area. For short periods during the rainy season isolated occurrences of high runoff may occur. For the purpose of this study, rainfall runoff is considered minimal.

Water demand and all processes were reviewed to optimise water consumption figure and the expected consumption for Waterberg was calculated at 10.6 Ml per day, based on the following:

- Conventional lined RDF (50% RDF Recovery).
- Raw water make-up for the process plant at 0.44 ton/ton of ore milled.
- Underground mining water at 1.0 ton/ton of rock mined.
- An industry standard allowance for fissure water at 0.15 ton/ton of rock has been used to determine fissure water inflow into underground working. This results in excess return water from underground to be made available to the Process plant as raw water.
- 5% loss due to moisture in ore from underground.
- 5% loss due to moisture in exhaust ventilation air.
- Other means of dust suppression will be investigated rather than using raw water. This would include environmentally friendly products with penetrating and aggregating properties which preventing fine particles from being blown away as dust.
- Treated return water from the sewage treatment plant will be made available to the process plant as raw water via the return water dam.
- The moisture content in ore is accounted in the process plant’s water requirements.
- Raw water supply for the potable water treatment plant will initially be sourced from boreholes on site.

18.3.2 Bulk Water Source

Consider the bulk water source options as described is Section 18.3. The option of wellfields in combination with an effluent water pipeline from Bochum (Senwabarwama Ponds) is the most favorable with the least risk and is considered to be the base case. This infrastructure would allow the collection of water from various sources along the way, thereby ensuring a more sustainable bulk water supply to the Waterberg site.

The wellfields in combination with Waste Water Treatment Works (WWTW) pipeline from Bochum also creates the following opportunities:

- access to groundwater from various wellfield areas along the route to supplement supply. This water is considered unsuitable for human consumption and would therfore have little impact on community water requirements;
- collection of water from smaller WWTW at Mogwadi;
- possible future expansion of the pipeline to collect effluent from Makhado WWTW.

18.3.3 Storm and Contaminated Water

Storm water is defined as the clean water that enters the mine area during a rainfall event, either by direct rain on non-polluted areas in the mining area or as collected storm water from outside the mining area. Storm water is the water that has to be managed around the property and cannot be used by the mine but has to be discharged downstream from the mining area.

Contaminated water is all water that is contaminated from mining operations and has to remain within the closed loop water balance internal to the mining area. Typical sources are rainwater falling on contaminated or dirty areas or spillage water.

The storm water management measures will include cut-off berms to divert storm runoff upstream of the mining area.

The contaminated water management measures include the following features:

- Run-off drains local to the process plant and portal areas to collect all polluted water.
- Site wide run-off drains to collect polluted water from other areas in the mining area and deposit it to the pollution control dam.
- Dedicated contaminated water drainage systems around the stockpile areas.
- A pollution control dam to capture this water and return to the mining and process water circuits.
- Silt traps to collect water from run-off drains and remove grit before discharge into the pollution control dam.

In accordance with the overall water balance, water will be pumped out from the pollution control dam back into the water circuit.

All contaminated and storm water systems will be estimated in accordance with the requirements of the EMP and Integrated Water Use Licence.

18.4 Electrical General Infrastructure

18.4.1 Predicted Electrical Load

The predicted electrical load based on connected load and mining schedule resulted in the below shown load profile (see Figure 18-6). The implementation dates of the plant Phase 1 and phase 2 combined with the increase in mining activities drives the requirement for the Eskom 132 kV OHL from Burotho Transmission station as indicated in Figure 18-6.

The steady state load was calculated at 144MW in July 2031 and will remain at these levels until the production levels decrease towards the end of the mine life. Power factor was calculated at 0.95 with the inclusion of PFC at the main Consumer substation.

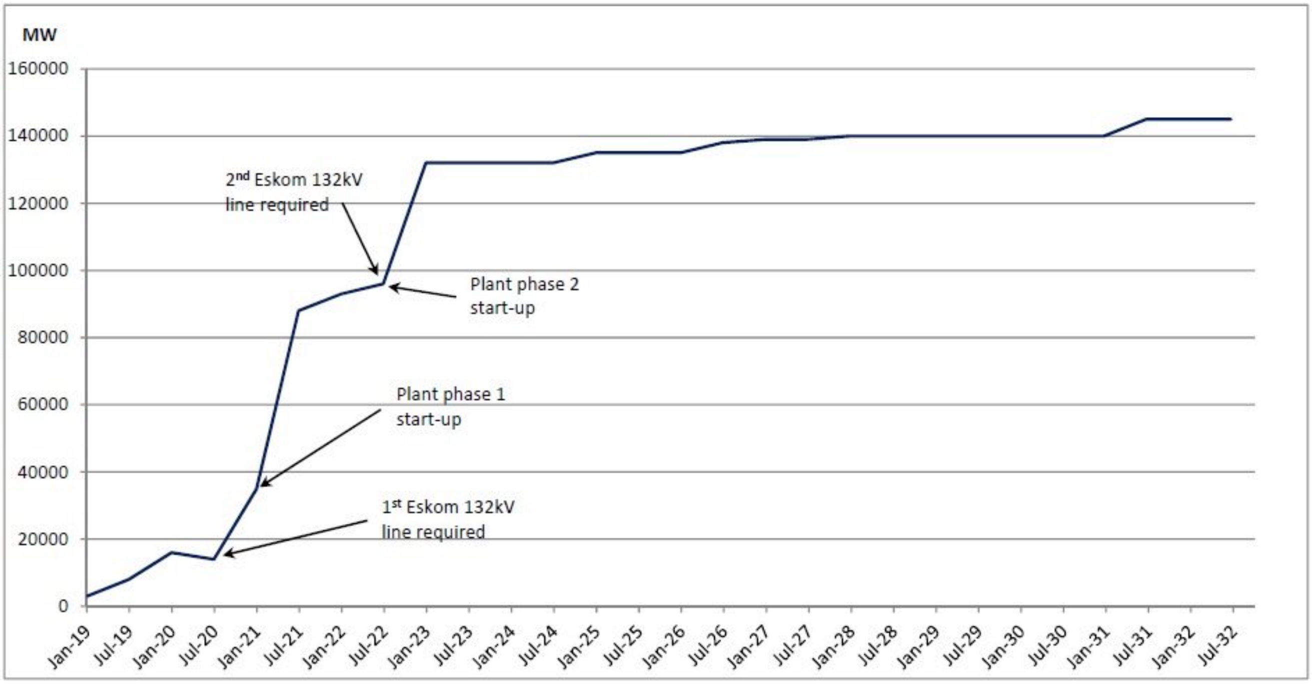


Figure 18-6: Predicted Electrical Load Profile

18.4.2 Temporary Power Supply

The existing 22 kV rural reticulation network in close proximity to the planned mining location on the farm Ketting, has a limited capacity. The plan for the provision of a temporary power supply of 10MVA therefore comprises of the upstream strengthening of this network. This 10 MVA supply will be used for initial construction power and first stages of the sinking. After the commissioning of the Ketting 132 kV substation and main Consumer substation, the 22 kV, 10 MVA can be used for the housing scheme with an estimated load requirement of 6 MVA.

18.4.3 Bulk Power Supply

The bulk electricity supply for the project is being planned to cater for mining and plant production rates of up to 600ktpm, which correspond to an electrical load of up to 160MVA.

Existing 66kV and 132kV networks approach to within 25km from the project site, however, it has been determined that the capacities of these networks are inadequate to supply the project load.

The updated electricity supply plan compiled by Eskom therefore provides for the establishment of new 132kV overhead lines from the Eskom Burotho 400/132kV main transmission substation, which is located approximately 77km south of the project site. Eskom has confirmed in principle the availability of capacity from this system to supply the mine.

The proposed bulk electricity supply infrastructure comprises the following:

- Two 77km long 132kV overhead lines from Burotho transmission substation;
- Two 132kV line feeder bays for these new lines at Burotho transmission substation; and
- A 132kV switching substation and step-down substation located on the project site.

The development of the abovementioned infrastructure is being done in conjunction with Eskom on a Self-Build basis in terms of which Waterberg JV Resources is responsible for most of the development work.

This work is already in an advanced stage, with line route planning and environmental impact assessment work having progressed well (refer Figure 18-7 which shows some of the 132kV overhead line route options).

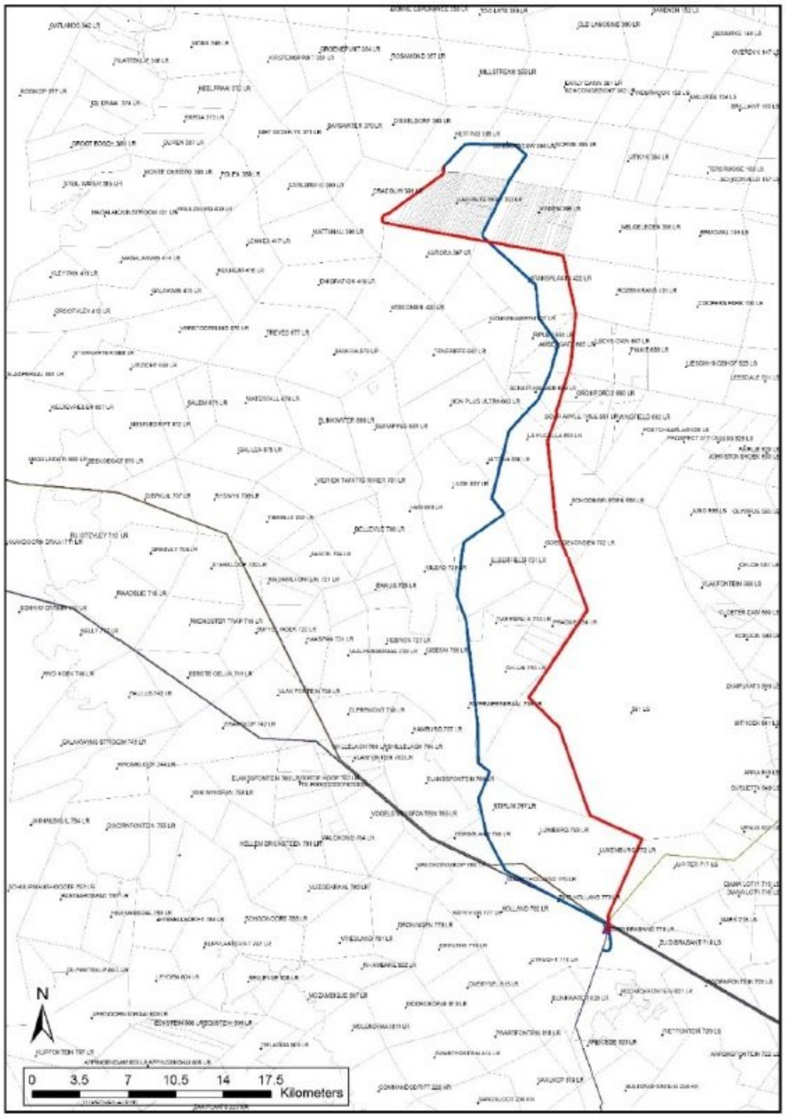


Figure 18-7: 132kV Overhead Line Route Options

18.4.4 Permanent Site Electrical Distribution

From the Main Consumer Substation the T-Zone surface infrastructure, Surface Ventilation and Refrigeration, Reduction Plant, T-Zone underground substation and the F-South substation will be fed directly from the Main consumer substation. From the T-Zone and F-South underground substations the water handling, material handling and mining activities will be fed to a network of 11 kV miniature substations and low voltage reticulation.

The Central portal substation will be fed from the Main consumer substation and the F-Central underground operations from the Central Portal substation. From the F-Central underground substation the water handling, material handling and mining activities will be fed to a network of 11 kV miniature substations and low voltage reticulation.

The North Portal substation will feed from the Main Consumer substation by means of a ring feed overhead line configuration to the ventilation substations and North Portal substation. The F-North and F-Boundary underground substations will feed from North Portal substation and water handling, material handling and mining activities will be fed to a network of 11 kV miniature substations and low voltage reticulation.

The surface general infrastructure and Underground will be fed from the 11kV network of substations with low voltage supplied by miniature substations and where required, power transformers with associated Motor Control Centers and distribution points.

Allowance were also made for all low voltage infrastructure required for offices, change houses, workshops, water handling, material handling and general area lighting.

18.4.5 Alternative Power Supply

At the Main 11 kV Consumer substation, two 2.2 MVA Diesel Generators will be installed and connected to 11 kV network. The requirement for power during an Eskom outage will be for surface fans for underground ventilation and underground water pumping to prevent flooding plus essential process plant equipment as well as lighting and fire services. All other activities will cease and can only resume once the Eskom supply can be restored.

18.5 General Surface Services Infrastructure

18.5.1 Fuel and Lubrication Offloading and Storage Facilities

Fuel and lubricants will be delivered to the mine by delivery trucks or tankers. Fuel and lubrication offloading and storage facilities are provided for at each portal and are adequately sized to cater for 4 days of operation during steady state. These facilities will be suitably isolated from nearby infrastructure and suitably ventilated. The storage containers will be self-bunded to prevent fuel contamination.

18.5.2 Fire Protection Facilities

Fire pump stations will be constructed as part of each portal’s infrastructure.

Fire water will be supplied from a dedicated portion of the potable water tanks for the surface fire protection system.

Each fire pump station will be equipped with a pressure maintenance (Jockey) pump, a primary electrical pump and a secondary diesel pump if power is not available.

Adequate fire ring mains will be installed within the surface infrastructure and underground infrastructure, to meet the minimum requirements, for a fire protection system. Fire hydrants and hose reels will be connected to the ring mains. Every hydrant will have a designated fire hose cabinet containing two 30m length of hoses with an instantenous coupling and a nozzle.

Fire water for the underground fire protection system will be supplied from a dedicated portion of the surface service water tanks.

The underground fire water system for conveyors will make provision for fire hydrants and hose reels with a sprinkler system included at the transfer stations, head and tail ends of selected conveyors (not the full length of the conveyor) and will extend for 20 metres (m) along the length of the conveyor. Spray nozzles will be located to spray optimally on both the carry and return belts. Linear heat cables will be installed between the main and return belts on conveyors with the Infra-Red detection system.

Pressure reducing stations will be used to regulate the water pressure at all points on surface and underground.

Fire extinguishers will be located around the mine where water is either not suitable for fire extinguishing or simply not available. Portable fire extinguishers will be positioned at the entrance of each building.

Electrical switchgear and electrical motor control centres will be protected with dry power canisters inside the panels to automatically deploy if a fire or arc is detected. The systems will comprise an early warning detection system connected to the fire indicator panel.

18.5.3 Fire Water Supply Requirements

The fire system (supply, storage and distribution) will be designed in accordance with SANS 10400W, and NFPA codes were required.

Adequate fire mains will be installed within the surface and underground infrastructure to meet the minimum requirements for a fire protection system.

The sizing of the fire main and the water pressure, required within each section of the system will be adequately designed to meet the minimum requirements of the applicable code/regulation, for all of the fire protection systems installed.

The surface fire main system will be dedicated solely for the purposes of fire-fighting and no other feed should be allowed to be taken off the fire main system for process or domestic water purposes.

Underground firewater will be taken from the mine service water supply and reticulation. Surface firewater will be taken from the potable water tank.

A surface firewater ring main system will be provided for the mine footprint. The ring main will be buried and divided into sections by accessible isolation valves so that any damage to one section of the ring main will not compromise the fire-fighting capability of the entire system. Below ground section of the fire water mains should be run in six inch PN16 HDPE piping as a minimum.

The aboveground fire water mains piping will be carbon steel piping and painted red.

18.5.4 Change houses

Change house facilities will be provided at each portal complex as well as the concentrator plant. The change house infrastructure will be capable of accommodating 2,000 people in total.

18.5.5 Office Accommodation

The mine administration offices will be located at each portal but outside of the mining area security fence. The office building at the central complex will house the mine management and shared services personnel.

18.5.6 Temporary Construction Camp

A temporary construction camp will be established on Harriets Wish, the property just south of Ketting where the mine is located. Specific areas are allowed for contractors of different trades. The estimated labour force during peak construction amount to 2,248 people as indicated in the table below:

Table 18-1: Temporary Construction Labour Estimate

Contractor	No of People
Mining	900
Earthworks / Civil	200
Mechanical	88
Electrical	60
Concentrator Plant	1,000
Total	2,248

The initial stages of construction (about 4 months of portal construction) will require the civil/earthworks team to be accommodated after which the labour force is expected to ramp up to the full contingent of 2,248 people over a period of 6 months.

The temporary contractor accommodation facilities will be provided and managed by the owner. Costs for the accommodation and meals would be considered in the contractors rates during the tender phase of each of the contracts.

The abovementioned assumptions were used to determine the capital costs associated with establishing and servicing the construction camps for the financial model.

18.5.7 Communication Systems

18.5.7.1 Surface communication systems

The surface communications will consist of the following:

- Telecommunications network
- IT Network
- Control Network
- Radio Network

Telecommunications network will consist of an external supplier providing a data link to site. Telephone communication will be via data connection substituted with cellular. On site data network will be fibre network interconnecting all infrastructure and underground operations with redundancy capabilities.

The control system with centralised control, SCADA and HMI interfaces will be connected via fibre network for surface and underground operations.

A radio network will also be available for site communications and operational staff.

18.5.7.2 Underground communication systems

Underground telephones will be installed at all substations, conveyor drive, loading stations, pump stations, refuge areas and assembly points.

A leaky feeder will be installed for underground communications. The leaky feeder will be linked with the aid of radios to all key personnel required to communicate on a frequent basis.

For all monitoring and detection functions such as fire, access control, pump station and material loading control, sysmic, to name but a few, will be accommodated with a fibre network back to the surface main control room, portal control rooms and SCADA systems where required.

The centralised blasting system will also be controlled from the control room via the underground fiber network.

18.6 Underground Infrastructure

The underground infrastructure consists of:

- The decline conveyors from the three decline portals will feed the surface overland and silo feed conveyors. The decline conveyors, approximately 1,000m in length, will commence at the sub-level conveyor transfer points under the underground ore silos.
- The underground ore silos will be equipped to receive Run-of-Mine (ROM) ore from the sub-levels via multiple haulage and spiral ramps. LHDs will be used to load the material into dump trucks, which will in turn transport the material to the nearest underground silo / surge bin above the conveyor declines.
- Grizzlys and rock breakers will be installed on top of each o / surge bin to ensure the design material lump size is not exceeded.
- Vibrating feeders will be used to extract the ore from the silos and feed the decline conveyors for transport to surface.
- Vibrating feeders will be used to extract the ore from the silos and feed the decline conveyors for transport to surface.
- Associated mechanical equipment and facilities that includes Maintenance Cranes, Pump Stations, Water Clarifiers, Workshops and water storage will be provided.
- Associated services such as Compressed Air, Mine Service Water, Potable Water, Mine Return Water, and Fire Protection System will be provided.

18.6.1 Ore and Waste Handling Systems

The underground infrastructure for ore handling will comprise the following:

- A footwall conveyor system to transport ore from the mining areas to the main underground silos.
- The capacity of the main silos will be sized to accommodate at least a full shift’s production.
- Small surge silos to allow for intermediate loading of material from a 40t truck onto the conveyor system. Each of these silos would be equipped with fixed grizzly and rockbreakers to limit the material lump size to 350mm before it gets onto the conveyor. No provision has been made for LHDs to load material directly onto the conveyor system.

- Vibrating feeders installed under each silo to control the material feed rate onto the conveyor system in order to prevent silo discharge hung-up or spillage.
- Sacrificial transfer conveyor to feed material from the underground main silo to the main decline conveyor. The main decline conveyor would transport the material directly to the surface overland conveyor system.

Waste generated during decline development and access development would be transported directly to the surface stockpile using 40t trucks. No conveying of waste material has been considered.

18.6.2 Mine Ventilation and Cooling Design

The underground ventilation system infrastructure will comprise of multiple exhaust and intake shafts with multiple fans installed on these shafts.

These fans will be main fans and located on surface.

Auxiliary fans will be installed underground to ventilate the underground workings.

When underground mining exceeds 700m (vertical depth) below surface the fresh air alone will not be sufficient to keep the wet bulb temperature below the recommended 29°C.

Therefore, bulk air coolers will be installed on surface to cool the intake fresh air supply for the underground workings.

18.6.3 Compressed Air System

The primary use of compressed air will be limited to refuge bays underground.

Miscellaneous pneumatic small workshop tools will be powered by portable compressor units.

A total of 6000 CFM has been allowed for and will be supplied by four screw compressors located at the surface compressor house.

18.6.4 Fuel and Lubricant Distribution

A fuel bay and lubrication bay will be established at the underground main workshop.

Diesel powered equipment within the underground workings will visit the underground workshop for re-fueling and maintenance.

When the traveling distances to the workshop become prohibitive, cassette carriers will be used to distribute diesel fuel and lubrication oil from the main underground workshop to the satellite workshops at the working areas.

18.6.5 Water Management and Dewatering Systems

Mine service water will be transported to underground and distributed using steel pipes. Service water will be used for drilling equipment and cleaning of areas. Flexible hoses will be used to connect drilling equipment to the supply pipelines.

Potable water will also be transported to underground and distributed using steel pipes. Potable water will be used as a source of fresh drinking water.

Pressure reducing stations will be used to regulate the water pressure for potable and service water to a level that can be used by personnel and machinery.

Sources of water inflows to the underground mine include groundwater (fissure water) inflows, drilling water, cleaning water, drinking water, and water used for firefighting. This water, collectively referred to as dirty water, will gravitate to the lowest point or sump at point of generation and be picked up by submersible or vertical spindle pumps.

Dirty water on sub-levels will be directed by gravity or pumps to pump stations located at no more than 750m intervals, from where it will be pumped horizontally to a dirty water pumping station.

Dirty water pumping stations will be located to provide a 100m vertical interval from one dirty water pump station to the next until the underground high rate settlers is reached. The settler infrastructure will utilize a water treatment method known as lime neutralization to treat the dirty water. The lime treatment essentially consists in bringing the pH of the dirty water to a point where the metals of concern become insoluble and precipitation occurs.

The slurry is then brought in contacted to a flocculant and fed to a settling tank for solid/liquid separation. The sludge is collected from the bottom of the settling tank and can either be pumped to a storage area or pressure-filtered to increase its density prior to transportation. The settler overflow is discharged into a clean water dam for pumping to surface.

A clean water pump station will be established at the main underground station to pump the clean water from the clear water dam to surface.

The main dirty water pipelines and the clean water pipelines will be capable of dewatering no less than $360m^3$ per hour.

Once on surface, the clean water is recycled and sent back underground as service water.

18.6.6 Underground Workshops

The underground main workshop should be operational before stoping commences in that mining area. The main workshop would be located close to the centre of the orebody and fitted with a discrete workshop, service bay, refuelling bay and wash bays. Water from the wash bays must go to an oil separation unit. Spare part stores, hoses and a tyre bay should be adjacent to the workshop with space for a tyre handling facility.

Satellite workshops would be established in close proximity to the mining areas. These would be equipped with a concrete slab, a single ramp, good lighting, minimal lifting gear and a small store. Activity in the satellite workshop would be limited to refuelling, daily examinations and lubrication.

Drill rigs travelling in confined spaces are subject to excessive boom and feed damage. If workshops are not constructed timeously, it is recommended that mobile servicing units service drill rigs daily. It is also recommended that drill rigs and LHDs are refuelled at, or close to, the mining areas.

18.7 Underground Logistics

To support the planned production rate, diesel-powered mobile mining equipment comprising 40ton dump trucks, 17t LHDs, UVs, LDVs, cassette and cassette carriers, and personnel carriers will be used to transport rock, materials, equipment and personnel to and from the underground.

Provision will be made for underground personnel waiting places, refuge bays, workshop facilities, equipment parking bays and toilets.

The designated mobile equipment parking bays would be in close proximity to the satellite workshop to facilitate shift change over.

18.7.1 Ore Transportation

Ore loaded in the mining sections will be transported from the stopes with LHDs. The LHDs will transfer the ore to 40t trucks, which will transport the ore to the main underground silos or intermediate surge silos.

Early in the LoM, waste from decline development and ore from initial mine development will be hauled in 40t haul trucks directly to a waste and ore stockpiles on surface.

Once the main decline conveyor system is commissioned, ore will be conveyed from the underground silos to the surface overland conveyor system.

At the transfer point between the main decline and overland conveyors, a throw out conveyor will be built to dump reef in a mucking bay. The dumped reef will be re-handled and dumped onto the reef stockpile using surface mobile equipment.

18.7.2 Material

Underground consumables will be delivered via diesel-powered utility vehicles. These will typically be 8t cassette carriers with the capability to offloading cassettes containing supply materials and equipment at the destination, and self-loading empty cassettes for removal to surface.

Another type of material transport equipment will be the 3M4T, a 4t utility vehicle equipped with an onboard crane to offload and load material as well as carry a maximum of 3 people.

18.7.3 Equipment

The mining equipment will be sized and selected based on the suitability for underground mechanized mining application.

Underground mobile equipment will be equipped with engines utilizing technology to optimize efficiency and to minimize emissions.

Collision avoidance and people detection devices will be fitted to all underground mobile mining equipment.

Steering and break interlocking safety systems, on-board fire suppression systems and engine protection systems will be standard features.

18.7.4 People

Underground workers will be transported from surface by means of dedicated personnel carriers from designated pick-up points.

Once underground, workers will be required to walk from the drop off point to their respective working places. No truck haulage will take place during major shift movements.

18.7.5 Fuel

Diesel fuel for underground mobile equipment will be delivered by pipeline to a receiving tank at the underground workshop re-fuel bay. Initially, prior to the installation and commissioning of the pipeline, diesel fuel will be delivered to underground using cassettes in addition to refueling on surface.

Diesel powered equipment within the underground workings will visit the underground diesel fuel bay to refuel as required.

18.7.6 Explosives

Explosives for development and production will be of the emulsion type.

Emulsion will be delivered by the manufacturer to the bulk silos on surface. From the bulk silos, emulsion will be loaded into emulsion cassettes for delivery to underground as required. Blast holes will be charged pneumatically using emulsion charging vehicles.

The manufacturer will be responsible for delivery of explosive accessories to the designated places on surface. Explosive accessories will be delivered to underground explosive storage boxes using the appropriate underground vehicles.

18.8 Waste Facilities

Operational and domestic waste handling facilities will be provided at each mining operation where administrative and operational activities are to be conducted.

The following waste handling areas will be provided, either separate or combined depending on mine layout and requirements:

- Operational waste separation for salvaging of mine equipment and scrap.
- General domestic waste such as produced by the offices will be separated into organics and recyclables (Metals, plastics, glass, paper etc.)
- Hazardous storage areas for hazardous waste requirements such as, batteries, lubricants, and other hazardous substances will be provided for and be disposed of by an accredited service provider.
- Medical waste disposal facilities would be provided for each of the portal first aid stations and the Central Medical station facility. From the Central Medical station, the medical waste will be disposed of by an accredited service provider.

18.9 Stockpiling and Reclamation

The stockpiling and reclamation methodology allows for two separate facilities to stockpile ore produced from the three mining complexes on site. These facilities will cater for stockpiling material produced from first production of ore to the start of each of the two process plant modules. Post plant start-up these facilities will remain in operation for any future stockpiling and reclamation requirements of ore over life of mine.

The design of these facilities allows for the separate stockpiling of the two different ore types mined (T-Zone and F-Zone). These ore types are viewed to be of marginally different ore potential and would therefore be required to be processed either separately or in a controlled blend in the process plants to maximize process plant recovery.

The position the separate stockpile facilities are located adjacent to source or on route to the process plant to minimize cost of additional infrastructure from sources to a central stockpiling facility. Therefore the one stockpile is located adjacent to the south mining complex and is dedicated to ore mined from the south complex i.e. T-Zone. The second stockpile is located adjacent to the central mining complex and will accommodate the F-Zone ore mined from the south, central and north mining complexes.

Ore from underground operations will initially be trucked to surface from where ore of equivalent metallurgical characteristics will be stockpiled together in an area adjacent to the stockpile re-loading stations. Once the decline conveyors are operational the design of these conveyors cater for a divert system where ore will be discharged onto a loading pad from where material will be transferred to the stockpile by means of a Front-end Loader (FEL) or a combination of FEL and tipper trucks or equivalent.

Based on the current mine production schedule it is anticipated that a total stockpile size will be in excess of 500,000 tonnes prior to the start of each of the two process plant modules. During the initial months of plant operation the plants will be fed from a combination of ore mined and ore reclaimed from the stockpiles.

As mining production increases the reclaimation of ore from the stockpiles will proportionally decrease. Ore from the stockpiles will be reclaimed by manually reloading ore via the reloading station provided onto the conveyors that feed the coarse ore silos at the process plant. The opportunity presents itself to optimize the size of the stockpile facilities in future phases of the project once a more defined mine production schedule is available.

The stockpile facilities are designed to provide the necessary flexibility to manage the ore characteristics to the process plant thereby allowing for the ability to feed the plant with a controlled blend of ore to the mills. This flexibility is viewed of significant importance to allow the plant the means to affect and optimize metallurgical recovery.

18.10 Central Assay Laboratory

The design allows for a centralized analytical laboratory that will cater for the analytical requirements for both that of underground mining, metallurgical process plants, environmental and fleet lubrication samples. The analytical laboratory has the capacity to process approximately 350 samples per day for a 600ktpm mine with 5 different reefs plus 2 large plant modules seems low for the necessary grade control and everything else that goes with a large mine and will cater for amongst others the analysis of:

- PGMs (fire assay and MS ICP)
- Base Metals (ICP and AA)
- Sulphur (Leco)
- Wet chemistry
- Water sample analysis for environmental management

The laboratory facility is located adjacent to the main administrative complex and consists of a laboratory building fully equipped inclusive of the amenities and utilities required to run the facility e.g. offices, server rooms, Laboratory Information Management System (LIMS), dust extraction, fume extraction, demineralized water system, compressed air, reticulation for gas, chilled water system.

The current allowance caters for the manual operation of the analytical laboratory therefore the operating costs estimate allows for the full complement of 34 management and staff members to operate the laboratory. An alternative option would be to consider a fully automated robotic analytical laboratory. The capital cost of such a facility is approximately double that of the manual laboratory but the staff complement is halved. It is recommended that a more detailed assessment be conducted in the future stages of the project to consider these options against each other in a more detailed manner once a more accurate estimate of the mining sample and analysis requirements are available. Such an assessment can further consider 3rd party regional laboratories.

18.11 Residue Disposal Facility

Epoch Resources (Pty) Ltd *(Epoch)* was appointed by Advisian (Pty) Ltd (*Advisian*) to undertake the Pre-Feasibility Design (*PFD*) of the Residue Disposal Facility (*RDF*) and its associated infrastructure.

18.11.1 RDF Design Criteria

Table 18-2: Design Criteria associated with Waterberg's RDF

Description	Value	Unit
*Design Life of Facility	20	Years
Processed Ore	Platinum	
*Residue Deposition Rate:		
Ramp up from Year 1 – Year 3:	2 400 000 – 6 840 000	Dry tpa
Steady State Production: Year 4 – Year 20	6 840 000	Dry tpa
Particle Specific Gravity	2.9	
In-situ Void Ratio	1	
Particle Size Distribution	80% passing 75 µm sieve	
Placed Dry Density of Residue (RDF)		
Sub-aqueous	1.4	t/m^3
Sub-aerial	1.6	t/m^3
Average	1.5	t/m^3

* Mine planning currently based on shorter LoM. Depending on the extent of the deviation from these values, the RDF can be revised to accommodate any changes by adjusting the height.

18.11.2 Site Selection

A site selection study was undertaken to find the most favourable site. The study found that Ketting farm was the most favourable based on:

- The topography is such that it will not require expensive measures to contain the tailings;
- The risks associated with this site were deemed the lowest out of the other options due to its location relative to people and mining infrastructure; and
- The site is located within the mine tenement area.

18.11.3 Geochemical Testing and Liner Requirements

Geochemical test work undertaken by GCS Environment Engineering (Pty) Ltd (*GCS*). Residue samples from the T and F ore zones were tested. Based on the total concentration tests (*TCT*), the concentrations of 4 elements were found to be above the limiting concentrations for Waste Type 4, therefore the tailings was classified as Waste Type 3.

According to the: *National Environmental Management Waste Act* , a Waste Type 3 requires a Class C liner system. A Class C liner system comprises:

- 1.5 mm HDPE geomembrane;
- 300 mm Compacted Clay Layer (CCL); and
- A leakage detection system below the liner.

A source of clay* was identified for use in the liner, however the suitability of the clay has not yet been confirmed.

**The clay source was identified late in the project, as such the drawings show a Geosynthetic Clay Liner rather than a CCL, as it was previously understood that there was no nearby source of clay. The costs have however been adjusted for clay rather than the CCL.*

18.11.4 Depositional Trade-off Study

A trade-off study was undertaken to determine a suitable depositional methodology as well as to highlight the advantages and disadvantages of each methodology. The following methodologies were investigated:

- Conventional/thickened tailings;
- Cycloned tailings;
- Paste tailings; and
- Dry-filtered tailings.

The following conclusions were drawn from the study:

- Paste disposal is untested in the platinum industry and would pose a significant risk and require an extensive testing regime to consider implementing;
- Dry Stacking is a possible option and the potential water recoveries could make this option feasible, however the high capital and operational costs associated with dry stacking may not be feasible compared to a conventional/thickened tailings dam;
- Cycloned tailings may provide a cost saving due to the higher rates of rise achievable, however test work is required prior to recommending this option;
- Conventional/thickened tailings are the safest option and well understood in the platinum industry and has been selected as the preferred option for Waterberg.

18.11.5 Economic Depositional Methodology Trade-off Assessment

An Economic Assessment of the various depositional methodologies was undertaken to determine which methodology would provide a cost effective solution given the scarcity of water at the site. Thickened/conventional tailings, cyclone tailings and filtered tailings were assessed and compared based on the Capital, Operational and Water consumption costs.

The purpose of this assessment was to determine which option would result in the most cost effective solution in terms of water cost, therefore the costs were only taken to a conceptual level. The results, as shown in Figure 18-8, show that filtered tailings will only be feasible if the water cost exceeds R62/m^3 or if water is not available entirely. Therefore conventional/thickened tailings was taken forward as the preferred option for Waterberg.

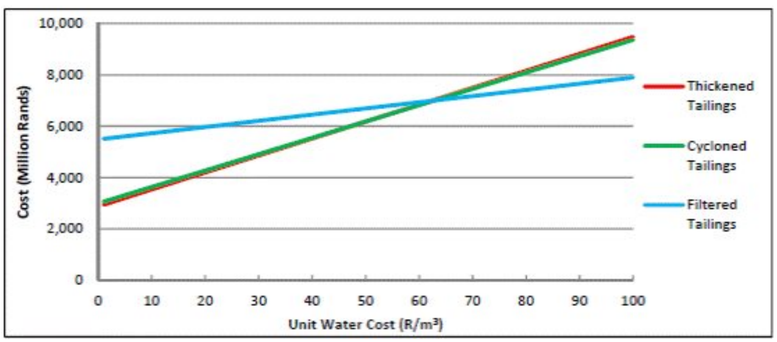


Figure 18-8: Total Costs vs. Water Costs

18.11.6 Key Design Features

The key design features of the RDF (Figure 18-9) are as follows:

- Total design capacity of 140 million tonnes deposited
- A compacted earth fill starter wall at elevation 1000 m.a.m.s.l.;
- A penstock system will be used to decant water from the RDF;
- A RWD with sufficient capacity for the 1 in 50 year storm event (340 000 m ^{3});
- The RDF has a total footprint area of 297 Ha, a maximum height of 55 m and a final rate of rise of <3 m/year;
- A concrete lined solution trench to convey seepage water to the RWD;
- Lined toe paddocks to collect contaminated run-off water from the RDF side slopes; and
- A slurry spigot pipeline along the crest of the RDF.

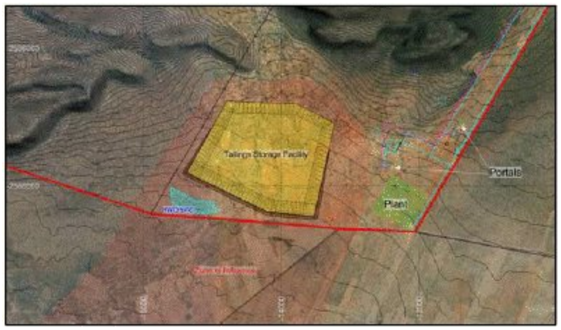


Figure 18-9: RDF Layout

18.11.7 Capital Expenditure

The capital expenses (*CapEx*) associated with the RDF have been estimated to an accuracy of ± 25 percent and are as follows:

Table 18-3: RDF Total Capital Cost

Description		Total
Residue Storage Facility		
Site Clearance	R	21 065 725
Earthworks and Excavations	R	68 130 261
Subsoil Drainage	R	4 875 517
Concrete Structures	R	17 549 666
Pipework	R	11 003 137
Catwalk	R	575 816
Liner and Associated Drainage	R	325 301 644
Miscellaneous	R	538 000
SUBTOTAL	**R**	**449 039 766**
Return Water Dam		
Site Clearance	R	625 000
Earthworks and Excavations	R	6 724 449
Liner and Associated Drainage	R	20 212 611
Concrete Structures	R	165 513
SUBTOTAL	**R**	**27 727 573**
MEASURED WORKS	**R**	**476 767 339**
Contingencies (@ 10% of Measured Works)	R	47 676 734
TOTAL WORKS	**R**	**524 444 073**
Preliminary & General (@ 20% of Total Works)	R	104 888 815
TOTAL CAPITAL COST	**R**	**629 332 888**

The capital costs associated with the RDF have been estimated based on past projects by Epoch.

The clay for the CCL must be transported some distance from the site. An estimate has been provided for the hauling costs by WorleyParsons RSA (Pty) Ltd Trading as Advisian (*WorleyParsons or Advisian*) and 2 other contractors. The rates supplied by Contractor 1 were considerably higher than Contractor 2 and Advisian, corresponding to an additional ±R50 million in CapEx if Contractor 1's rates are used. WorleyParsons and Contractor 2's rates were reasonable however they may be as high as Contractor 1, therefore there is a risk that additional costs may be incurred.

18.11.8 Operational Expenditure

Paragon Tailings (Pty) Ltd (*Paragon*) provided an operating cost estimate. The operating costs associated with the RDF have been estimated as follows (at an accuracy of 25%):

- R 6.31 million/annum for operation and management of the RDF, for tailings production of 288 000 tpm (first 6 years);

- R 7.86 million/annum for operation and management of the RDF, for tailings production of 576 000 tpm (6 - 25 years);
- Additional operating costs may include:
  - R 0.3 million/annum for general works associated with the RDF (i.e. pipe and valve replacements, cleaning out of trenches, etc.); and
  - R 0.4 million/annum for consulting services.

18.11.9 Closure Costs

Closure and aftercare capital costs have been accounted for in the Environmental section, Section 20.6. This cost will also incur over a period of approximately 2 years following mine closure.

18.11.10 Life of Mine Costs

The total LoM cost associated with the Waterberg RDF over the duration of the project life (Feasibility Study to Post Closure) is estimated at R 848 million.

18.11.11 Conclusions

Based on the Pre-Feasibility Design of the RDF as described above it is concluded that:

- Preferred sites have been identified for the establishment of the RDF which has the capacity to accept the life of mine tailings;
- The preferred site for the establishment of the RDF has been identified as being adjacent to the plant area on Ketting Farm;
- Based on the results of the geochemical analyses, the design of the RDF has incorporated provision for the installation of a Class C liner, which comprises:
  - A 1.5 mm HDPE liner,
  - A 300 mm Compacted Clay Layer,
  - A leakage detection system,
- The preferred disposal method (filtered, cycloned and thickened tailings) was determined in a trade-off study, whereby all the costs for the duration of the project for each option were compared. The results showed that thickened tailing provides the lowest cost option for Waterberg;
- An upstream, spigot method has been proposed for the development of the RDF;
- The RDF will be phased in order to defer capital expenditure;
- The total capital cost associated with the RDF has been estimated at R 629.3 million (accuracy of ± 25 percent) where R 469.3 million are required for Phase 1;
- The costs of the Class C liner comprised R 333.9 million (excluding contingencies and P&Gs) of the total cost of the RDF. The costs associated with the CCL were sourced from Worley Parsons and 2 other contractors;
- The operating costs associated with the RDF have been estimated for the life of the facility (accuracy of ± 25 percent) comprising:
  - Residue deposition and operations management (Paragon):
    - R 6.31 million/annum during Phase 1; and
    - R 7.86 million/annum for the remainder of the life of the facility;

  - R 0.3 million/annum for general works associated with the RDF (i.e. pipe and valve replacements, cleaning out of trenches, etc.); and
  - R 0.4 million/annum for consulting services.
- Closure and aftercare costs (accuracy of ±35 percent) have been estimated at R 45.77 million with R 30 million of this cost being incurred in the first 2 years following mine closure;
- The total LoM cost associated with the Waterberg RDF over the duration of the project life (Feasibility Study to Post Closure) is estimated at R 848 million.

18.12 Risks and Opportunities

18.12.1 Risks

- Bulk water sustainable supply
- Bulk water supply line and security of infrastructure
- Local groundwater Reserve could be threathened during ground water extraction
- Locality of the site to be considered when medical treatment facilities and casavac are designed
- Access and regional gravel roads in poor condition
- Local services infrastructure very poor (Communication, internet, electricity, sewage STP, clean water supply)
- Zone of influence of the RDF (if other infrastructure and public is not considered)
- Real estate and location of contractors camp
- Real estate and location of the employee accommodation
- Skill shortage and turnover relating to mechanized mining equipment and location of the mine (maintenance and training)
- Lack of detailed hydrogeological data may lead to underestimating water demand quantity.
- Premature procurement and contract awards prior to completion of DFS

The possible project risks associated with the current RDF design are as follows:

- A geotechnical investigation has not yet been undertaken on the soils underlying the RDF, therefore it is not known whether there could be problem soils in the area, such that an alternative site may be required;
- A clay source has been identified, however the suitability has not been determined, if found unsuitable, additional costs will be incurred in replacing the clay with a GCL;
- The cost estimate for hauling the clay has been highlighted as a risk as the distances required to haul the clay are great and significant costs may be incurred;
- Identification of a suitable borrow area for material for the starter wall has not been undertaken. This will be required to price the cost of hauling material to the site. If no suitable area is identified near the site significant costs may be incurred;
- Geotechnical tests have not been performed on a tailings sample. The design must be reviewed once these tests have been performed. Self-raising, as well as cycloning, is heavily dependent on the strength of the tailings;

- No stability analyses have been undertaken in the phase and is recommended in the next phase of the project. If stability is found to be an issue the configuration of the facility would need to be reassessed;
- A hydrology study has not been undertaken and will be required in order to determine any flood lines that may affect the facility;
- The sourcing of suitable construction materials (e.g. filter sand, stone, geo-fabrics, etc.) must be determined as this could raise costs;
- Construction rates based on Epoch's past projects and not project specific. Acquiring construction rates from a contractor is required in the next phase of the project;
- The facility is upstream of a river and homesteads, which means that the facility will be ranked as a high risk facility; and
- A geophysical study has not been undertaken to identify possible faults or dykes in the footprint of the RDF; and
- A geo-hydrological study has not been undertaken to identify any possible aquifers under the RDF footprint and how the RDF may affect them.
- The availability of skilled labour resources, for both construction and operational phases, is limited and the training and skills development program will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Waterberg Project.

18.12.2 Opportunities

- Water from the STP (WWTW) will reduce OpEx as bulk water cost per unit will be lower
- Load out of stockpile and reclaim facility to mitigate production loss risk and allow blending of different products during the reclaim process.
- Plant BOS transfer belts feeding primary crusher circuit could be removed.

The possible opportunities associated with the current RDF design are as follows:

- The potential relaxation of the liner requirements may be possible based on a letter from the Department of Water and Sanitation (*DWS*) which may reduce the costs associated with the liner. There may be a case to motivate this to the DWS, however additional studies will be required, some of which are highlighted in the recommendations below;
- It may be possible to convert the RDF to a cyclone facility if the material characteristics are suitable. This would require extensive test work, however this could reduce the footprint area of the facility, thereby reducing costs associated with the liner; and
- Additional capacity is available if the need arises for a greater storage volume.
- It may be possible to convert the RDF to a cyclone facility if the material characteristics are suitable. This would require extensive test work, however this could reduce the footprint area of the facility, thereby reducing costs associated with the liner; and
- Additional capacity is available if the need arises for a greater storage volume.

19. Market Studies and Contracts

19.1 Market Review and Metal Prices

No formal marketing study has been completed as part of this PFS but as one of the project owners is a PGE concentrate producer, the marketing aspects are understood 'in house'.

There has been a significant growth of 'independent' concentrate producers during the last 15 to 20 years and as such toll treatment of flotation concentrates or purchase agreements are common within the PGE industry with the major producers. Waterberg will be one of the future 'independent' concentrate producers and initially, a concentrate sales agreement will be required to be formalised to treat the production from the mine.

No smelter operators have been formally approached to express interest in the toll treatment of the Waterberg concentrate, but informally, there is reported to be significant interest in processing this flotation concentrate.

19.2 PGM and Base Metal Contribution

Based on the project revenue calculations and the base case metals pricing, the contribution from the 'pay metals' is indicated in the following table. This is based on the 'prill splits' for the two major geological zones to be mined, but clearly indicates that the PGE's are the major revenue contributor at more than 87%.

Table 19-1: Revenue Contribution in Concentrate

	T-Zone	F-Zone
Pt	32%	36%
Pd	31%	45%
Au	23%	6%
Rh	1%	1%
4E's	**87%**	**89%**
Ir	1%	1%
Ru	0%	0%
Cu	7%	2%
Ni	5%	8%
TOTAL	**100%**	**100%**

The markets and prices for platinum, palladium and associated PGM metals have been extremely volatile over the past several years. The Company relies on research from various third party providers to assist in the analysis and understanding of supply and demand trends.

PGM prices experienced significant drops in 2015 based on a number of factors including negative macro market sentiment and ETF selling while underlying demand from autocatalyst manufacturers and industrial users was very strong.

Supply from South African producers recovered strongly in 2015 from the negative impact of a protracted strike in 2014 while the supply of recycled metal has dropped off due to a steep fall in car scrappage. The depreciating South African Rand has muted USD denominated metal prices whilst benefiting South African producers.

Based on research from Johnson Matthey (PGM Market Report - May 2016) both platinum and palladium markets showed fundamental deficits during the 2015 calendar year. Continued deficits are forecast for 2016. Multi-year deficits have been made up from the sale of existing surface stock and pipeline inventories. Although the size of readily available stock of PGM's is opaque the massive drawdown of stockpiles in recent years likely removes future flexibility in delivering non-mined supply and could add support to prices going forward.

In 2016, thus far the precious metals markets have rebounded with positive sentiment for gold leading to fresh interest in platinum and palladium. Global auto sales and autocatalyst demand for both diesel and gasoline remain strong. Supply from deep level, conventional UG2 mining operations in South Africa continue to struggle with the possibility of labour action looming. After a number of years of cyclically depressed prices, the PGM markets appear to have stabilized.

Table 19-2: Platinum Supply and Demand

Platinum Supply and Demand '000 oz			
Supply	**2014**	**2015**	**2016**
South Africa	3,537	4,569	4,288
Russia	700	670	679
Others	871	837	932
Total Supply	**5,108**	**6,076**	**5,899**
Gross Demand			
Autocatalyst	3,241	3,433	3,497
Jewellery	2,897	2,827	2,929
Industrial	1,755	1,749	1,919
Investment	277	451	332
Total Gross Demand	**8,170**	**8,460**	**8,677**
Recycling	-2,071	-1,725	-1,917
Total Net Demand	**6,099**	**6,735**	**6,760**
Movements in Stock	-991	-659	-861

Palladium Supply and Demand '000 oz			
Supply	**2014**	**2015**	**2016**
South Africa	2,125	2,683	2,521
Russia	2,589	2,434	2,487
Others	1,374	1,309	1,382
Total Supply	**6,088**	**6,426**	**6,390**
Gross Demand			
Autocatalyst	7,462	7,629	7,757
Jewellery	272	225	215
Industrial	2,076	2,138	2,185
Investment	943	-659	-295
Total Gross Demand	**10,753**	**9,333**	**9,862**
Recycling	-2,752	-2,460	-2,629
Total Net Demand	**8,001**	**6,873**	**7,233**
Movements in Stock	-1,913	-447	-843

Source: Johnson Mathey, PGM Market Report, May, 2016. www.platinummatthey.com

Base metals are relevant in terms of overall project return. As with all industrial commodities prices continue to be volatile. Nickel and copper markets are closely linked to Chinese demand which continues to be difficult to predict.

The Company is listed on the NYSE-MKT exchange and requires that economic studies consider trailing average prices over a 3-year period. These prices have been calculated and updated.

In summary, the metal prices considered in this Technical Report are summarised in Table 19-3 below.

Table 19-3: Three-Year Average Trailing Metal Prices

Price	3 Year Trailing Average as at 31 July 2016
Platinum (US$ / oz.)	1,212
Palladium (US$ / oz.)	710

Price	3 Year Trailing Average as at 31 July 2016
Gold (US$ / oz.)	1,229
Rhodium (US$ /oz.)	984
T-Zone Basket 4E (US$ / oz.)	967
F-Zone Basket 4E (US$ / oz.)	889
Copper (US$ / lb)	2.56
Nickel (US$ / lb)	6.1
ZAR/US$	15

19.3 Concentrate Production and Quality

The Waterberg Mine project will be producing a flotation concentrate, which is to be sold, or toll treated such that the project receives revenue from the contained economic metals within the concentrate at a negotiated payability. It is expected that the project will produce up to 23 000 tonnes of concentrate per month at steady state production or in excess of 285 000 tonnes per annum.

The quality of this concentrate has been evaluated during the metallurgical test work programme conducted at Mintek, Johannesburg. Whilst this is a 'snapshot' based on a few samples from drillcore, the table below indicates what is anticipated to be produced and thus treated in the subsequent recovery process in terms of economic metals and elements of interest.

Table 19-4: Concentrate Quality — Major Elements

		Concentrate Contents		
Element	Units	Individual	Minimum	Maximum
Pt	(g/t)	24	9	35
Pd	(g/t)	62	18	69
Rh	(g/t)	2	1	2
Ru	(g/t)	1	ND	ND
Ir	(g/t)	3	ND	ND
Au	(g/t)	3	2	27
2E+Au	(g/t)	89	30	108
5E+Au	(g/t)	95	ND	ND
Cu	(%)	1.9	1.0	9.2
Ni	(%)	2.5	1.1	5.0
Fe	(%)	14.5	11	22
SiO2	(%)	41.3	23	43
MgO	(%)	20.8	6	24
S	(%)	6.5	3	19

Source: Mintek MF2 phase 1a

The minor elements that have been evaluated during the test work programme are indicated in the following table and show the potential for deleterious elements being fed into the subsequent recovery process. There are no expected deleterious elements indicated in the flotation concentrate.

Table 19-5: Concentrate Quality — Minor Elements

Waterberg Concentrate Minor Elements (Nominal)

Al	%	1.56	Ag	ppm	8.4
Cl	%	0.03	Cd	ppm	<0.05
K	%	0.04	In	ppm	5.7
Ca	%	1.63	Sn	ppm	<0.05
Ti	%	<0.05	Sb	ppm	<0.05
V	%	<0.05	Te	ppm	4.5
Cr	%	0.06	I	ppm	<0.07
Mn	%	0.08	Cs	ppm	<5
Co	ppm	711.8	Ba	ppm	36.3
Zn	ppm	678.6	La	ppm	<12
Ga	ppm	<0.05	Ce	ppm	<2.6
Ge	ppm	<0.05	Hf	ppm	<2.0
As	ppm	<0.05	Ta	ppm	712.1
Se	ppm	28.1	W	ppm	<1.2
Br	ppm	3.1	Hg	ppm	2.0
Rb	ppm	6.5	Tl	ppm	3.8
Sr	ppm	36.1	Pb	ppm	66.0
Y	ppm	4.4	Bi	ppm	<0.5
Zr	ppm	6.3	Th	ppm	11.6
Nb	ppm	2.5	U	ppm	5.0
Mo	ppm	9.8			

Additional economic metals, which may be considered for the project, include cobalt and silver from the above table, although the revenue stream, which may be generated from these metals, will be insignificant.

The mineralogical composition of the concentrate is as detailed in Table 19-6 below.

Table 19-6: Concentrate Mineralogical Composition

Mineral	Primary Cleaner Concentrate	Sec. & Tert. Cleaner Concentrate
Pentlandite	12.46	12.39
Pyrrhotite	4.83	6.06
Chalcopyrite	14.76	3.51
Other Sulphides	0.34	0.13
Silicates	27.39	22.39
Serpentine	12.47	19.69
Talc	24.42	32.59
Fe-Oxides	1.80	1.70
Dolomite	1.22	1.14
Others	0.31	0.40
TOTALS	100.00	100.00

Based upon the expected flotation concentrate quality, the product is regarded as a 'desirable' feedstock into the subsequent recovery process for blending with other PGE bearing concentrates.

19.4 Concentrate Treatment Options

Flotation concentrate containing PGE's and base metals are traditionally smelted in the South African industry. The smelting process delivers a furnace or convertor matte containing greater than 96 to 98% of the economic metals and a low grade furnace slag or waste product. The convertor matte is leached using sulphuric acid in a hydrometallurgical base metal refining process which delivers copper as cathode, nickel in various forms but usually as a nickel salt, an unleached 'sludge' containing the concentrated PGE's and a leachable impurity stream containing any impurities recovered into the furnace matte.

The PGE 'sludge' is further treated in the precious metal refinery which delivers refined high purity metal or metal salts for marketing.

There would be reluctance on the part of the South African government to allow concentrates to be sold 'off-shore' as this would go against the stated intent to beneficiate locally. Intermediate higher value products such as furnace or convertor matte and the PGE 'sludge' could be treated off-shore for a limited period of time as local facilities are expanded to treat this material.

A number of alternative treatment options have been proposed and some have been piloted but there are no commercially available alternatives to the smelting route.

This study assumes that the conventional treatment route will be adopted for the recovery of the economic metals contained in the flotation concentrate.

19.5 Material Contracts

There are no material contracts in place for the Waterberg Project apart from those related to the Resource definition drilling, confirmatory metallurgical testing and the completion of the current PFS.

High level discussions with the national grid power provider (ESKOM) and the local water authorities are progressing, but no supply agreements or contracts have been entered into. Eskom has entered into a contract to work on power supply engineering and the permitting process for the power line has commenced.

19.6 Capacity Available Locally

There are four PGE producers, which have downstream smelting, and refining capabilities within the South African industry. Currently there is furnace and refinery capacity available for additional concentrate treatment from independent producers such as the Waterberg Project. One of these four smelter operators is installing additional smelter capacity during the next few years.

The production from Waterberg Mine will place a high demand on the available smelting capacity.

Blending the low chromitite Waterberg concentrates with the high chromitite UG2 concentrates will manage the negative impacts of the higher sulphur and iron content to the benefit of the project and the smelter operator using conventional smelter technology.

Elsewhere in Southern Africa, there is limited smelting capacity in Zimbabwe and Botswana, which could be considered, but this would require statutory approval and is expected to be a short term solution during the ramp up phase of the project only.

It is estimated that there is adequate available smelter capacity for the Waterberg project but steady state production will place a significant strain on this capacity. It is expected that additional smelting capacity will need to be constructed to be able to treat the flotation concentrate from Waterberg and the other potential Platreef miners. Whilst this may be a considerable investment, the proposed off-take agreement terms are in place to offer an adequate return on this investment, either as part of the Waterberg project or as an independent smelter operator.

19.6.1 Alternative Treatment Option

There are alternative hydrometallurgical treatment options, which could be considered to be applicable to the Waterberg concentrates, but unfortunately, none of these are proven on a commercial scale.

Significant developmental test work would be required before any of these processes could be considered for treating the concentrate.

19.7 Smelting and Refining Contracts

No smelting or refining contracts are in place for the Waterberg Mine and only early discussions have commenced with some potential interested parties.

19.8 Metal Payability or Treatment Terms

Typical economic metal recoveries for the conventional smelting and refining route are between 96% and 98%.

There are a number of tolling agreements in place between the different smelter operators, but they can be summarised into two categories:

- A negotiated payability for each economic metal in the flotation concentrates which includes a provision for the treatment charge. The payability can vary between 80% and 86% depending upon the operator and the desirability of the concentrate.

- A negotiated payability for each economic metal plus a treatment charge for the concentrate and a refining charge for each contained economic metal in the concentrate. The payability for this option is as high as 95% or more and the treatment charges can be variable, depending upon the desirability of the concentrate.

The former of these options is the most common in use in the South African PGE industry for independent concentrate producers.

It is proposed that the financial evaluation be based upon a fixed payability %age of an average of 85% for PGE's, 73% for copper and 68% for nickel. These are regarded as fair and reasonable although negotiations may change these terms based on the desirability of the concentrate. A similar Merensky concentrate would be expected to attract a payability of between 83% and 84%.

Trade off studies associated with concentrate quantity and quality should use a payability of 96% with a treatment and refining charge of R6 000 per tonne of dry concentrate.

19.8.1 Concentrate Transport

The concentrate will be transported by the project to the available smelters within South Africa. There are three smelting hubs within South Africa, namely Polokwane (109km south east), Northam (312km south west) and Rustenburg (417km south south west).

As no negotiations have commenced for the off-take agreement, it is assumed that 33% of the concentrate production will be shipped to each smelting hub, resulting in an average distance for shipping of 280km.

The average transport cost for concentrate is based on the actual shipping cost for PTM's Maseve mine at R1.42 per wet concentrate tonne per kilometer. The concentrate moisture will be 12% thus resulting in the cost per dry tonne delivered being R452 which is based on transport rate and moisture content reduction.

19.9 Payment Pipelines

The PGE smelting and refining process from concentrate to refined metal takes a significant amount of time and this is reflected in the payment terms in conventional toll smelting agreements. There is no reason to believe that the Waterberg concentrate will be smelted and refined more quickly than any other concentrate being treated at a toll smelting facility.

Each of the payable metals, namely Pt, Pd, Rh, Au, Cu and Ni has a different 'release' period from the tolling facility, but for simplicity most operators apply a fixed 'release' period to all metals following acceptance of concentrate.

It is expected that the negotiated metal release terms will have metals fully available after 16 weeks for all metals. In terms of payment, there are mechanisms which have been adopted for the project whereby an upfront payment for 85% of the contained metals is available during the month of delivery, subject to an interest charge of LIBOR (US$ basis for 1 month at approximately 0.529%) plus 3.9%. The balance of 15% of the payment will then be available after the full 'release' period of 16 weeks.

This debtor financing arrangement is common in the PGM and gold market and indicative terms have been provided to PTM. The pre-payment option is expected to be in place for the duration of the LOM.

These payment arrangements can be reviewed during the production period, subject to the ongoing cash flow of the project.

19.10 Penalties

The terms within a conventional toll smelting agreement will include penalty clauses against the seller of the concentrate for high moisture, lower than negotiated PGE grade, potentially high chromitite content and possible other deleterious elements such as Fe, As, Bi Se, Te, MgO and SiO2.

The concentrate from Waterberg will have negligible chromitite but the other elements could, cause penalties applied for deleterious elements, but this is most unlikely.

The concentrate is expected to be a desirable product as a result of the low chromitite level and the expected high level of sulphur and base metals in the material, for blending purposes with the forecast increasing UG2 concentrate production (high chromitite content) within South Africa.

19.11 Pure Metal Sale Agreements

The metal pricing that is applied to the delivered concentrate for any particular month is to be based on the arithmetic average PGE pricing for the month of delivery of the concentrate or as negotiated with the smelter operator. Base metal pricing may be based on London Metal Exchange monthly average with discounts or premiums depending upon the end user requirements.

The study financial modelling will use the project metal price for concentrate valuation for PGE and base metals. Base metal discounts of $US200 per tonne of copper and $US100 per tonne of nickel are to be applied.

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Issues that could Materially Impact the Issuer's Ability to Extract the Mineral Resources or Mineral Reserves

The future development and delivery of the Waterberg Project will be underpinned by a sustainable development philosophy. This means effectively managing and mitigating social and environmental impacts.

The Waterberg project environmental plans and programs will be guided by international standards A deliberate and concerted effort has been made to align all design and engineering decisions with the environmental objectives and laws.

It is understood that the Waterberg Area is environmentally sensitive. The environmental management plan will have to ensure that all relevant local and Regional Stakeholders are adequately consulted during the planning Phase of the Project.

It is envisaged that the following activities will affect both the communities and environment, and plans will be considered and strategies to manage or mitigate those impacts:

- Mining activities could affect local groundwater flow due to groundwater abstraction activities, which could lower the water table, and make it more difficult for local communities to access drinking water from groundwater wells, particularly during the dry season. Most of the communities within the project area rely on hand pumped wells for domestic water supply.

- The natural landscape of the area will be significantly disrupted through the establishment of the mine. The visual and landscape impacts will be significant for the adjacent villages. The visual impacts of the underground access, plant, waste rock dumps and RDF will be significant and permanent. Because of mining activity, vegetation will be cleared, large industrial structures will be built and vehicles and earth moving equipment will become familiar in the landscape. Thus, the aesthetics of the project area will change due to the mine and associated infrastructure.

- The establishment of a mine results in vegetation being cleared in the mine path and adjacent areas for secondary infrastructure. In this instance it will result in the removal of topsoil together will all associated vegetation.

- Similarly, any watercourse/drainage lines impacted by mining operations is likely to have a permanent and irreversible impact on the pre-existing hydrological function, although it is possible that final landform rehabilitation can replicate its basic function successfully, it will be difficult to do so.

- Rural communities in South Africa place high importance on cultural heritage, including graves. Graves are an integral part of families and communities. The physical removal or relocation of graves is a sensitive impact that could cause social disorientation and psychological insecurity to communities. Relocation could also increase social tension within the household, disrupting social stability.

- The direct and indirect impacts associated with an influx of labourers and expats are likely to have significant impacts on the villages, as it usually results in many social, cultural, economic and political changes.

20.2 Requirements and Plans for Waste and Residue Disposal, Site Monitoring and Water Management both during Operations and Post-Mine Closure

Provision has been made for a waste rock dump, which will be utilised for storage of waste generated during decline and ventilation shaft development until the permanent material handling system is installed and commissioned.

Waste rock dumps will be designed to rigorous geotechnical and environmental standards.

There are a number of options for the long term management and closure rehabilitation of these facilities.

These include capping with a stable cover that minimises potential for ARD generation and supports revegetation, rehandle to a preferred, permanent waste rock storage facility or processing of waste rock through the milling circuit and disposal as part of the tailings stream. These options will be investigated in detail during the next project phase.

20.2.1 Waste Management

The EMP will include a waste management plan developed for the safe handling and storage of materials in a manner that minimizes the potential for accidents and provides for rapid mitigation of accidental releases. The waste management plan will be developed to meet all South African legislation and guidelines.

Wastes include domestic and industrial wastes, both hazardous and non-hazardous, as well as incinerator wastes.

The plan also addresses management of waste rock from mining, which will be stored on site.

The plan requires that the waste rock storage area is designed, regularly inspected and repaired as necessary such that potential environmental impacts are minimized.

The RDF area will be constructed and managed to incorporate spill protection systems, including containment. Regular inspections and maintenance will be conducted to ensure the integrity of the dam structure. RDF management includes development of an operating, maintenance and surveillance manual to define procedures for the safe operation, maintenance and surveillance of the tailings and water management throughout the facilities life cycle.

20.2.2 Water Management

The development of an Integrated Water and Waste Management Plan (IWWMP) is based on achieving compliance with the Integrated Water Use License application requirements, which requires the mine to investigate and put into practice any water saving strategies. This requirement is in terms of Section 21 (Water Use), chapter 4 of the National Water Act (Act 36 of 1998). The project, furthermore, strives to comply with the provisions and objectives of the National Environmental Management Act, 1998 (Act no. 107 of 1998) [NEMA], National Environmental Management: Waste Act, 2008 (Act no. 59 of 2008) [NEMWA] and the Minerals and Petroleum Resources Development Act, 2002 (Act no. 28 of 2002) [MPRDA, as amended].

An Environmental Impact Assessment (EIA) of all potential mining related impacts on ground and surface water resources, together with observations / findings from site inspections, the professional judgement of various project team members, stakeholder inputs and inputs from various technical staff will be used to assess where the focus of the IWMMP should be set in order to reduce the significance of those impacts deemed to be of moderate to high significance.

The IWWMP has a two-fold focus on the following key management levels, in to ensure that maximum effect is derived from the IWWMP in achieving holistic, resource focused, water and waste management:

- Surface water management techniques and specific actions to separate clean and dirty storm water flows. The main focus of which is to minimise the amount of 'dirty / contaminated' water unnecessarily generated over the mining site, whilst encouraging natural infiltration of storm water into soil surfaces and the diversion of reasonable excesses thereof towards natural surface water resources;

- Alignment of various industrial type mine water uses with already 'contaminated' sources for such, thereby reducing the overall demand on clean water resource.

Truly integrated water and waste management planning relies on the productive input from all stakeholders either directly or indirectly affected by, or with a mandate to administer, activities undertaken at project site that could impact negatively on ground and surface water quality and quantity.

The IWWMP will require updating and amendment on an annual basis to ensure that the IWWMP is treated as a dynamic / working document that remains legislatively compliant and applicable. The successful implementation of the IWWMP is underpinned by senior mine management's commitment and support for the plan and particularly support for those parties mandated to ensure its implementation. The IWWMP also prescribes self and external regulation to review compliance by the mine in implementing the specific actions of the IWWMP. Reporting of audit findings to the Department of Water and Sanitation on an annual basis is also a provision of this IWWMP.

20.2.3 Environmental Monitoring

Environmental monitoring programmes have been developed for the aspects listed below and will be implemented upon commencement of the construction phase of the project:

- Aquatic ecology — biomonitoring.

- Air quality.

- Noise.

- Surface water.

- Groundwater.

20.3 Project Permitting Requirements

Prior to construction and operation of a mine, the following local legislative authorisations would be required:

- In support of a Mining Right Application (MRA), authorisation in terms of Section 22 of the Mineral and Petroleum Resources Development Act, 2002 (Act No. 28 of 2002) (MPRD Act) by the Department of Mineral Resources (DMR) is required.

- Environmental Authorisation as per the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEM Act) and Associated Environmental Impact Assessment (EIA) Regulations (GNR. 543, 544 and 545 of 18 June 2010) from the Limpopo Department of Economic Development, Environment and Tourism (LEDET).

- A Water Use License in terms of Section 21 of the National Water Act, 1998 (Act No. 36 of 1998) from the Department of Water and Sanitation (DWS).

- A Waste Management License for categorised waste activities in terms of the National Environmental Management Waste Act, 2008 (Act No. 59 of 2008) (NEMWA) from the National Department of Environmental Affairs (DEA).

20.4 Social or Community Related Requirements and Plans for the Project

A study on social impact assessment on the community has been conducted to ensure compliance with local and international requirements including:

- This social economic assessment entails research on a variety of amongst others like household status, level of education, health sector source of income, sanitation, availability of water and electricity, etc.

- This study requires door to door interaction in the community and engaging all the local stakeholders.

This study will assist in the compilations of the social and labor plan. This study assists the company to formulate the strategy to empowering the community.

Social and labor plan (SLP) is to form integral part of the mining right in terms of the mineral an petroleum resources development act 28 of 2002 (MPRDA) SLP in essence is a Social Contract.

The Social and Labor Plan (SLP) has two components, namely: Human Resources Development (HRD) and Local Economic Development (LED). This intervention is enforced by the MPRDA

The drafting of the SLP will only come into play when the Mining right is granted and the Social contract with the community.

20.5 Status of any Negotiations or Agreements with Local Communities

PTM is engaging the community of Ketting, Early Dawn and other local communities regarding the security of Tenure. There are ongoing negotiations on either lease the farm for a period of more than thirty (30) years or outright buy of the farm.

This community has another farm called Disseldorp 369LR, which is next to Ketting 368LR farm. In the event that we agree with community, there is a plan to relocate the entire community to the next farm

PTM is also considering signing a 5 year surface lease agreement whilst these negotiations are still taking place.

The other form where PTM have signed 5 year surface lease agreement is Early Dawn 361LR farm.

20.6 Mine Closure Requirements and Costs

In terms of the MPRDA, Regulation 54, the EMP outlines the specifications for the closure and rehabilitation of the mining area after operations cease and requires that a financial guarantee be held. The quantum of pecuniary provision is updated annually and the rehabilitation cost determination is reflected in the financial guarantee required. Calculations are based on the DMR methodology with current disturbed areas surveyed and used in the guideline. The EMP states that for the first year of construction an amount of 30% is required i.e. ZAR68.2 million of the total estimated rehabilitation costs of ZAR205 million to be posted.

Under the current mine plan for the Project, mining operations at the Waterberg Joint Venture Project site are expected to continue for approximately 19 years or more. The Waterberg JV Project Rehabilitation Strategy and Implementation Plan (RSIP) will address the closing the mine and rehabilitating the mine site as well as off-site infrastructure at the cessation of operations.

The RSIP will focus on the engineering works and rehabilitative aspects of closure along with the complex set of socio-economic issues associated with mine closure and its impact on the community.

The principal goal of the closure planning process will be to ensure that the potential environmental impacts and risks of the decommissioned mine (together with their associated financial and legal liabilities) are identified early on in the process and measures are implemented to avoid or minimize these impacts and risks through the facility site, design and operations of the project. The primary objectives of the RSIP are to:

- Achieve chemical, physical and biological stability, in accordance with applicable regulations and standards, as soon as practicable.
- Protect the health and safety of the public over the long term.
- Avoid or minimize long-term environmental impacts.
- Restore disturbed land and watercourses as soon as possible and as close to pre-development conditions, as practicable.
- Minimize long term site and facility maintenance requirements.
- With input from the consultations with local communities, government agencies, and other stakeholders, develop an end-use plan, which is beneficial to the local communities and maximizes sustainable development.
- Provide a basis for estimating cost for closure.

The RSIP will continue to be developed throughout operations, incorporating new monitoring data, advancements in rehabilitation techniques / technologies and local community input regarding end uses.

21. Capital and Operating Costs

21.1 Introduction

The capital and operating costs are divided into functional costs areas described below:

21.1.1 Capital

Figure 21-1 depicts the project capital spend, where all capital spent up to the end of Dec 2022 (Month 54) is part of capital to full production and the capital spend from Jan 2023 (Month 55) is seen as sustaining capital.

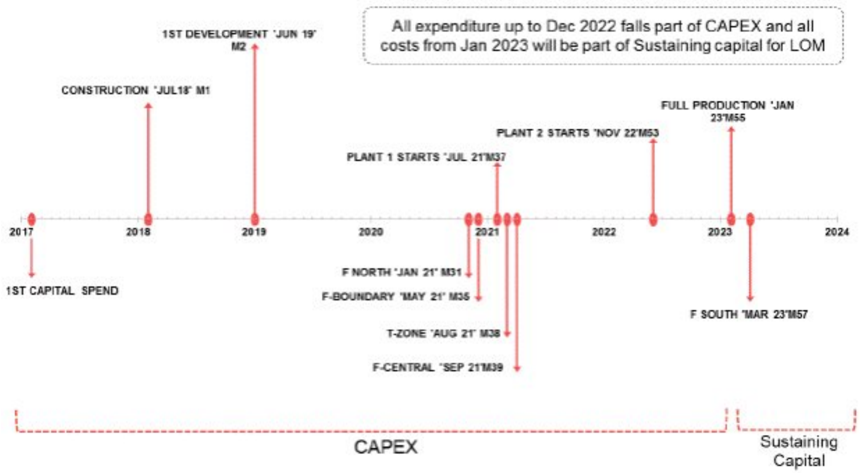


Figure 21-1: Cashflow

The capital estimate is further divided up and expressed in the following areas:

- Underground Mining
- Process
- Shared Services & Infrastructure
- Regional Infrastructure
- Site Support Services
- Project Delivery Management
- Other Capitalised Costs
- Contingency

21.1.2 Operating Costs

- Mining
- Process
- Engineering & Infrastructure
- General and Administration

Indirects including Owners Costs, contingency and closure costs are included within each area. The estimate base date has been defined as 31 July 2016. All operating costs exclude VAT.

21.2 Capital Cost Summary

This document has been prepared to support the development of the Capital Cost Estimate for the Waterberg Project.

The intended accuracy for this estimate was in the range of ± 25% (Advisian class 2 type cost estimate).The development of this cost estimate are based on the deliverables necessary to achieve this level of accuracy.

Project capital costs total ZAR 27,374M, consisting of the following:

- Initial Capital Costs — includes all costs to develop the property to a sustainable production of 600ktpm. Initial capital costs total ZAR 15,906M and are expended over a 72 month period from January 2017 to Dec 2022 including the pre-production construction and commissioning period; and

- Sustaining Capital Costs — includes all costs over the 16 year mine life related to expansion of production from the initial 300ktpm to 600ktpm and the acquisition, replacement, or major overhaul of assets required to sustain operations. Sustaining capital costs total ZAR 11,468 M and are expended in operating years from Jan 2023 to Jul 2038.

21.2.1 Extent of the Estimate

The capital cost of the Project is defined as the capital expenditure required for engineering, designing, procuring, fabricating, delivering, constructing and commissioning of the Waterberg mineral Project. This includes indirect costs which consist of Owners' team costs, insurance costs, Engineering, Procurement and Construction Management (EPCM) costs, allowances for contingency and foreign exchange content applicable to the Project.

The resultant scope of this estimate covers the total cost of civil works, bulk earthworks, mechanical work, structural steelwork, piping, electrical work, control and instrumentation, mining, reimbursable costs for professional services, owners cost and other project overhead costs. The capital cost estimate is forecast in real terms.

The Capital Cost are reported in two areas namely: Pre-Production Capital and Sustaining Capital and is presented as:

- Capital direct costs
- Capital indirect costs
- Capital costs per discipline
- Capital costs per WBS (Level 1, 2, 3 and 4)
- Capital costs per commodity group
- Capital costs per month and financial year
- Owners costs
- Contingency

The estimate is presented in such a way that it is seamlessly incorporated into the financial model as an input, expressed in a monthly cash flow as per WBS level one.

21.2.2 Key Qualifications

The following qualifications were noted when preparing the Capital Cost Estimate;

- Estimate base date is 31 July 2016
- No formal rigging lift or logistics study has been completed.
- No formal logistics study has been completed.
- The estimate reflects material take offs supplied by Advisian and other consultant Engineering teams. Where this does not exist, allowances and provisions have been included where required
- The estimate is expressed in South African Rand (ZAR). Where applicable, prices/rates obtained in other currencies have been converted to ZAR using the rates of exchange applicable to the approved exchange rates.

21.2.3 Direct Field Costs

Direct costs are the permanent facilities and services required for their installation and include plant and equipment, bulk material, contractor/sub-contractor costs, freight and vendor representatives. These items are explained further below:

- Plant and equipment include the mechanical, electrical and instrumentation components of a plant, which are either, shop assembled, modularised or pre-assembled on site
- Bulk materials are material such as rebar, piping, cables, and light steel which are purchased based on quantity
- Installation refers to the manual labour and contractor/sub-contractor costs to install or erect the plant equipment and bulk materials
- Contractor/sub-contractor costs are those costs which cover construction equipment and other support required to support and deploys installation labour. Cost components covered by these rates include:
  - temporary facilities including mobilisation and demobilisation
  - maintenance of temporary facilities and equipment
  - ownership and operation of construction equipment
  - tools and consumables
  - site office operation
  - staff and supervision
  - home office and corporate overheads
  - profit
- Freight costs are associated with the transport of plant, equipment, and material from the point of manufacture to site.
- Vendor representation is a cost associated with equipment suppliers representation on site during the installation and pre-operational testing of equipment, including mobilisation and demobilisation of the representative and any special tools

21.2.4 Work Breakdown Structure (WBS)

The estimate has been structured in accordance with the approved WBS developed by the engineers and the lead estimator, Table 21-1 summarises the level 1 to 3 facility codes.

Table 21-1: Work Breakdown Structure

Facility Code 1	Facility Description 1	Facility Code 2	Facility Description 2	Facility Code 3	Facility Description 3
2000	Underground Mining	2100	Portal (South Complex)	2110	Site Development
2000	Underground Mining	2100	Portal (South Complex)	2120	Infrastructure
2000	Underground Mining	2100	Portal (South Complex)	2130	Infrastructure - Electrical
2000	Underground Mining	2100	Portal (South Complex)	2140	Services
2000	Underground Mining	2100	Portal (South Complex)	2150	Services - Electrical
2000	Underground Mining	2100	Portal (South Complex)	2160	Ventilation & Refrigeration Plant
2000	Underground Mining	2100	Portal (South Complex)	2170	Dewatering infrastructure
2000	Underground Mining	2100	Portal (South Complex)	2180	Surface buildings
2000	Underground Mining	2100	Portal (South Complex)	2190	Workshops, Fuel Bay & Batch Plant
2000	Underground Mining	2200	Portal (Central Complex)	2210	Site Development
2000	Underground Mining	2200	Portal (Central Complex)	2220	Infrastructure
2000	Underground Mining	2200	Portal (Central Complex)	2230	Infrastructure - Electrical
2000	Underground Mining	2200	Portal (Central Complex)	2240	Services
2000	Underground Mining	2200	Portal (Central Complex)	2250	Services - Electrical
2000	Underground Mining	2200	Portal (Central Complex)	2260	Ventilation & Refrigeration Plant
2000	Underground Mining	2200	Portal (Central Complex)	2270	Dewatering infrastructure
2000	Underground Mining	2200	Portal (Central Complex)	2280	Surface buildings
2000	Underground Mining	2200	Portal (Central Complex)	2290	Workshops, Fuel Bay & Batch Plant
2000	Underground Mining	2300	Portal (North Complex)	2310	Site Development
2000	Underground Mining	2300	Portal (North Complex)	2320	Infrastructure
2000	Underground Mining	2300	Portal (North Complex)	2330	Infrastructure - Electrical
2000	Underground Mining	2300	Portal (North Complex)	2340	Services
2000	Underground Mining	2300	Portal (North Complex)	2350	Services - Electrical
2000	Underground Mining	2300	Portal (North Complex)	2360	Ventilation & Refrigeration Plant
2000	Underground Mining	2300	Portal (North Complex)	2370	Dewatering infrastructure
2000	Underground Mining	2300	Portal (North Complex)	2380	Surface buildings
2000	Underground Mining	2300	Portal (North Complex)	2390	Workshops, Fuel Bay & Batch Plant

Facility Code 1	Facility Description 1	Facility Code 2	Facility Description 2	Facility Code 3	Facility Description 3
2000	Underground Mining	2400	T-Zone	2410	UG Mine Access Infrastructure
2000	Underground Mining	2400	T Zone	2420	UG Development
2000	Underground Mining	2400	T-Zone	2430	UG Mine Mobile Equipment
2000	Underground Mining	2400	T-Zone	2440	Ventilation shaft
2000	Underground Mining	2500	F-Central	2510	UG Mine Access Infrastructure
2000	Underground Mining	2500	F-Central	2520	UG Development
2000	Underground Mining	2500	F-Central	2530	UG Mine Mobile Equipment
2000	Underground Mining	2500	F-Central	2540	Ventilation shaft
2000	Underground Mining	2600	F-Boundary	2610	UG Mine Access Infrastructure
2000	Underground Mining	2600	F-Boundary	2620	UG Development
2000	Underground Mining	2600	F-Boundary	2630	UG Mine Mobile Equipment
2000	Underground Mining	2600	F-Boundary	2640	Ventilation shaft
2000	Underground Mining	2700	F-North	2710	UG Mine Access Infrastructure
2000	Underground Mining	2700	F-North	2720	UG Development
2000	Underground Mining	2700	F-North	2730	UG Mine Mobile Equipment
2000	Underground Mining	2700	F-North	2740	Ventilation shaft
2000	Underground Mining	2900	F-South	2910	UG Mine Access Infrastructure
2000	Underground Mining	2900	F-South	2920	UG Development
2000	Underground Mining	2900	F-South	2930	UG Mine Mobile Equipment
2000	Underground Mining	2900	F-South	2940	Ventilation shaft
3000	Concentrator	3100	Plant 1 (Phase 1)	3110	ROM Handling & Crushing
3000	Concentrator	3100	Plant 1 (Phase 1)	3120	Milling
3000	Concentrator	3100	Plant 1 (Phase 1)	3130	Flotation
3000	Concentrator	3100	Plant 1 (Phase 1)	3140	Concentrate Handling
3000	Concentrator	3100	Plant 1 (Phase 1)	3150	Tailings Disposal
3000	Concentrator	3100	Plant 1 (Phase 1)	3160	Reagents
3000	Concentrator	3100	Plant 1 (Phase 1)	3170	Utilities & Services
3000	Concentrator	3100	Plant 1 (Phase 1)	3180	Plant Buildings
3000	Concentrator	3100	Plant 1 (Phase 1)	3190	Plant Wide Infrastructure
3000	Concentrator	3200	Plant 2 (Phase 2)	3210	ROM Handling & Crushing

Facility Code 1	Facility Description 1	Facility Code 2	Facility Description 2	Facility Code 3	Facility Description 3
3000	Concentrator	3200	Plant 2 (Phase 2)	3220	Milling
3000	Concentrator	3200	Plant 2 (Phase 2)	3230	Flotation
3000	Concentrator	3200	Plant 2 (Phase 2)	3240	Concentrate Handling
3000	Concentrator	3200	Plant 2 (Phase 2)	3250	Tailings Disposal
3000	Concentrator	3200	Plant 2 (Phase 2)	3260	Reagents
3000	Concentrator	3200	Plant 2 (Phase 2)	3270	Utilities & Services
3000	Concentrator	3200	Plant 2 (Phase 2)	3280	Plant Buildings
3000	Concentrator	3200	Plant 2 (Phase 2)	3290	Plant Wide Infrastructure
4000	Shared Services & Infrastructure	4100	Mine Industrial Area	4120	Site Development - Mine Industrial Area
4000	Shared Services & Infrastructure	4100	Mine Industrial Area	4140	Infrastructure
4000	Shared Services & Infrastructure	4100	Mine Industrial Area	4160	Services
4000	Shared Services & Infrastructure	4300	Mine Accommodation Facilities	4310	Main Accommodation Facility
4000	Shared Services & Infrastructure	4400	Service Corridors	4470	Site Development Portal to Ventilation Shaft
5000	Regional Infrastructure	5100	Access Roads and Corridors	5120	& Access Roads
5000	Regional Infrastructure	5100	Access Roads and Corridors	5160	Bridges
5000	Regional Infrastructure	5200	Residue Disposal Facility	5210	Disposal Facility
5000	Regional Infrastructure	5200	Residue Disposal Facility	5220	Return Water Dam
5000	Regional Infrastructure	5300	Water & Waste Water	5330	Waterberg JV Raw Water Supply
5000	Regional Infrastructure	5400	Power Supply	5410	Substation
5000	Regional Infrastructure	5400	Power Supply	5420	Main Substation
5000	Regional Infrastructure	5500	Aggregate and Gravel	5510	Site Development
6000	Site Support Services	6200	Temporary Construction Camps	6220	Infrastructure Camps
6000	Site Support Services	6400	Temporary Utilities (common)	6410	Temporary Power Supply
6000	Site Support Services	6400	Temporary Utilities (common)	6420	Labour
6000	Site Support Services	6400	Temporary Utilities (common)	6430	Utilities
6000	Site Support Services	6400	Temporary Utilities (common)	6440	Materials & Supplies
6000	Site Support Services	6400	Temporary Utilities (common)	6450	Plant
6000	Site Support Services	6700	Construction Services & Supplies	6750	Site Security

Facility Code 1	Facility Description 1	Facility Code 2	Facility Description 2	Facility Code 3	Facility Description 3
6000	Site Support Services	6900	Project Overheads	6910	Initial Fills, Spares and Inventories
7000	Project Delivery Management	7100	Professional Reimbursable Services	7110	EPCM Fees
7000	Project Delivery Management	7100	Professional Reimbursable Services	7120	Other Consultants
7000	Project Delivery Management	7100	Professional Reimbursable Services	7130	Plant
7000	Project Delivery Management	7200	Owners Management Team	7210	Owners Management Team - Home Office
7000	Project Delivery Management	7200	Owners Management Team	7210	Owners Management Team - Site Based
8000	Other Capitalised Costs	8100	Environmental Costs	8110	Closure Cost
8000	Other Capitalised Costs	8100	Drilling	8120	Drilling
8000	Other Capitalised Costs	8200	Studies	8210	Studies
8000	Other Capitalised Costs	8500	Land Acquisition	8510	Land purchases/lease
9000	Contingency	9400	Contingency	9410	Contingency

21.2.5 Exclusions from Capital Estimate

The following items were excluded from the Capital Cost Estimate;

- Removal of demolished material and excess spoil from the site.
- Foreign Exchange rate variations.
- Escalation beyond estimate base date of 31 July 2016
- Duties and taxes on imported goods and services.
- No provision for delay costs with regard to permitting (e.g. excavation permits, confined space permits etc.,), beyond what would be reasonably expected.
- No allowance has been made for delay costs associated with obtaining statutory approvals (e.g. building or development approval).
- Sunk costs.
- Influence of market forces such as concurrent projects and resource/commodity prices on labour.

21.2.6 Quantity Derivation

The majority of quantities used have been based on the engineering material take-offs (MTO's) and Basis of Design (BOD) supplied by engineers.

Quantities by commodity were developed by Advisian and other engineers, based on the scope which was reviewed during the design review in the form of drawings, sketches, equipment list and MTO's.

21.2.7 Pricing Derivation

Pricing rates for bulk materials and equipment are generally based on WorleyParsons Advisian RSA data base and recent vendor enquiries for the similar scope.

Budget quotations were obtained for the following:

- Primary and Secondary fleet
- Conveyor drives, pulleys, belting, motors
- Pumps
- Bulk water supply
- Bulk electricity supply
- Ventilation & Refrigeration
- Concentrator Plant
- RDF and Return water dam

21.2.8 Capital Summary

Table 21-2 presents the capital to full production and the sustaining capital for the Life of Mine (LOM), this includes all indirect costs from facility code 6000 to 9000

Table 21-2: Total Capital

Facility Code	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
2000	Underground Mining	6,092	9,766	406	651
3000	Concentrator	2,850	159	190	11
4000	Shared Services & Infrastructure	1,063	43	71	3
5000	Regional Infrastructure	2,566	—	171	—
6000	Site Support Services	691	67	46	4
7000	Project Delivery Management	1,399	147	93	10
8000	Other Capitalised Costs	246	83	16	6
9000	Contingency	999	1,202	67	80
Total Capital		**15,906**	**11,468**	**1,060**	**765**

21.2.9 Cashflow

The facility level summary of the capital as well as the capital expenditure for LOM is depicted in Figure 21-2.

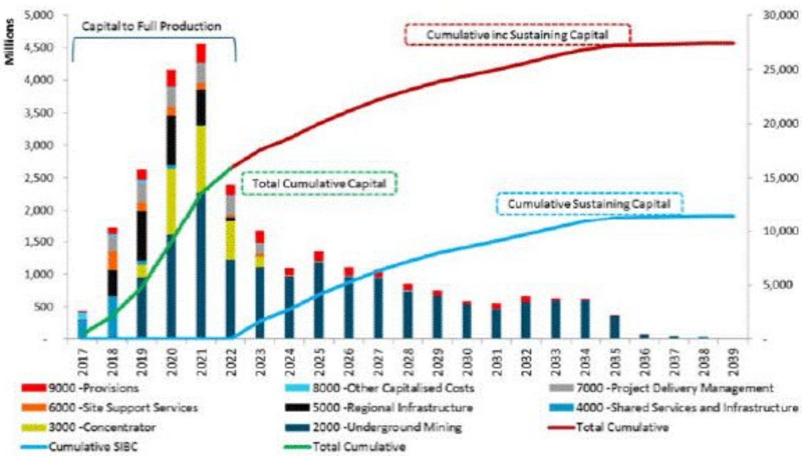


Figure 21-2: Cashflow

21.2.10 Underground Mining Capital

This section describes the basis and the development of the mining estimate, the owner's cost, construction management (EPCM) and contingency in the overall estimate and is detailed in Section 21.

21.2.10.1 Development

Underground mining is divided into the various portal locations and mining areas as presented in Section 16.

Each mining area's development is priced according to the mining schedules by applying a rate per cube for owner and contractor mining. The items mentioned below are all scheduled throughout the LOM and divided up in the following development descriptions.

- Material Decline
- Primary Conveyor Decline
- Anc Dev Main Acc Declines
- Secondary Conveyor Decline
- Anc Dev Sec Belt Decline
- Underground Workshops
- Silos and Vertical Dams
- Other Access Development
- Lateral Ventilation Development
- Ventilation Raises
- Sub-Level On-reef BLR

- Sub-Level On-reef Long SLOS
- Sub-Level On-reef Transv SLOS
- Sub-Level Off-reef BLR
- Sub-Level Off-reef Long SLOS
- Sub-Level Off-reef Transv SLOS
- Raisebore Holes

All on-reef, off-reef, other access and lateral ventilation development will be contractor mined until first stoping where it will then be owner mined and seen as OPEX cost, the stoping timeline can be seen in Figure 21-1.

Silos, Vertical Dams, Ventilation Raises and Raisebore Holes will be contractor mined throughout the LOM due to the specialized equipment needed for this type of development. The remaining items will be contractor mined up to stoping and thereafter owner mined.

All development capital is split into capital to full production and sustaining capital regardless of contractor or owner operated. Table 21-3 presents the total capital for each mining zone.

Table 21-3: Underground Mining

Facility Code	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
2000	T Zone	1,467	2,070	98	138
2000	F-Central	1,730	2,853	115	190
2000	F-Boundary	498	2,191	33	146
2000	F-North	1,377	1,761	92	117
2000	F-South	217	892	14	59
Total Capital		**5,290**	**9,766**	**353**	**651**

21.2.10.2 Fleet and Replacements

Sustaining refurbishment and replacement cost for fleet amounts to R 4,055M over LoM and consist of bucket and bowl replacement, refurbishment and replacement cost. The fleet requirements for the Waterberg project are detailed under Section 16.5 for mining and comprise of development, production, logistics and construction fleet. Engineering fleet mainly consists of logistics fleet for personnel and materials handling and transport. Fleet refurbishment and replacement costs were calculated based on when operating hours reaches the specified refurbishment or replacement interval. Fleet refurbishments and replacement philosophies as well as utilisations and availabilities to derive operating hours were provided by the engineering team. Quotes were supplied by various OEMs (Original Equipment Manufacturers).

21.2.11 Concentrator Plant Capital

The concentrator plant estimation was designed and costed by DRA and therefore should be read as a standalone project, all costs are aligned with DRA's estimation expect for the in-direct cost which is captured within the overall estimate.

The concentrator cost is all captured under facility code 3000, commissioning spares and first fills are captured under facility code 6000 whereas EPCM and contingency are absorbed in the overall project estimate

Phase 1 and 2 plants are commissioned at different times as depicted in Figure 21-1 the capital estimation for the two plants can be seen in Table 21-4.

Table 21-4: Concentrator Capital

Facility Description	Facility Code 3	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
Plant 1	3110	ROM Handling & Crushing	359	—	24	—
Plant 1	3120	Milling	274	—	18	—
Plant 1	3130	Flotation	209	—	14	—
Plant 1	3140	Concentrate Handling	165	—	11	—
Plant 1	3150	Tailings Disposal	34	—	2	—
Plant 1	3160	Reagents	49	—	3	—
Plant 1	3170	Utilities & Services	43	—	3	—
Plant 1	3180	Plant Buildings	58	—	4	—
Plant 1	3190	Plant Wide Infrastructure	486	22	32	1
Plant 2	3210	ROM Handling & Crushing	340	—	23	—
Plant 2	3220	Milling	257	—	17	—
Plant 2	3230	Flotation	208	—	14	—
Plant 2	3240	Concentrate Handling	15	—	1	—
Plant 2	3250	Tailings Disposal	34	—	2	—
Plant 2	3260	Reagents	13	—	1	—
Plant 2	3270	Utilities & Services	21	—	1	—
Plant 2	3280	Plant Buildings	15	2	1	—
Plant 2	3290	Plant Wide Infrastructure	270	135	18	9
Total Capital			**2,850**	**159**	**190**	**11**

21.2.11.1 Capital Estimate Base Date and Exchange Rate

The base date for the capital estimate is 31 July 2016 for the process plant and plant infrastructure.

The estimate has been presented in South African Rand (ZAR) and, where required, prices obtained in other currencies have been converted to ZAR using the applicable exchange rates as quoted by vendors, suppliers or contractors.

Fluctuations in the exchange rates from the date of quotation to the date of purchase have not been considered in the cost estimates.

21.2.11.2 Scope of Estimate

Capital estimates for the process plant are based on the equipment and structures described in Section 17. Also included in the estimate are permanent installations, e.g. compressed air, service water, potable water reticulation and return water columns as well as electrical supply and reticulation form the Plant Consumer substation.

The plant infrastructure includes stormwater berms and drains to divert rainwater from the plant and to collection rainwater falling in the plant in a pollution control dam, this water will be captured for use in the process and not discharged to the environment.

The estimate provides for the fencing of the plant and controlled access. Offices, a store, a workshop, and a weighbridge are included to support plant operations.

21.2.11.3 Accuracy and Basis of Estimate

The process plant estimate was determined using a combination of detailed, semi-detailed and factorised costs. The estimate has been produced using vendor quotations and in-house data and is based strictly on the equipment as described within this study.

The estimate considered the costs required to complete the design, supply, fabrication, delivery to site and construction of the earthworks, civil engineering works, structural steel, platework, mechanical equipment, piping, electrical equipment and reticulation as well as the required instrumentation and control systems. The estimate made provision for indirect costs including those to complete the engineering, procurement and construction management of the plant, cost for maintenance support vehicles, first fills of consumables and an allowance for critical spares.

The estimated costs were determined by obtaining budget prices from reputable suppliers for the mechanical equipment. Using the general arrangement drawings completed for the study, estimates of the quantities required for the major structures were compiled into Material take offs (MTO). Quantities of minor structures were estimated by factorization of similar designs.

Material take offs (MTO) were completed for the structural steel, platework, electrical and civil engineering disciplines. Costs for the fabrication and erection of structural steel and platework, electrical equipment as well as the construction of the civil engineering works were estimated by applying rates from DRA’s database for South African work to these quantities.

The costs for in-plant piping, engineering costs and instrumentation were determined by factorization.

Parametric estimating techniques were applied to the estimates for infrastructure and tailing pipe lines, the brick buildings being estimated on a rate per square meter and the overland piping on a rate per meter of piping.

Preliminary and General Costs for site establishment, on going site management and supervision, various items of plant, transport and accommodation of labour and costs for HR/IR functions was provided for the main contractors.

Allowances have been made for the first fills of process grinding media and reagents and for consumables based on DRA estimates. A provision was made for commissioning assistance by the equipment suppliers. Spare parts costs for commissioning and strategic/critical spares have been included in the capex estimate based on a factorised estimate of equipment estimates.

A quantitative risk assessment of the accuracy of the estimate was conducted and a Design Development Allowance of 13.7% was allowed to provide for the accuracy level of the rates and quantities used in the estimate.

The estimates for the scope of work, within the given battery limits, and subject to the qualifications, assumptions and exclusions contained in this report, are considered to be within the accuracy range required for a Pre-Feasibility Study of ±25%.

21.2.11.4 Estimating Assumptions

In preparing the Processing Plant Capital Estimates the following assumptions were applied:

- The project is to be executed using an engineering, procurement and construction management (EPCM) project execution strategy.
- The 600ktpm plant will be constructed in two phases, each phase completing a plant module capable of processing 300ktpm.
- The construction activities of each phase will be carried out in a continuous program.
- The limited geotechnical information for the site is indicative of the whole plant site.
- That commercial source of earthworks structural backfill material is freely available within 80km of the site.
- Concrete batch plant will be established at site and adequate aggregate will be available within 80km from the site.
- Bulk materials such as rebar, structural steel and plate, electric cable and piping are all readily available in the scheduled timeframe
- Concrete construction assumes any exposed surfaces are wood floated and vertical concrete faces are done with smooth formwork.
- Capital equipment is available in the timeframes scheduled since availability has been verified with suppliers.
- Construction work pricing based on unit price rates.
- The supplied non-binding budgetary quotes for major equipment and materials are within the required accuracies.
- That adequate supplies of water and electricity can be supplied by others.
- The estimate of the plant and infrastructure costs are stated exclusive of all taxes, royalties, duties and levies which may be imposed resulting from the purchase and transportation of the materials and use of services; including, but not limited to customs duties, permitting costs and value added tax.

21.2.11.5 Battery Limits

The capital estimate is for the process plant and infrastructure inside the following battery limits:

- ROM Material is received from the underside of the conveyor head chute located at the top of the ROM Silos within the plant boundary.
- Electricity is received as an 11kV supply at the incomer of the Plant MV Consumer Substation.
- Raw Water is provided by other to the top of the raw water tank.
- Potable Water is provided by others received at the top of the potable water tank.

- Sewerage is discharged at the plant fence for treatment by others.
- Plant Tailings are pumped to the fence/boundary of the Tailing Storage Facility for deposition by others.
- Return Water is received at the suction of the Return Water Pumps at the Return Water Dam.
- Earthworks within the indicated plant boundary.
- In-plant road construction to the plant boundary fence to join roads built by others.

21.2.11.6 Exclusions from Concentrator Costs

The following costs are excluded from the Process Plant capital estimate or provided for by others:

- All royalties, commissions, lease payments, rentals and other payments to landowners, title holders, mineral rights holders, surface right holders, and / or any other third parties.
- All taxes, royalties, duties and levies which may be imposed including, but not limited to customs duties/ import duties, surcharges, permitting costs, value added tax as well as any other statutory taxation, levies or government duties.
- Escalation.
- Costs resulting from scope changes.
- Costs resulting from labour disputes.
- Costs resulting from community engagement process.
- Environmental permitting activities.
- Cost of financing.
- Interest on capital loans.
- Any owner's team and/or pre-production costs not specified in the pre-production section of the estimate.
- Sunk costs.
- Any costs to be expended prior to Board approval for project implementation including additional environmental and feasibility studies prior to project implementation.
- Forward cover for any foreign content.
- All operating costs.
- Any work outside the defined battery limits.
- Any provision for project risks outside of those related to design and estimating confidence levels.
- Acquisition cost for mineral rights and the purchase or use of land.
- Project insurances.

21.2.12 General Infrastructure

This section covers the shared and regional infrastructure for the Waterberg Project inclusive of bulk power, water supply and access roads, but excludes the specific concentrator infrastructure covered above.

The estimate for site infrastructure was compiled based on GA drawings and layouts. Where possible quantities were measured from these drawings and priced based on recent contracts for similar work.

The capital for replacement of fixed engineering equipment is calculated based on engineering replacement intervals and costs together with when equipment reaches its operating life.

21.2.12.1 Civil Works

21.2.12.1.1 Bulk Earthworks, Roads and Terraces

Bulk earthworks quantities were based on preliminary drawings for terraces. Bulk earthworks rates were based on contract rates obtained from a contract that's currently in execution and compared to rates recently obtained for other similar work.

21.2.12.1.2 Sources of Fill Material

Fill material for G6 and wearing courses would be obtained from commercial sources.

21.2.12.1.3 Overhaul

The rates for excavations included a free-haul distance of 2 km. Overhaul for fill material was based on between two and four kilometers depending on the location of the work. This was to allow possible haulage of fill material from the waste dumps

21.2.12.1.4 Hard Rock

Provisions for blasting of hard rock were made depending on the location of the respective structures and available geotechnical information.

21.2.12.1.5 Concrete Work

Concrete work rates were based on current contracts rates for similar work. Allowances based on square meters were used to calculate concrete quantities as no detail design drawings are currently available.

21.2.12.1.6 Main Buildings & Smaller Buildings

The footprint areas of main buildings were generally measured off preliminary drawings. A rate per square meter was used based on previous projects for similar work.

Preliminaries and General were based on the assumption that the contractor will supply and install all materials including steelwork, except for mechanical equipment.

Building works were priced based on the rates and prices obtained from recent contracts of similar work.

Smaller buildings were also estimated on the same principle, however, preliminary layout drawings were not available for some of the buildings.

21.2.12.1.7 Structural steelwork

Rates for structural steelwork were based on recent quotes for similar work obtained by the engineer.

21.2.12.1.8 Security and Fencing

An allowance of R12M was included for security infrastructure as requested by the client. Rates used for fencing were based on rates obtained from current contract rates, and additional capitalised OPEX cost for security was included in the capital.

21.2.12.1.9 Potable Water

Rates for the potable water treatment plant and piping were based on recent quotes obtained by the engineer.

21.2.12.1.10 Sewerage

Sewer water reticulation quantities were based on preliminary layouts. The treatment plant and piping rates were based on recent quotes obtained by the engineer.

21.2.12.1.11 Residue Disposal Facility

A detailed report for the design and costing of the RDF and return water dam is captured in Section 18.11

21.2.12.1.12 Preliminaries and General

Preliminaries and General used in the estimate were based on the rates obtained for the current execution contracts of similar work. P&Gs were determined by applying various percentages for the various disciplines

One factor must be noted and carried through during the execution of this project. The P&G for work executed on site was based on the mine providing accommodation at rates below market rates at the existing camps already established on the site.

21.2.12.1.13 Infrastructure Capital

The total capital for the infrastructures can be seen in Table 21-5, this includes the infrastructure for the portals in the various complexes.

Table 21-5: Infrastructure Capital

Facility Description	Facility code 3	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
Underground Mining	2100	Portal (South Complex)	224	—	15	—
Underground Mining	2200	Portal (Central Complex)	334	—	22	—
Underground Mining	2300	Portal (North Complex)	244	—	16	—
Shared Services & Infrastructure	4100	Mine Industrial Area	1,041	43	69	3
Shared Services and Infrastructure	4400	Service Corridors	21	—	1	—
Regional Infrastructure	5100	Access Roads and Corridors	556	—	37	—
Regional Infrastructure	5200	Residue Disposal Facility	568	—	38	—
Regional Infrastructure	5300	Water and Waste Water	240	—	16	—
Regional Infrastructure	5400	Power Supply	1,201	—	80	—
Regional Infrastructure	5500	Aggregate and Gravel	1	—	—	—
Total Capital			**4,432**	**43**	**295**	**3**

21.2.13 Indirect

Indirect costs are the costs associated to support the purchase and installation of the direct costs. This includes the materials and services required for field construction, but are not incorporated into or accounted for as part of the permanent facilities. A standard set of indirect costs with detailed descriptions is calculated in the estimate. Table 21-6 reflects all the indirect cost for the Waterberg project.

Table 21-6: In-directs

Facility Description	Facility code 2	Facility Description	To Full Production ZAR (M)	Sustaining Capital ZAR (M)	To Full Production USD (M)	Sustaining Capital USD (M)
Site Support Services	6200	Temporary Construction Camps	225	—	15	—
Site Support Services	6400	Temporary Utilities (common)	242	49	16	3
Site Support Services	6700	Construction Services & Supplies	54	—	4	—
Site Support Services	6900	Project Overheads	169	18	11	1
Project Delivery Management	7100	Professional Reimbursable Services	1,147	121	76	8
Project Delivery Management	7200	Owners Management Team	252	27	17	2
Other Capitalised Costs	8100	Drilling	60	—	4	—
Other Capitalised Costs	8100	Environmental Costs	99	83	7	6
Other Capitalised Costs	8200	Studies	52	—	3	—
Other Capitalised Costs	8500	Land Acquisition	35	—	2	—
Total Capital			**2,336**	**298**	**156**	**20**

21.2.13.1 Site Support Services

Indirect costs for this capital estimate are both quantified and factorised. The factors used are based on previous Greenfield studies with similar aspects. Site support services are inclusive of temporary construction camps, all labour, security, utilities, supplies, and power to operate the site during the construction phase as well as for plant commissioning and spares.

21.2.13.2 Project Delivery Management

The EPCM, consultants and owners cost budgets were factorised based on previous Greenfield projects of the same magnitude. The project delivery management covers all the EPCM costs as well as for specialised consultants furthermore, owners costs is divided in home office and site based.

21.2.13.3 Other Capitalised Costs

These capitalised costs include drilling, environmental closure cost, future studies, and relocation of community and land purchases.

21.2.13.4 Contingency

The underlying rationale supporting development of the contingency amount is based on the following overarching principles:

- Captures all risk and uncertainty arising from:
  - Design quality and accuracy
  - Estimation (quantities) quality and accuracy
  - Ground conditions (underground development and surface earthworks, excluding market-driven price and rates risk (i.e., real escalation in labour rates arising from a hot market; real increases in steel, copper, energy, etc., prices; unit price-based changes to equipment supply, etc.)
- Excludes foreign exchange

21.2.13.5 Contingency Assessment

Assessments were conducted wherein the project team was involved. Risks applicable to the project were identified and incorporated into the designs where possible, in order to eliminate or mitigate the risk. Where specific risks were identified but could not necessarily be quantified, these were catered for in the contingency determination criteria as unknown risks.

The contingency in the capital model was assessed by conducting a 10 000 iteration Monte Carlo analysis. With Monte Carlo analyses, it is important to define the weights or frequencies for each point in the distribution. However, there is uncertainty regarding the various ranges assumed; therefore, weightings were defined for the upper and lower ranges as described below.

A detailed contingency assessment was conducted by dividing the estimate into logical cost elements, variables that affect these elements were identified and an appropriate risk factor distribution were determined. A cost influence matrix was developed from this and applied to the various cost elements to determine the contingency. There is no contingency applied on mining costs when it moves over to owner operated as well as for refurbishments and replacement cost.

In the simulation results show there is a 90% likelihood that the estimate will be between (R18,475M) and (R23,113M), the (R18,475M) refers to the "best case" whereas (R23,113M) refers to the "worst case", by using 50% from cumulative probability distribution it gives us the "most likely cost" of (R20,822M) which calculates to a 12% contingency.

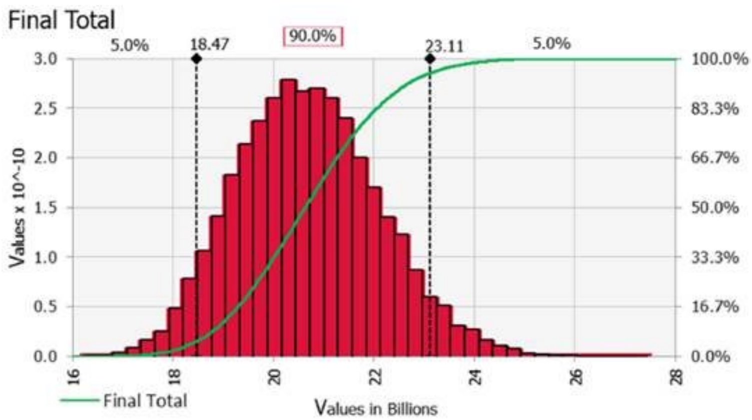


Figure 21-3: Contingency Analysis

21.3 Operating Cost Summary

For the study, OPEX has been defined as:

- All on-reef development as soon as first stoping tonnes are achieved,
- Off-reef development associated with ongoing access and Reserve generation within, when first stoping tonnes are achieved. (These include sub-level off reef, lateral ventilation and other access development),
- All ongoing production related activities after first stoping ore is mined,
- Operating costs associated with the mobile mining equipment and fixed engineering equipment,
- Maintenance of mobile mining equipment and fixed engineering equipment.
- Initially the mine will be contractor operated and once first stoping ore is mined for a particular mining zone, it will become owner operated. This excludes some contracted services over LoM such as raise bore, ventilation raises, silo and vertical dams, main access, primary conveyor decline and material decline development. The TSF facility will also be contracted out. The owner mined operation per zone will coincide with when operating costs starts being incurred. All costs not associated to a particular mining zone will be reported under shared services and will include general and administration and processing cost.

The start dates for Opex and the owner mined operation is displayed in Table 21-7 below.

Table 21-7: Mining Zone and Financial Year Start Dates

Mining Zone	Financial Year Start Date	Month No	Opex Start Date
T-Zone	1 January 2021	Month 38	1 August 2021
F-South	1 January 2023	Month 59	1 May 2023
F-Central	1 January 2021	Month 39	1 September 2021
F-Boundary	1 January 2021	Month 36	1 June 2021
F-North	1 January 2021	Month 34	1 April 2021
Shared Services	1 January 2021	Month 36	1 June 2021

The operating cost model was developed by following the typical steps and processes prescribed by the WorleyParsons Advisian RSA OPEX Estimation standards and methodologies. Methodologies utilised includes first principle costing for the labour, lifecycle costing for all equipment, infrastructure and fleet, zero-based costing for mining consumables and fixed/variable costing for the remainder of operating cost items. The estimate methodology is aligned to preliminary engineering designs and budgetary quotations for major equipment and consumable cost and conforms to the +-25% accuracy level of a Pre-Feasibility Study.

The operating cost estimate is modelled annually in ZAR. Costs reported in USD was converted from ZAR by using and exchange rate of R 15 per USD. A base date of July 2016 was used as costing basis. Costs are reported in real money terms with no escalations or contingency modelled. Quotes and cost rates were sourced from South African suppliers with foreign component cost not having an impact on the operating costs estimate.

The average LoM operating cost for the Waterberg Pre-Feasibility Study project is estimated at R 574.62 per ore tonnes broken (USD 38.31 /t). As indicated in Table 21-8, the total LoM cost amounts to R 58,99 billion (USD 3,93 billion). Average LoM costs are also detailed on a high level per area in ZAR and USD.

Table 21-8: Average LoM Operating Cost Rates and Totals per Area in ZAR and USD

	Average LOM (ZAR/t)		Total LOM (ZAR M)		Average LOM (USD/t)		Total LOM (USD)	
Mining	R	271.90	R	27 915	$	18.13	$	1 861
Engineering & Infrastructure	R	107.49	R	11 036	$	7.17	$	736
General & Admin	R	40.71	R	4 180	$	2.71	$	279
Process	R	154.52	R	15 864	$	10.30	$	1 058
Total OPEX Cost	**R**	**574.62**	**R**	**58 994**	**$**	**38.31**	**$**	**3 933**

It is to be noted that the process cost quoted in this section is higher to that quoted in Section 18. This is due to additional operating costs included for an assay lab and stockpiling mobile equipment (fleet) together with their associated labour. The RDF operating cost is also reported under the process area which elevates the total quoted process cost rate.

The information in the table above is visually represented in Figure 21-2 to provide a better understanding of the breakdown per area of the LoM operating cost.

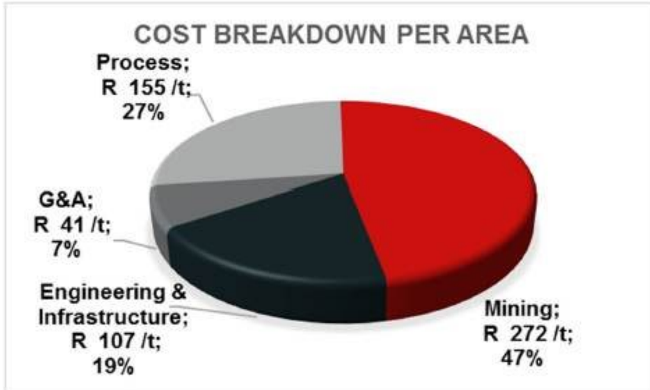


Figure 21-4: LoM Average R/t Operating Cost Breakdown per Area

From the figure it is evident that mining comprise the bulk of the operating cost at 47%, followed by process at 27% and engineering and infrastructure at 19%. General and administration cost contributes a small portion (7%) of the total operating cost.

Figure 21-5 presents the total operating costs over LoM overlaid with the ore tonnage profile. The cost increase observed in 2022 is due to the start of the second process plant in November 2022 (month 53) combined with an increase in tonnage. Steady state is observed around 2024 when the process plant will process 7,2 Mtpa. The process, general and administration and engineering and infrastructure operating cost remain fairly constant throughout the LoM, whilst the mining operating cost closely resembles the tonnage profile. The two phased ramp down starting in year 2035 is clearly visible towards the end of LoM. The dip in operating cost displayed in year 2036 is a result of only one process plant being operational to process 200 ktpm for a duration of approximately 17 months, until ore tonnes are depleted.

Figure 21-5: Operating Cost per Zone over LoM relative to Ore Tonnes

The operating cost model was developed to enable reporting per zone (e.g. F South), per area (e.g. mining) and per cost category (e.g. labour).

21.3.1 Operating Cost per Mining Zone and Area

Table 21-9 presents the total operating cost per zone and per area, of which shared services and processing comprises the bulk at 37%. This is greatly attributed to process operating cost of around R 15,8 billion.

Table 21-9: Summary of Total LoM OPEX Cost per Mining Zone and Area

	T Zone		F South		F Central		F Boundary		F North		Processing & Shared Services		Total	
	Total LOM (ZAR M)		Total LOM (ZAR M)		Total LOM (ZAR M)		Total LOM (ZAR M)		Total LOM (ZAR M)		Total LOM (ZAR M)		Total LOM (ZAR M)	
Mining	R	5 283	R	3 718	R	8 573	R	4 736	R	5 551	R	54	R	27 915
Engineering & Infrastructure	R	2 015	R	105	R	2 658	R	1 625	R	1 861	R	2 772	R	11 036
General & Administration	R	149	R	118	R	192	R	122	R	159	R	3 440	R	4 180
Process	R	21	R	0	R	0	R	0	R	20	R	15 823	R	15 864
Total OPEX Cost	**R**	**7 469**	**R**	**3 940**	**R**	**11 423**	**R**	**6 483**	**R**	**7 590**	**R**	**22 089**	**R**	**58 994**

21.3.2 Operating Cost per Cost Category

Various cost categories used to further detail the operating cost include: materials and supplies, labour, utilities, fixed overheads and external services.

Figure 21-6 provides an overview of the cost breakdown per cost category for the total LoM average operating cost.

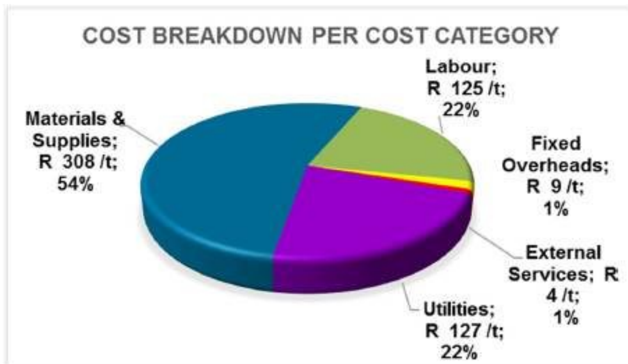


Figure 21-6: LoM Averate R/t Operating Cost Breakdown per Cost Category

Materials and supplies comprise 54% of the total cost followed by utilities and labour at 22% each.

21.3.2.1.1 Materials and Supplies

Materials and supplies comprise of operating consumables, maintenance consumables and spares and can be detailed as follows:

- Mining consumables
  - Explosives
  - Drilling
  - Support
  - Equipping
- Process consumables and spares
  - Grinding media
  - Reagents
  - Sampling
  - Crushing and mill liners
  - Maintenance consumables
- Fleet (mobile equipment) consumables and maintenance
  - Fuel
  - Lubrication
  - Tyres
  - Hydraulic hoses
  - Trailing cable
  - Maintenance consumables
- Engineering and infrastructure maintenance consumables and spares
- General and administration consumables such as stationery, printing and general office consumables

The total LoM materials and supplies cost for the Waterberg Project amounted to R 31,6 billion. The breakdown of the total operating cost per area is provided in Table 21-10 below.

Table 21-10: Total LoM Materials and Supplies Cost Breakdown per Area

Material and Supplies Cost per Area	Total LOM (ZAR Million)	
Mining	R	19 699
Engineering & Infrastructure	R	3 944
General & Administration	R	60
Process	R	7 943
Total OPEX Cost	**R**	**31 646**

21.3.2.1.2 Utilities

The cost of utilities comprise 22% of the total LoM operating cost at R 13,07 billion. From Table 21-11 it is evident that approximately 50% of the power cost can be attributed to process.

Table 21-11: Total LoM Utilities Operating Cost Breakdown per Area

Material and Supplies Cost per Area	Total LOM (ZAR Million)	
Mining	R	3 083
Engineering & Infrastructure	R	3 435
Process	R	6 556
Total OPEX Cost	**R**	**13 075**

Water consumption and cost rates for mining and process have been estimated by the relevant engineers and is detailed under Sections 21.3.5.1.2 and 21.3.4.3.3. The estimate was based on bulk water consumption for supply cost, sewage and potable water treatment. The total water cost over LoM amounts to R 84,5 million.

The power cost consist of fixed and variable portions. The variable power cost is derived from estimated consumptions for mobile equipment (development and production fleet), fixed equipment, engineering and mining infrastructure per zone, process plant and general and administration, as well as regional infrastructure power. Load lists defining absorbed power together with power profiles over LoM is utilised to determine power consumption. The fixed power cost portion is comprised of a services and admin fee, together with fixed cost charges based on calculated mWh, kVA and KVArh. Fixed power costs for the mine (excluding process) reflect under engineering and infrastructure in Section 21.3.5.1.2 together with variable portions of power consumed associated to engineering and infrastructure. The total power cost over LoM amounts to R 12,99 billion. Power averaged at R 0.77 per kWh (total power cost excluding process).

Power cost rates used were based on the 2016/2017 Eskom Megaflex tariffs for non-local authority for a transmission zone between 300 km to and including 600 km and a voltage range between and including 66 kV and 132 kV. Table 21-12 details the Eskom power tariffs used.

Table 21-12: Eskom Megaflex Tariffs for Non-local Authority (2016/2017)

Rate Type	Description	Unit	Amount (Real Cost Rates)	
Fixed rates	Service charge	R/day	R	3 483.16
	Admin charge	R/day	R	111.24
	Total:	R/day	R	3 594.40
	Total:	R/Month	R	109 329.67
Variable rates	Distribution Network demand charge	R/kVA/month	R	9.39
	Distribution Network access charge	R/kVA/month	R	5.07
	Transmission network charge	R/kVA/month	R	6.99
	Urban Low voltage charge	R/kVA/month	R	12.50
	Electrification and Rural Network Subsidy	c/kWh	R	6.93
	Affordability Subsidy Charge	c/kWh	R	2.65

Rate Type	Description	Unit	Amount (Real Cost Rates)	
	Reactive Energy Charge - high season	c/kVArh	R	12.52
	Ancillary Service Charge	c/kWh	R	0.33
	Average active energy charge	R/kWh	R	0.616

21.3.2.1.3 Labour

Labour cost also constitutes close to 22% of the total operating cost at R 12,9 billion over LoM. From the numbers presented in Table 21-13, it can be deduced that mining labour makes up 40% of the total labour cost.

Table 21-13: Total LoM Labour Operating Cost Breakdown per Area

Labour Cost per Area	Total LOM (ZAR Million)	
Mining	R	5 133
Engineering & Infrastructure	R	3 657
General & Admin	R	2 872
Process	R	1 219
Total OPEX Cost	**R**	**12 880**

Labour rates were associated to Patterson grades to derive labour costing for the labour complement specified for the Waterberg Pre-Feasibility Study. The labour model was modelled from first principles together with the project team. Labour rates were based on benchmarked, total cost to company packages and were inclusive of housing allowance, overtime pay and a 13th cheque for high bands. No provision for production bonuses were made.

Underground mining and engineering shift cycles comprised of two 10.5 hour shifts daily, 243 days per year, with three labour streams operating on an eight shifts on, four shifts off roster. Working hours for this particular shift cycle averages around 49 hours per week (45 hours normal time and three hours overtime per week, which is within the limits stated in the basic conditions of employment act). Process plant shifts entailed four eight hour shifts daily, whilst majority of general and administration personnel is envisaged to wort regular office shifts of eight hours.

An allowance for unavailables (due to leave, absenteeism and training) was made for core personnel. For B grade operators, a fifteen percent allowance was made whilst ten percent was allowed for all A grade and remaining B grade personnel (non-operators and non-critical positions).

Labour cost was introduced three month prior to inception of the owner operated mining to allow for training, induction and medicals.

Figure 21-7 displays the labour complement per area over LoM relative to production. The maximum labour complement of 3361 can be observed in year 2025.

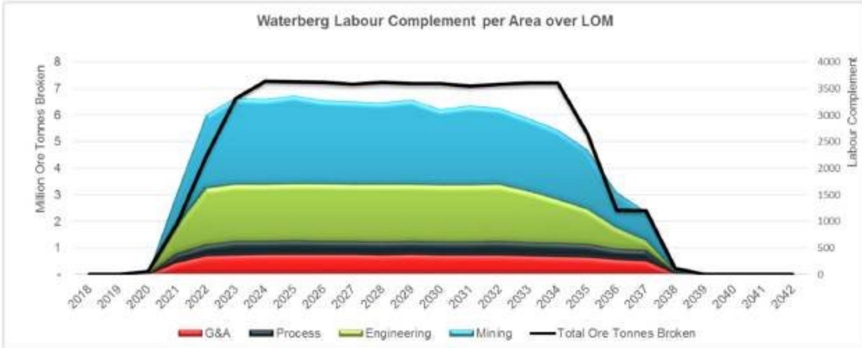


Figure 21-7: LoM Labour Complement per Area relative to Ore Tonnes

21.3.2.1.4 Fixed Overheads

The fixed overhead cost amounts to R 959 million in total, 1.6 % of the total LoM operating cost. It is made up of insurance and an allowance for owner team cost and severance pay which has been included as part of overheads.

The insurance cost is based on current insurance coverage for operations managed by Platinum Group Metals (RSA) (Pty) Ltd. Insurance cost were scaled to the Waterberg Project and indicative premium rates were obtained from the current PTM insurance brokers. Insurances coverage included in the operating cost estimate amounts to R 250 million over LoM and entails:

- General and umbrella liability,
- Property including machinery breakdown,
- SASRIA on all the above where applicable.

The annual allowance for owner team, IT and security was based on one percent of the total operating cost and totals R 581 million over LoM.

For the end of LoM, a provision for retrenchment packages were included as part of the operating cost estimate. The calculation was based on one weeks' severance pay for each year worked, assuming a four year employee turnover period. The cost amounts to approximately R 125 million in total.

21.3.2.1.5 External Services

The total external services cost over LoM amounts to R 435 million. Due to the mine being owner operated, very few services impacting operating cost will be contracted out, hence external services contributing only 1% to the total LoM operating cost. External services made provision for in the estimate include the RDF operation and management, contracted security services and waste removal. Security services has been estimated at R 14.4 million per year. (A security management structure is costed as part of the labour estimate.) Waste removal was calculated by estimating the frequency of trips required to remove domestic, industrial and medical waste from the site together with cost rates based on travel distance, waste disposal and service fee estimates.

An annual RDF operation and management cost for plant 1 has been estimated at R 6,31 million and R 7,86 million for plant 2. In addition, a general works cost of R 300 million and consulting services of R400 million has been specified annually. RDF operating cost has been estimated and compiled by Epoch.

21.3.3 Mining Operating Costs

Mining operating cost amounts to R 27,9 billion (R 272 /t) comprising the bulk of the total operating cost at 47% as displayed in Figure 21-4. Table 21-14 provides a breakdown of the mining cost per cost category.

Table 21-14: Total LoM Mining Operating Cost Breakdown per Cost Category

Mining cost per Cost Category		Total LOM (ZAR Million)		Average LOM (ZAR/t)
Materials & Supplies	R	19 699	R	191.87
Labour	R	5 133	R	49.99
Utilities	R	3 083	R	30.03
Total OPEX Cost	**R**	**27 915**	**R**	**271.90**

Materials and supplies together with labour will be elaborated on in the subsections below.

Utilities for mining (comprising solely of electricity) which has been estimated for fixed equipment and infrastructure associated to mining, is reported under the mining G&A section as power is shared amongst the different sub areas. Smaller power costs reported under each of the sub areas comprise of power associated to mobile equipment (fleet) and is calculated based on their associated power pack hours and power pack rating.

Mining operating cost can be further detailed into development, production, logistics, construction and G&A. Figure 21-8 below provides a visualization of the contribution of each of these categories.

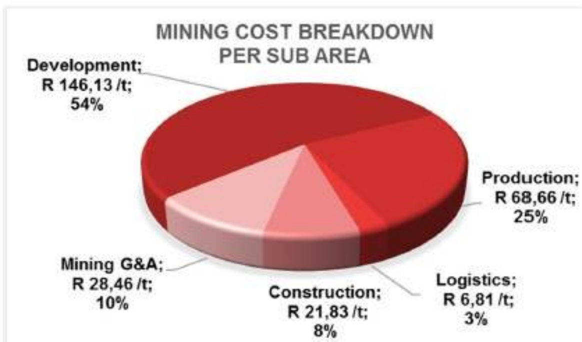


Figure 21-8: LoM Average R/t Mining Operating Cost Breakdown per Sub Area

Development constitutes 54% of the total mining cost, followed by production at 25%. provides in depth information detailing LoM totals and average ZAR per ore tonnes broken cost, per sub area, per cost category and sub category. Costs area also expressed as a percentage of total mining operating cost in the last column.

Table 21-15: Mining Cost Detail per Sub Area and Cost Category

Sub Area	Cost Category	Sub Cost Category	Total LOM (ZAR M)		Average LOM (ZAR/t)		% of Total Mining Cost
Construction	Labour	Direct labour	R	1 054	R	10.26	3,8%
	Materials & Supplies	Consumables	R	1 187	R	11.57	4,3%
Development	Labour	Direct labour	R	2 361	R	22.99	8,5%
	Materials & Supplies	Consumables	R	6 655	R	64.82	23,8%
		Drilling	R	105	R	1.02	0,4%
		Explosives	R	491	R	4.78	1,8%
		Equipping	R	3 244	R	31.60	11,6%
		Support	R	2 021	R	19.69	7,2%
	Utilities	Power	R	126	R	1.23	0,5%
Logistics	Materials & Supplies	Consumables	R	699	R	6.81	2,5%
Mining G&A	Utilities	Power	R	2 922	R	28.46	10,5%
Production	Labour	Direct labour	R	1 718	R	16.73	6,2%
	Materials & Supplies	Consumables	R	4 341	R	42.28	15,6%
		Drilling	R	258	R	2.52	0,9%
		Explosives	R	696	R	6.78	2,5%
	Utilities	Power	R	35	R	0.35	0,1%
Total			**R**	**27 915**	**R**	**271.90**	**100%**

21.3.3.1 Development

Development can be broken into material and supplies, labour and utilities. Materials and supplies comprise the majority of the development cost at R 121.91 per ore tonnes broken of which fleet consumables are R 64.82 per ore tonnes broken. Costing for mining material and supplies are derived from zero based costing by combing relevant tonnage or cube drivers with rates for drilling, explosives, support and equipping. Mining rates used for development are listed in Table 21-16.

Table 21-16: Mining Development Cost Rates

Development Rates

Rate Type	Rate	Unit
Explosive rate	7.64	R/t
Drilling rate	1.63	R/t
Support rate for BLR mining method	83.01	R/m3
Support rate for Long SLOS mining method	79.87	R/m3

Development Rates

Rate Type	Rate	Unit
Support rate for Transverse SLOS mining method	81.67	R/m3
Support rate for decline and other	105.96	R/m3
Temporary development equipping rate	142.31	R/m3

Operating and maintenance consumables for primary and secondary development fleet such as fuel, lubrication, tyres, trailing cable, hydraulic hose, GET (ground engagement tools) and maintenance consumables and spare parts are reported under consumables. Fleet operating costs are derived through lifecycle costing methodologies aided by OEM (original equipment manufacturer) operating metrics and costing together with utilisations and availabilities based on cycle times calculated by the project mining engineer.

The labour cost rate for mining development ranks the highest when comparing labour costs for mining sub areas, averaging at R 22.99 per ore tonnes broken over LoM. An average development labour complement of 545 is required with 750 personnel at peak (year 2023) development.

21.3.3.2 Production

Similarly to development, production can be broken into material and supplies, labour and utilities. Materials and supplies comprise the majority of the production cost at R 51.58 per ore tonnes broken of which fleet consumables are R 42.28 per ore tonnes broken. Costing for mining material and supplies are derived from zero based costing by combing relevant tonnage or cube drivers with rates for drilling and explosives. Production rates used for costing stoping are listed in Table 21-17.

Table 21-17: Mining Product Cost Rates

Production Rates

Rate Type	Rate	Unit
Explosive rate for BLR mining method	10.37	R/t
Explosive rate for Long SLOS mining method	8.32	R/t
Explosive rate for Transverse SLOS mining method	11.01	R/t
Drilling rate for BLR mining method	3.28	R/t
Drilling rate for Long SLOS mining method	3.74	R/t
Drilling rate for Transverse SLOS mining method	3.83	R/t

Operating and maintenance consumables for primary and secondary production fleet are reported under consumables. Operating costs for production fleet are also derived through lifecycle costing methodologies aided by supplier operating metrics and costing together with utilisations and availabilities based on cycle times calculated by the project mining engineer.

Mining production labour cost averages at R 16.73 per ore tonnes broken over LoM. An average production labour complement of 382 is required with 567 personnel at peak (year 2025) production (stoping).

21.3.3.3 Construction

Construction is a support service to development and can be broken into material and supplies and labour. For construction, materials and supplies only entail consumables for underground construction support fleet (scissors lift) and amounts to R 11.57 per ore tonnes broken as specified in Table 21-19.

Mining construction labour cost averages at R 10.26 per ore tonnes broken over LoM. An average construction labour complement of 320 is required with 434 personnel at peak (year 2023) development.

21.3.3.4 Logistics

Similar to construction, logistics is also a support service and consists off consumables for underground logistic support fleet (Light diesel vehicles) at a cost of R 6, 78 / tonne.

21.3.4 Plant Operating Cost Estimates

21.3.4.1 Basis of Plant Operating Cost Estimate

This operating cost estimate is applicable to the steady state operation of a single 300 ktpm module during Phase 1, and two 300ktpm modules for Phase 2, with production start dates as outlined in Section 17.1.

This estimated is supported by the test work (as outlined in Section 13) and engineering input (as per Section 17) and is expected to have an accuracy of ±15 to ±25%. The plant operating costs have been based on costs from June 2016, and were calculated in ZAR.

The process plant life-of-mine operating cost has been calculated as R141.40/t milled.

Refer to Figure 21-2 for a summary of the process plant operating cost average over the life-of-mine.

Refer to Figure 21-9 for a summary of the process plant operating cost average over the life-of-mine.

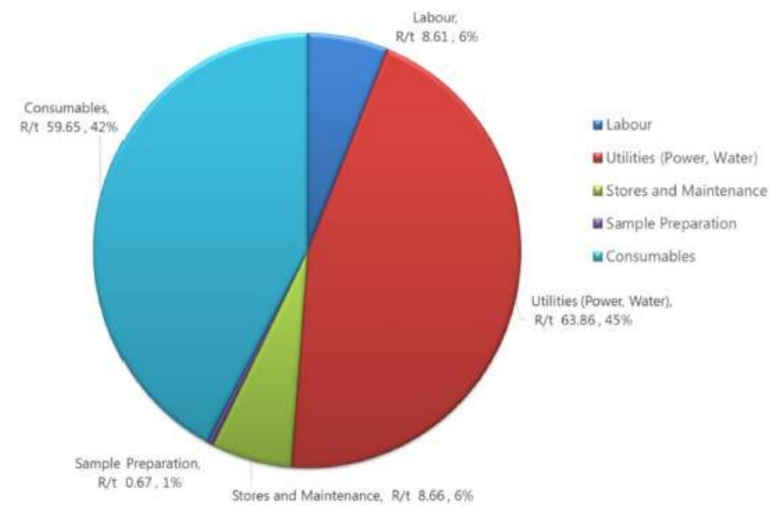


Figure 21-9: Process Plant Operating Cost Summary over Life-of-Mine

21.3.4.2 Plant Operating Cost Exclusions

The following are specific exclusions from the plant operating cost estimate:

- No allowance is made in the operating cost estimate for ramp-up of fixed costs prior to the start of hot commissioning — i.e. on-boarding of operating personnel etc.
- All VAT, import duties and / or any other statutory taxation, levies.
- Contributions to social programs.
- All owner's budget costs, head office, administration charges, payroll etc., other than labour.
- Contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements.
- "Stay in business" capital.
- Closure costs.
- Ore handling and reclaiming from stockpiles.
- Residue Disposal Facility management cost.
- Assaying costs.
- Concentrate transport to offtake partners.
- Concentrate loading into trucks prior to dispatch to smelters.
- Concentrate off take agreement costs / penalties.

Any contingency allowances for the operating cost estimate should be evaluated as part of a sensitivity analysis of the financial model.

21.3.4.3 Plant Operating Cost Inputs

21.3.4.3.1 Process Plant Labour

Labour costs were determined based on a typical staffing model for PGM concentrator plants. The steady state staffing complement is outlined in Table 21-18 . It is further noted that shared services staffing is allowed for under shared services.

Table 21-18: Waterberg Processing Plant Staffing Model per Phase

Function	At Work Compliment Phase 1	At Work Compliment Phase 2
Total	**149**	**204**
Management & Overheads	4	4
Administration	7	7
Office and Change house	13	17
Metallurgy	2	3
Plant Process	72	112
Engineering	38	48
Concentrator Stores	5	5
Plant Sample Prep Lab	8	8

The following staff is allocated under shared services:

- Information Technology (IT)
- Accounting
- Procurement
- Human Resources (HR)
- Sanitation
- Safety
- TSF management
- Security and access control

Additional provision has been made in the overall complex operating cost to cater for unavailability of personnel. The rates included in the operational cost estimate for the labour portions is based on typical rates obtained from databases and is aligned with the mining labour rates. These rates compare well to similar concentrators.

21.3.4.3.2 Process Plant Power

The rates used in the operational cost estimate for power are based on Eskom Megaflex tariffs. These tariffs were used in conjunction with the Eskom defined time periods to calculate an average power rate of 91 c/kWh, including network and administration charges; a variable rate of 61.6 c/kWh was applied.

The connected and absorbed loads used are as per the Waterberg mechanical equipment list which took into account total utilisation and efficiency factors of each installed drive.

The total connected load for Phase 1 is estimated at 45.6MW, with an absorbed load of 30.6MW, with a total connected load of 91.2MW for Phase 2. The absorbed load for Phase 2 is estimated at 62.5MW.

21.3.4.3.3 Water

The water consumption is based on the process plant water balance and anticipated water losses associated with the TSF.

The water requirement is calculated at 3.5 Ml/day for Phase 1 and 6.9 ML/day for Phase 2, at a cost of R2.15/m^3.

This cost is based on the assumption that the majority of the process plant raw water will be sourced from underground sources with the balance from sewage plant effluent.

21.3.4.3.4 Stores and Maintenance

The stores and maintenance costs included in the operation costs estimated is based on replacement factors applied to the mechanical equipment supply costs .

21.3.4.3.5 Concentrate Transport

No provision has been for concentrate loading and transport costs within the process plant operating costs.

21.3.4.3.6 Laboratory

A centralized laboratory facility is included in the Waterberg project design. No cost for assaying has been included in the process plant operating costs, as these costs have been allowed for within the centralized services cost estimate.

A monthly allowance has been made for the process plant sample preparation consumables, while labour for the process plant sample preparation laboratory has been included in the plant labour cost (See Table 21-18).

21.3.4.3.7 Residue Disposal

No provision has been for any RDF management cost within the process plant operating costs.

21.3.4.3.8 Consumables

21.3.4.3.8.1 Mill Liners

Allowance has been made for replacement of liners based on calculations incorporating the material abrasion index data from test work and grinding media consumptions as per simulations from the DRA in-house comminution consultant.

The liner costs used are as per the pricing received from the preferred mill supplier.

21.3.4.3.8.2 Crusher Liners

The costs used for the primary, secondary and tertiary crusher liners were as per the pricing and expected consumption data received from the preferred crusher supplier.

21.3.4.3.8.3 Reagents and Grinding Media

Reagent cost quotations were received from the following reputable reagent suppliers:

- ChemQuest
- Axis House
- SENMIN, and
- Protea Mining Chemicals.

The reagent costs used in the operational cost estimate are based on the average values from the quotations received from the above mentioned suppliers.

The reagent consumptions are based on test work consumptions, and no allowance has been made for build-up of reagents in the process water circuit which could ultimately lead to lower reagent consumptions.

Grinding media consumptions are based on calculations by the DRA in-house comminution consultant.

21.3.5 Engineering and Infrastructure Operating Cost

Engineering and infrastructure operating cost amounts to R 11,04 billion (R 107 /t) comprising 19% of the total operating cost as displayed in Figure 21-4. Table 21-19 provides a breakdown of the engineering and infrastructure cost per cost category.

Table 21-19: Total LoM Engineering and Infrastructure Operating Cost Breakdown per Cost Category

Engineering and Infrastructure Cost per Cost Category	Total LoM (ZAR Million)		Average LoM (ZAR/t)	
Materials & Supplies	R	3 944	R	38.41
Labour	R	3 657	R	35.62
Utilities	R	3 435	R	33.46
Total OPEX Cost	**R**	**11 036**	**R**	**107.49**

Materials and supplies together will be elaborated on further in the subsections below.

Power costs estimated for fixed equipment and infrastructure associated to engineering and infrastructure is reported under the infrastructure sub area section. In reality this power cost will be shared amongst the different sub areas. Similarly, labour cost for engineering is reported under the maintenance sub area even though labour is shared amongst sub areas. Engineering and infrastructure operating cost can be further detailed into infrastructure, maintenance, logistics, construction and services. Figure 21-10 below displays a pie chart that provides a cost breakdown of each of these categories.

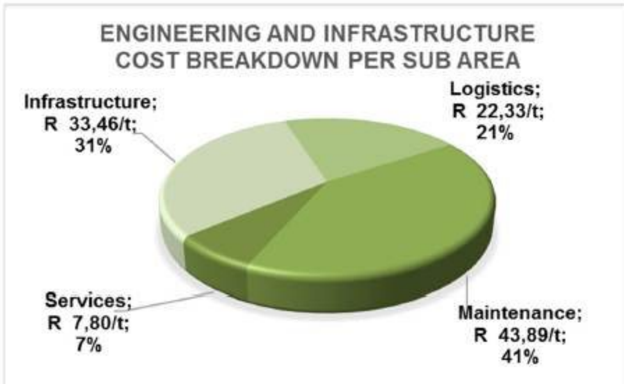


Figure 21-10: LoM Average R/t Engineering and Infrastructure Operating Cost breakdown per Sub Area

Maintenance constitutes 41% of the total engineering and infrastructure cost, followed by infrastructure at 31% and logistics at 21%.

Table 21-20 provides in depth information detailing LoM totals and average ZAR per ore tonnes broken cost, per sub area, per cost category and sub category. The far right column expresses the cost as a percentage of total engineering and infrastructure operating cost.

Table 21-20: Engineering and Infrastructure Cost Detail per Sub Area and Cost Category

Sub Area	Cost Category	Sub Cost Category	Total LOM (ZAR Million)		Average LOM (ZAR/t)		% of Total Engineering & Infrastructure Cost
Infrastructure	Utilities	Power	R	3 421	R	33.33	31.0%
		Water	R	14	R	0.13	0.1%
Logistics	Materials & Supplies	Consumables	R	2 293	R	22.33	20.8%
Maintenance	Labour	Direct labour	R	3 657	R	35.62	33.1%
	Materials & Supplies	Spares	R	850	R	8.28	7.7%
Services	Materials & Supplies	Consumables	R	801	R	7.80	7.3%
Total			**R**	**11 036**	**R**	**107.49**	**100%**

21.3.5.1.1 Maintenance

Engineering and infrastructure maintenance cost can be broken into material and supplies and labour. Materials and supplies comprise of spares and consumables required to maintain mechanical, piping, civil, electrical, platework, structural, ventilation and control and instrumentation components (fixed equipment and infrastructure). The cost for maintenance materials and supplies amount to R 8.28 per ore tonnes broken.

Engineering and infrastructure maintenance operating costs are derived through lifecycle costing methodologies. Maintenance philosophies were workshopped with the engineering team to derive maintenance intervals and costing supported by supplier pricing.

Engineering and infrastructure labour cost is the highest cost contributing component of the engineering and infrastructure operating cost, averaging R 35.62 per ore tonnes broken over LoM. An average labour complement of 847 is required with 1064 personnel at peak (year 2022) operation.

21.3.5.1.2 Infrastructure

Infrastructure comprise of power and water cost and is shared amongst all engineering and infrastructure sub areas. The total fixed and variable power cost for engineering equipment and infrastructure amounts to R 33.33 per ore tonnes broken and the cost for water is negligible at R 0.13 per ore tonnes broken. The bulk of the water supply will be sourced from ground water and a small bulk water consumption allowance for the mine (excluding process) was estimated at 1037 m^3 per day based on a water mass balance calculation performed the engineering team. A weighted average for the water supply rate amounted to R 1 per m^3.

21.3.5.1.3 Logistics

Engineering logistics entails support service and consists of consumables for underground logistic support fleet (such as utility vehicles and cassettes, personnel carriers, boom basket lifts, forklifts etc.) at an average LoM cost of R 22.33 per ore tonnes broken.

21.3.5.1.4 Services

Similarly, engineering services consists of consumables for underground services support fleet (such as graders, water trucks, light diesel vehicles etc.) at a cost of R 7.80 per ore tonnes broken.

21.3.6 General and Administration Operating Cost

General and administration operating cost constitutes a small portion (7%) of the total LoM operating cost at R 4,18 billion (R 41 /t) and consist of materials and supplies, labour, fixed overheads and external services. Table 21-21 provides a breakdown of the general and administration cost per cost category.

Table 21-21: Total LoM G&A Operating Cost Breakdown per Cost Category

G&A Cost per Cost Ca6tegory	Total LoM (ZAR Million)		Average LoM (ZAR/t)	
Materials & Supplies	R	60	R	0.59
Labour	R	2 872	R	27.97
Fixed Overheads	R	959	R	9.34
External Services	R	289	R	2.82
Total OPEX Cost	**R**	**4 180**	**R**	**40.71**

From Figure 21-11 it is evident that labour comprise the bulk of the general and administration cost at 69%, followed by fixed overheads at 23% and external services at 7%.

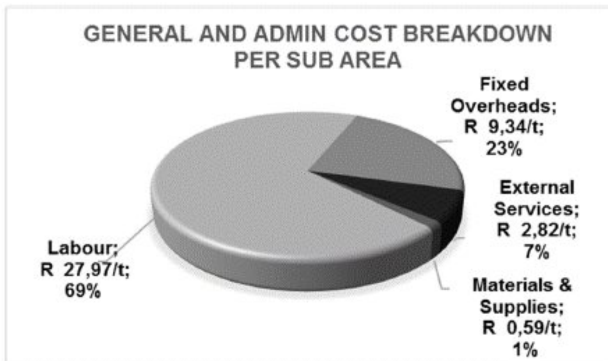


Figure 21-11: LoM Average R/t G&A Operating Cost Breakdown per Cost Area

Labour cost is the highest cost contributing component of general and administration operating cost, averaging R 27.97 per ore tonnes broken over LoM. General and administration labour include general office staff such as finance, human resources as well as technical services and health and safety personnel. An average labour complement of 279 is required with 377 personnel at peak (year 2025) operation.

The fixed overhead cost comprising of insurances coverage, retrenchment packages and owner team, IT and security cost is reported under the general and administration area and amounts to R 9.34 per ore tonnes broken over LoM.

Contracted security and waste removal cost forms part of external services for the general and administration area and averages R 2.82 per ore tonnes broken of the total LoM operating cost.

A small allowance for stationary, printing and general office consumables amounts to R 0.59 per ore tonnes broken over LoM for materials and supplies.

21.3.7 Concentrate Transport Operating Cost Summary

The concentrate will be transported by the project to the available smelters within South Africa. There are three smelting hubs within South Africa, namely Polokwane (109km south east), Northam (312km south west) and Rustenburg (417km south south west).

As no negotiations have commenced for the off-take agreement, it is assumed that 33% of the concentrate production will be shipped to each smelting hub, resulting in an average distance for shipping of 280km.

The average transport cost for concentrate is based on the actual shipping cost for PTM's Maseve mine at R1.42 per wet concentrate tonne per kilometer. The concentrate moisture will be 12% thus resulting in the cost per dry tonne delivered being R452 which is based on transport rate and moisture content reduction.

22. Economic Analysis

22.1 Summary of Financial Results

The key features of the Waterberg 2016 PFS include:

- Development of a large, mechanised, underground mine that is planned at a 7.2Mtpa throughput scenario;
- Planned steady state annual production rate of 744 koz of platinum, palladium, rhodium and gold (4E) in concentrate;
- Estimated Capital to full production requirement of approximately ZAR15,906 billion (US$1,060 million), including ZAR999 million (US$67 million) in contingencies;
- Peak funding ZAR13,694 million (US$914 million);
- After-tax Net Present Value (NPV) of ZAR4,805 million (US$320 million), at an 8% discount rate (three year trailing average price desk 31 July 2016 US$1,212/oz Pt, US$710/oz Pd, US$984/oz Rh, US$1,229/oz Au, US$/ZAR 15);
- After-tax Net Present Value (NPV) of ZAR7,610 million (US$507 million), at an 8% discount rate (Investment Bank Consensus Price) US$1,213/ozPt, US$800/oz Pd, US$1,000 Rh, US$1,300/oz Au, US$/ZAR 15;
- After-tax Internal Rate of Return (IRR) of 13.5% (three year trailing average price deck); and
- Internal Rate of Return (IRR) of 16.3% after tax (Investment Bank Consensus Price).

Mine production is shown in Figure 22-1 and the after tax cash flow is shown in Figure 22-2. The key production and financial results including Net Present Value at 8% Discount Rate (NPV8) and Internal Rate of Return (IRR) of the Waterberg 2016 PFS are shown in Table 22-1.

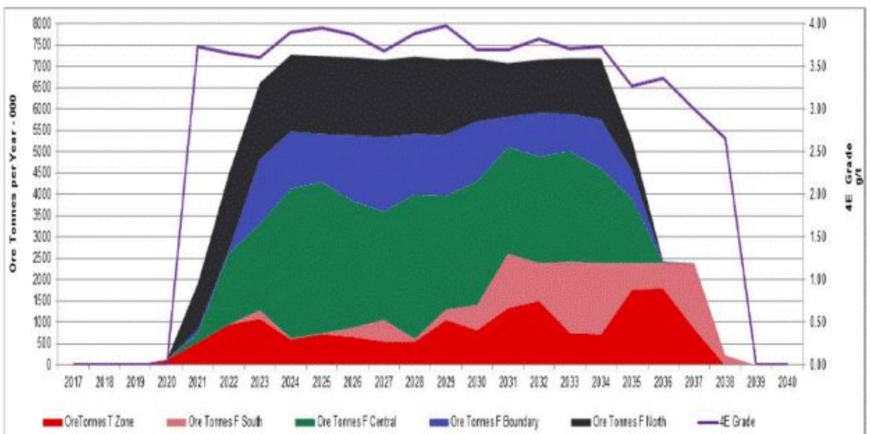


Figure 22-1: Mining Production

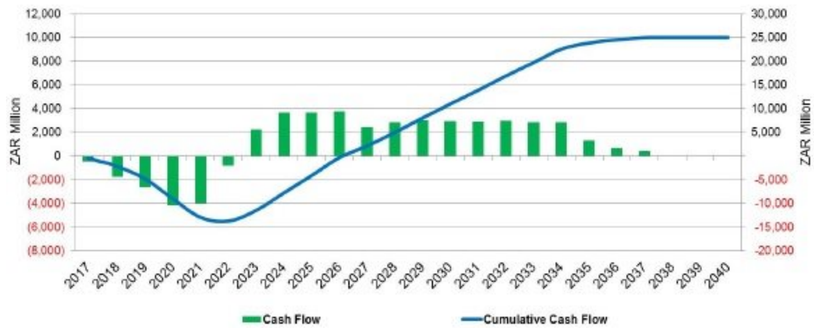


Figure 22-2: Cumulative Cashflow after Tax

Table 22-1: Waterberg 2016 PFS Results

Item	Units	Total
Mined and Processed	Mtpa	7.20
Platinum	g/t	1.11
Palladium	g/t	2.29
Gold	g/t	0.29
Rhodium	g/t	0.04
4E	g/t	3.73
Copper	%	0.08
Nickel	%	0.15
Recoveries		
Platinum	%	82.5%
Palladium	%	83.2%
Gold	%	75.3%
Rhodium	%	59.4%
4E	%	82.1%
Copper	%	87.9%
Nickel	%	48.8%
Produced in Concentrate		
Concentrate	ktpa	285
Platinum	g/t	24.2
Palladium	g/t	51.5
Gold	g/t	4.9
Rhodium	g/t	0.6
4E	g/t	81
Copper	%	1.9

Item	Units	Total
Nickel	%	1.8
Recovered Metal in Concentrate		
Platinum	kozpa	222
Palladium	kozpa	472
Gold	kozpa	45
Rhodium	kozpa	6
4E	kozpa	744
Copper	Mlbpa	11
Nickel	Mlbpa	12
Key Financial Results (3 Year Trailing Price Deck 31 July 2016 — US$/ZAR 15)		
Life of Mine	years	19
Capital to Full Production	US$M	1060
Mine Site Cash Cost	US$/oz 4E	389
Total Mine Cash Costs After Credits	US$/oz 4E	248
Total Cash Costs After Credits	US$/oz 4E	481
All in Costs After Credits	US$/oz 4E	661
Site Operating Costs	US$/t Milled	38
After Tax NPV @ 8%	US$M	320
After Tax IRR	%	13.5
Project Payback Period (Start First Capital)	years	10
Market Consensus Price Case		
After Tax NPV8	US$M	507
After Tax IRR	%	16.3

22.2 Model Assumptions

22.2.1 Pricing and Discount Rate Assumptions

The Waterberg Project level financial model begins on 1 July 2016. It is presented in real money terms, cash flows are assumed to occur evenly during each calendar year and a mid-year discounting approach is taken. The base case real discount factor applied to the analyses is 8%. No allowance for inflation or exchange fluctuation has been made in the analyses.

The following prices, based on a 3 year trailing average in accordance with U.S. Securities and Exchange Commission (“SEC”) guidance, was used for the assessment of Resources and Reserves.

The economic analysis uses price assumptions of US$1,212/oz Pt, US$710/oz Pd, US$1,229/oz Au, US$984/oz Rh, US$6.10/lb Ni, and US$2.56/lb Cu as at 31 July 2016 Realisation costs are described in Section 19. Costs estimated in ZAR have been converted to US$ at an exchange rate of 15 ZAR/US$. The Spot price and Investment Bank Consensus Price Deck were also valuated in Section 22.7 for Sensitivity Analysis.

For the purpose of the financial model, it is assumed that the option of taking payment advances will be exercised. Pipeline financing is the interest charged levied in terms of the Concentrate Offtake Agreement against payment advances on concentrate sales ahead of the settlement dates contained in the agreement.

The key economic assumptions for the analyses are shown in Table 22-2.

Table 22-2: Economic Assumptions

Parameter	Unit	3 Yr Trailing Average 31 Jul 2016	Spot Price 6 Oct 2016	Investment Bank Consensus Price 16 Sep 2016
Platinum	US$/oz	1.212	964	1.213
Palladium	US$/oz	710	668	800
Gold	US$/oz	1.229	1.255	1.300
Rhodium	US$/oz	984	675	1.000
Basket (4E)	US$/oz	899	798	960
Nickel	US$/lb	6.10	4.52	7.50
Copper	US$/lb	2.56	2.17	2.90
Base Metals Refining Charge	% Gross Sales	85%		
Copper Refining Charge	% Gross Sales	73%		
Nickel Refinery Charge	% Gross Sales	68%		

22.2.2 Taxes and Royalties

The majority of taxes and fees payable to the government under Republic of South Africa legislation are the Corporate Income Tax (28%) and a production royalty. The royalty rate for refined minerals is a percentage determined as per legislation and calculation shown below.

- Royalty Act 28 (2008; Government Gazette No. 31635), and the Mineral and Petroleum
- Resources Royalty (Administration) Act No. 29 (2008; Government Gazette No. 31642):
- Royalty % = 0.5 + [EBIT/(Gross Sales * 9)] * 100, with a maximum of 7% and a minimum of 0.5%, for production of refined minerals.

22.3 Indirect Costs

A allowance of ZAR 166 million (US$11 million) was provided for the Social Labour Plan as required to assists in the compilations of the Social and Labor Plan (SLP). The SLP forms part of the Mining Right application process.

22.4 Project Results

The results of the financial analysis show an After Tax NPV8% of ZAR4,805M. The case exhibits an after tax IRR of 13.5% and a payback period of ten years using three year trailing average prices. The estimates of cash flows have been prepared on a real basis as at 1 January 2016 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 21-18. The mining production statistics are shown in Table 21-18 and Table 22-4.

Table 22-3: Financial Results

Item	Discount Rate	ZAR Millions (Before Taxation)	ZAR Millions (After Taxation)	USD Millions (Before Taxation)	USD Millions (After Taxation)
Net Present Value	Undiscounted	36,096	25,042	2,406	1,669
	4.0%	18,213	11,883	1,214	792
	6.0%	12,666	7,808	844	520
	8.0%	8,565	4,805	571	320
	10.0%	5,519	2,584	368	172
	12.0%	3,249	939	217	62
	14.0%	1,555	-278	104	-19
Internal Rate of Return		16.6%	13.5%	16.6%	13.5%
Project Payback Period (Years)		10	10	10	10

Table 22-4: Mining Production Statistics

Item	Unit	Total LOM	LOM Annual Avg
Ore Production			
Mineral Reserve	Mt	103	7.2
Ore Milled	Mt	103	7.2
T-Zone	g/t	3.94	3.94
F South	g/t	3.78	3.78
F Central	g/t	3.59	3.59
F Boundary	g/t	3.75	3.75
F North	g/t	3.78	3.78
4E	g/t	3.73	3.73
Copper	%	0.08	0.08
Nickel	%	0.15	0.15
Recoveries			
Platinum	%	82.5	82.5
Palladium	%	83.2	83.2
Gold	%	75.3	75.3
Rhodium	%	59.4	59.4
4E	%	82.1	82.1
Copper	%	87.9	87.9
Nickel	%	48.8	48.8
Concentrate Produced			
Concentrate	kt	3,880	285
Platinum	g/t	24.2	24.2
Palladium	g/t	51.5	51.5
Gold	g/t	4.9	4.9
Rhodium	g/t	0.6	0.6

Item	Unit	Total LOM	LOM Annual Avg
4E	g/t	81	81
Copper	%	1.9	1.9
Nickel	%	1.8	1.8
Recovered Metal			
Platinum	koz	3,029	222
Palladium	koz	6,297	482
Gold	koz	715	45
Rhodium	koz	73	6
4E	koz	10,114	744
Copper	Mlb	168	11
Nickel	Mlb	163	12

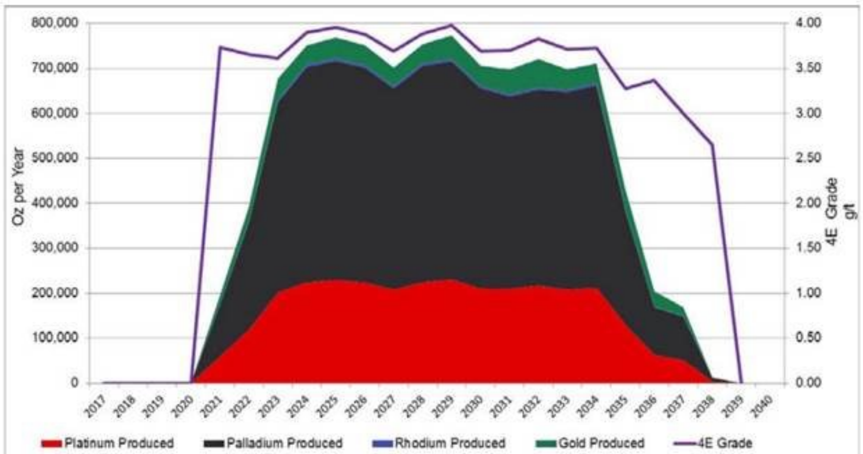


Figure 22-3: Mining Production

22.5 Capital and Operating Cost Summary

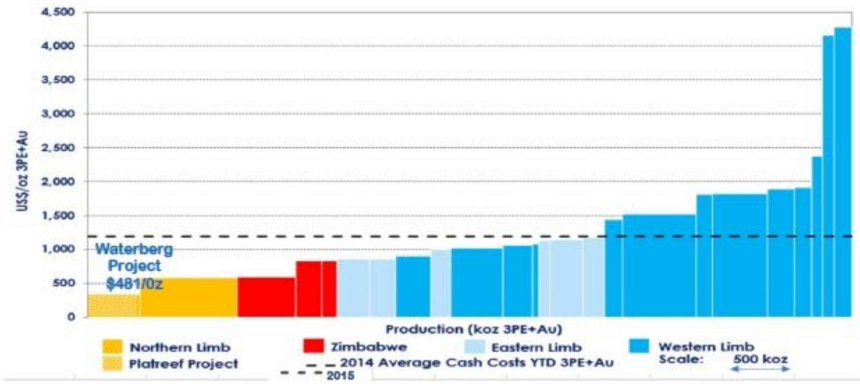
Mine site cash costs are summarised in Table 22-5. The revenues and operating costs are presented in Table 22-6.

The higher nickel and copper grades contribute to lower operating cash costs for the project as illustrated by Figure 22-4. Among the current and future Northern Limb producers, Waterberg's estimated cash cost of US$481 per 4E ounce, net of copper and nickel by-product credits, ranks in the lowest decile of the cash-cost curve for the South African platinum industry.

Table 22-5: Cash Costs after Credits

Item	US$/oz Payable 4E in Concentrate Life-of-Mine Average	5-Year Average 2022 - 2026	10-Year Average 2022 - 2031
Mine Site Cash Cost	389	390	374
Nickel Credits	98	97	98
Copper Credits	42	40	40
Total Mine Cash Costs After Credits	**248**	**253**	**236**
Realisation Cost (Smelter "cost", Transport)	232	224	231
Total Cash Costs After Credits	**481**	**477**	**467**

Exclusive of smelter discount, on site costs are estimated to be US$248 (15R/US$) per 4E ounce for the life of mine including copper and nickel as a credit.



Producer Cost Data Source: SFA (Oxford)

Figure 22-4: Cash Cost Comparison Waterberg and 2015 Producers

Table 22-6: Operating Costs and Revenues

Item	Life of Mine Total ZARm	Life of Mine Total US$M	Life of Mine Average ZAR/t	Life of Mine Average US$/t
Gross Sales Revenue	157,733	10,516	1,536.36	102.42
Less: Realisation Costs				
Transport Costs	1,747	116	17.02	1.13
Treatment & Refining Charges	26,626	1,775	259.35	17.29
Royalties	6,895	459	67.16	4.48
Total Realisation Costs	35,268	2,351	343.52	22.90
Net Sales Revenue	122,464	8,164	1,192.83	79.52
Site Operating Costs				
Mining	38,950	2,597	379.39	25.29

Item	Life of Mine Total ZARm	Life of Mine Total US$M	Life of Mine Average ZAR/t	Life of Mine Average US$/t
Processing & Tailings	15,864	1,058	154.52	10.30
General & Administration	4,180	279	40.71	2.71
Total Operating Costs	**58,994**	**3,933**	**574.62**	**38.31**
Operating Margin	63,470	4,231	618.21	41.21
Operating Margin (%)	40 %	40 %	40 %	40%

Table 22-7: Total Project Capital Cost

	Full Production ZAR (M)	Sustaining ZAR (M)	Total ZAR (M)	Full Production USD (M)	Sustaining USD(M)	Total USD (M)
Mining						
Underground Mining	5,281	5,297	10,579	352	353	705
Surface Infrastructure	803	—	803	54	—	54
Replacement & Refurb Cost	8	4,469	4,477	1	298	298
Subtotal	**6,092**	**9,766**	**15,859**	**406**	**651**	**1,057**
Treatment						
Plant 1	1,676	22	1,698	112	1	113
Plant 2	1,173	137	1,310	78	9	87
Subtotal	**2,850**	**159**	**3,008**	**190**	**11**	**201**
Infrastructure						
Shared Services & Infrastructure	1,063	43	1,106	71	3	74
Regional Infrastructure	2,566	—	2,566	171	—	171
Subtotal	**3,629**	**43**	**3,672**	**242**	**03**	**245**
Indirects						
Site Support Services	691	67	758	46	5	51
Project Delivery Management	1,399	148	1,547	93	10	103
50 Other Capitalised Costs	246	83	329	16	6	22
Contingency	999	1,203	2,202	67	80	147
Subtotal	**3,335**	**1,501**	**4,836**	**222**	**100**	**322**
Capex excl Contingency	**14,907**	**10,266**	**25,173**	**994**	**684**	**1,678**
Capex incl Contingency	**15,906**	**11,468**	**27,374**	**1,060**	**765**	**1,825**

22.6 Project Cash Flows

Cumulative cash flow after tax is depicted in Figure 22-5 and a complete cash flow is provided in Table 22-8.

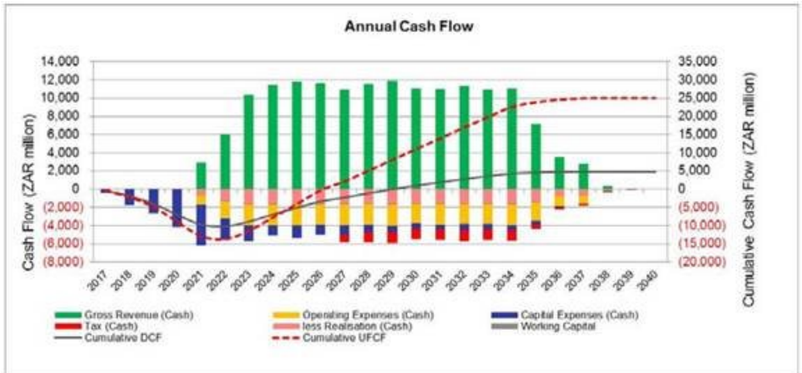


Figure 22-5: Cumulative Cashflow after Tax

Table 22-8: Cumulative Cash Flow

ZAR (M)	Year											
Description	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6-10**	**11-LOM**	**TOTAL**
Gross Revenue					2,925	6,001	10,376	11,510	11,805	56,988	58,127	157,733
Less Realisation Costs					564	1,133	2,292	2,615	2,652	13,007	13,006	35,268
Net Sales Revenue					2,361	4,869	8,085	8,895	9,153	43,981	45,121	122,464
Site Operating Costs												
Mining					1,125	2,269	2,669	2,611	2,639	12,884	14,753	38,950
Processing & Tailings					358	667	1,073	1,085	1,084	5,422	6,175	15,864
General & Admin					158	240	254	255	257	1,260	1,756	4,180
Less Site Operating Costs					1,641	3,176	3,996	3,951	3,980	19,566	22,684	58,994
Working Capital					-116	-85	-153	-155	-158	-627	-21	-1,315
Less Capital Expenditure	435	1,719	2,634	4,164	4,559	2,394	1,676	1,099	1,351	4,373	2,969	27,374
Net Cash Flow Before Tax	**-435**	**-1,719**	**-2,634**	**-4,164**	**-3,955**	**-786**	**2,259**	**3,691**	**3,663**	**19,415**	**19,447**	**34,781**
Less Income Tax Expense										4,294	5,445	9,739
Net Cash Flow After Tax	**-435**	**-1,719**	**-2,634**	**-4,164**	**-3,955**	**-786**	**2,259**	**3,691**	**3,663**	**15,122**	**14,002**	**25,042**

US$ M	Year										
Description	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6-10**	**11-LOM**	**TOTAL**
Gross Revenue					195	400	692	767	787	3,799	3,875	10,516
Less Realisation Costs					38	76	153	174	177	867	867	2,351
Net Sales Revenue					157	325	539	593	610	2,932	3,008	8,164
Site Operating Costs												
Mining					75	151	178	174	176	859	984	2,597
Processing & Tailings					24	44	72	72	72	361	412	1,058
General & Admin					11	16	17	17	17	84	117	279
Less Site Operating Costs					109	212	266	263	265	1,304	1,512	3,933
Working Capital					-9	-6	-10	-10	-11	-42	—	-88
Less Capital Expenditure	29	115	176	278	304	160	112	73	90	292	198	1,825
Net Cash Flow Before Tax	**-29**	**-115**	**-176**	**-278**	**-265**	**-52**	**151**	**246**	**244**	**1,294**	**1,297**	**2,319**
Less Income Tax Expense										286	363	649
Net Cash Flow After Tax	**-29**	**-115**	**-176**	**-278**	**-265**	**-52**	**151**	**246**	**244**	**1,008**	**934**	**1,669**

22.7 Sensitivity or Other Analysis using Variants in Commodity Price, Grade, Capital and Operating Costs

The project yields the following results using the spot price of 6 October 2016 and Investment Bank Consensus Price Deck of 16 September 2016.

Table 22-9: Sensitivity Analysis — Spot Price and Investment Bank Consensus Price Deck

		Spot Price 6 Oct 2016 @ ZAR/USD 15		Investment Bank Consensus Price @ ZAR/USD 15	
Item	**Discount Rate**	**ZAR Millions After Taxation**	**USD Millions After Taxation**	**ZAR Millions After Taxation**	**USD Millions After Taxation**
Net Present Value	8%	511	34	7,610	507
Internal Rate of Return		8.6%	8.6%	16,3%	16,3%

The following parameters have a significant impact on the project results. The project sensitivity has been conducted for various parameters with variations of plus and minus 10% in each input. The resented in the graphs below for post-tax cash flows .

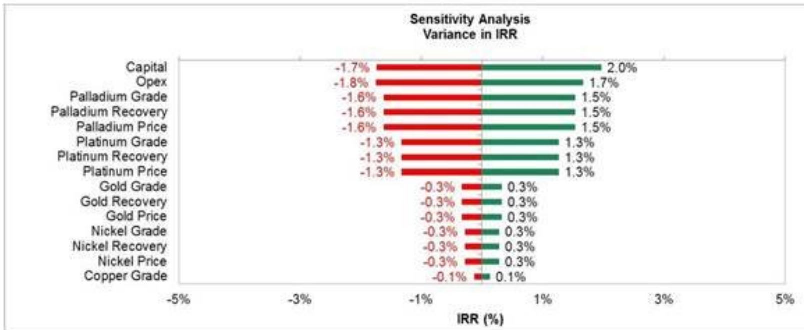


Figure 22-6: IRR Sensitivity with 10% Variance in Parameter

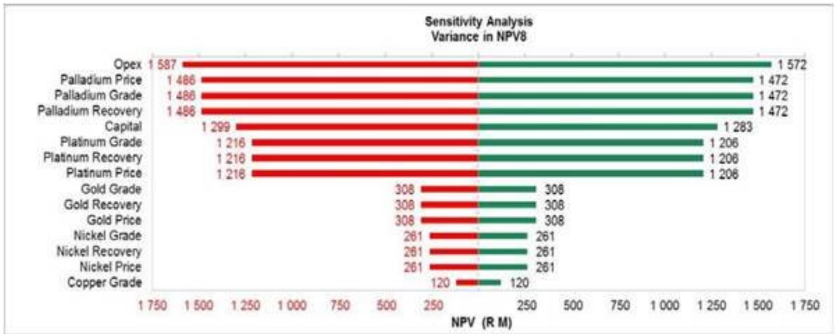


Figure 22-7: NPV [8] Sensitivity with 10% Variance in Parameter

The Project sensitivities has been conducted for metal price, head grade, capital costs and operating cost, with variations of plus and minus 10% and 20% in each input. The data is presented in graphs and in below for post-tax cash flows.

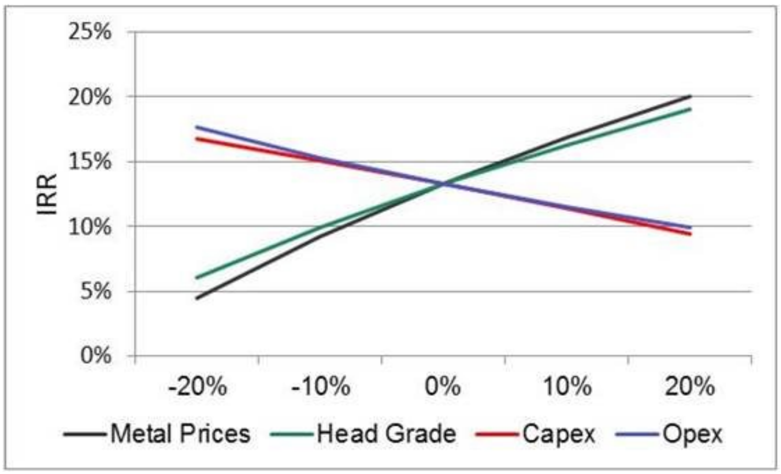


Figure 22-8: IRR Sensitivity — Post Tax

Table 22-10: Sensitivity Analysis — Post Tax

Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter
Metal Prices	-20%	-10%	0%	10%	20%
IRR (post-tax)	5%	10%	13.5%	17%	20%
NPV (8% Discount) (R000)	-2,467	1,211	4,805	8,344	11,854
NPV (8% Discount) ($000)	-164	67	320	556	790
Head Grade	-20%	-10%	0%	10%	20%
IRR (post-tax)	6%	10%	13.5%	16%	19%
NPV (8% Discount) (R000)	-1,513	1,562	4,805	7,562	10,505
NPV (8% Discount) ($000)	-101	104	320	504	700
Capex	-20%	-10%	0%	10%	20%
IRR (post-tax)	17%	15%	13.5%	12%	10%

Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter	Change in Parameter
NPV (8% Discount) (R000)	8,161	6,484	4,805	3,109	1,395
NPV (8% Discount) ($000)	544	432	320	207	93
Opex	-20%	-10%	0%	10%	20%
IRR (post-tax)	18%	16%	13.5%	12%	10%
NPV (8% Discount) (R000)	7,435	6,121	4,805	3,246	2,124
NPV (8% Discount) ($000)	496	408	320	216	142

It is noted that the project is most sensitive to metal prices, head grades and to a lesser extent operating cost.

23. Adjacent Properties

Numerous mineral deposits were outlined along the Northern Limb of the Bushveld Complex.

The T-Zone on the Waterberg Project is in a different position in the Northern Limb geology as reported for the other deposits and the T-Zone has distinctively different metal ratios with elevated gold values compared to the reported other deposit grades. The F-Zone has some similarities to the other Northern Limb deposits in metal prill splits, however there may be distinct differences in the geological units containing the mineralisation.

23.1 The Pan Palladium/Impala Platinum Joint Venture Supply

The Pan Palladium/Impala Platinum JV on the most northern farm on Platreef outcrop has reported resources of 50Mt at 1.19g/t (2PGE+Au), 0.07% Ni, 0.21% Cu (Pan Palladium Annual Report, 2003). The qualified person for this report was unable to verify the information on which it is based. It is noted that this estimate is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

23.2 Mogalakwena Mine

Some 60km south of the project is the world's largest opencast platinum mine, Mogalakwena Mine (formerly Potgietersrust Platinum Mine), which mines the Platreef and produced 392,000 platinum ounces in 2015. The latest Mineral Resource and Reserve statement for Mogalakwena Mine is available on the website www.angloplatinum.

23.3 Akanani Project

Akanani Project, majority held by Lonmin, is down dip of the Anglo Platinum Mogalakwena Mine, is an exploration project with studies continuing to develop it into a viable operation. Information pertaining to this project including the latest Mineral Resource and Reserve statement are available on the Lonmin website (www.lomin.com).

23.4 Boikgantsho Project

Located on the Northern Limb of the Bushveld Complex and adjacent to Anglo Platinum's Mogalakwena Mine, this project was acquired through a land acquisition by Atlatsa Resources (formerly Anooraq Resources) in 2000 and a joint venture with Anglo Platinum in 2004.

Historically, exploration drilling was conducted at the project site, which has led to the estimate of indicated and inferred Mineral Resources. A preliminary economic assessment was completed in 2005; the results of this work showed that the project warrants further investigation.

Details of the project as well as Mineral Resource and Reserve information are available via the company website (www.atlatsaresources.co.za).

23.5 Harriet's Wish and Aurora Projects

Sylvania Resources is undertaking exploration activities on the extreme northern end of the Northern Limb on the farm Harriet's Wish which is adjacent to and contiguous with the southern boundary of the Waterberg Project. According to Sylvania, the northern portion of Harriet's Wish is covered by the Waterberg Sediments and the drill holes have intersected PGM mineralisation with descriptions similar to that of mineralisation found in the Waterberg Project. The author has not been able to verify this data. No Mineral Resource or Reserve was declared. (www.sylvaniaplatinum.com)

23.6 Platreef Project (Ivanplats)

The Platreef Project, is jointly owned by Ivanplats (90%) and a Japanese consortium of Itochu Corporation; Japan Oil, Gas and Metals National Corporation (JOGMEC) and JGC Corporation (10%). The Platreef Project is a recently discovered underground deposit of thick, PGM-nickel copper mineralisation on the southern end of the Northern Limb of the Bushveld Complex (close to Mokopane). The Platreef Project hosts the southern sector of the Platreef on three contiguous properties namely:

- Turfspruit
- Macalacaskop
- Rietfontein.

Ivanplats has delineated a large zone of mineralisation within the Platreef, which essentially comprises a steeply-dipping, near-surface mineralized area and a gently-dipping to sub horizontal (<15º) deeper zone from approximately 700m depth downward to 1500m (the "Flatreef").

Ivanhoe has completed a Pre-Feasibility Study on sinking an exploration shaft. The mineralisation is considered open for expansion along the southern and western boundaries of the Flatreef deposit. The northernmost property, Turfspruit, is contiguous with, and along strike from, Anglo Platinum's Mogalakwena group of properties and mining operations. A Mineral Resource and a Mineral Reserve were declared. (www.ivanplats.com)

24. Other Relevant Data and Information

Some Sections of this report have associated appendices, which have not been attached to or included in this report. The size of the electronic media files are prohibitively large to distribute widely.

To the best of the author's knowledge, there is no other relevant data or information, the omission of which would make this report misleading.

25. Interpretation and Conclusions

25.1 Relevant Results and Interpretations of the Information and Analysis

As part of the PFS Project risk management process, risk workshops were held to identify and assess risks related to the project.

The key risks potentially impacting the achievement of the project objectives were identified, together with their root causes and potential consequences. Primary mitigating strategies currently in place to address the risks were documented and where the current risk rating was considered unacceptably high, additional action items agreed to reduce it to an acceptable level.

Results of this Pre-Feasibility Study demonstrate that the Waterberg Project warrants development to a Feasibility Stage to ensure that investment opportunities presented for approval are well defined, risks appropriately considered and that sufficient work has been completed to allow the investment to ultimately move into Implementation.

It is the conclusion of the QPs that the PFS summarized in this technical report contains adequate detail and information to support a Pre-Feasibility level analysis.

The report authors are unaware of any unusual or significant risks, or uncertainties that would affect Project reliability or confidence based on the data and information made available. For these reasons, the path going forward must continue to focus on drilling activities and obtaining the necessary permitting approval, while concurrently advancing key activities in the Feasibility Study that will reduce project execution time.

25.1.1 Geology and Estimation Resources and Reserves

A Mineral Resource may be declared for the PTM Waterberg project. This resource comprises an Indicated Resource of 31 Million tonnes at 3.88g/t 4E for the T-zone; and 186 Million tonnnes at 3.49 g/t 4E for the F-zone. Additional Inferred Resources of 19 Million tonnes at 3.79g/t 4E for the T-Zone and 77 Million tonnes at 3.37g/t 4E for the F-zone. These resources are reported at a 4E grade cut-off of 2.5 g/t.

25.1.2 Risks and Opportunities

The highest risks associated with the Mineral Resource element of this project are related more to structural concerns than to estimation concerns. These risks are mitigated by the extensive geophysical programme that has been implemented.

25.1.3 Geotechnical and Rock Engineering

25.1.3.1 Geotechnical

Portal designs were created based on professional experience in similar ground environment and geotechnical information gathered from the inspection of four boreholes drilled near the proposed portals location.

In general, ground conditions are considered favourable for the proposed portals.

The suggested preliminary portals designs presented will have to be supported and approved with the finite element and limit equilibrium methods during the Definitive Feasibility Study (DFS) to reach an acceptable Factor of Safety (FoS) determined for the project.

Conditions on the site are favourable for the proposed infrastructure, provided that the precautionary measures for foundations and services are incorporated in the design and development of the site including:

- Soil raft foundation
- Site Drainage; Site drainage and service precautions are required to prevent the ingress of water under the structure
- Materials encountered on site are suitable for use in engineered layer work applications

25.1.3.2 Rock Engineering

25.1.3.2.1 Geotechnical database, domains and rock mass block model

A geotechnical database was established containing log data, laboratory test results and derived rock mass classifications. A 3D geotechnical block model was constructed as visual aid to interpreting the results throughout the orebody . Much of the orebody is covered but there are shortcomings with respect to the T-Reef

The data did not allow for geotechnical domaining and domains were defined according to available geological lithology.

There are indications that shear zones throughout the rock mass in the area of interest and these will require further study at DFS stage. Sufficent geological work and geotechnical work was completed for the mine design at this stage of study.

The rock mass quality across most domains can be described as fair to good for design purposes. The exceptions are the sediments and the sills which are of somewhat poorer quality in places;

Variation of rock mass quality between domains is in a narrow band;

25.1.3.2.2 Stress

No information was available of the local or regional stress regime.

It is reasonable, within the given constraints, to assume that the stress gradient is equal to 0.03MPa per meter (based on the average density of the intact rock) and that the K-ratio = 1, for the expected depth range of the Waterberg complex, which is 700m below surface and less.

25.1.3.2.3 Mining Methods

Initial stepped room and pillar mining was replaced by a more practical approach called Blind Longitudinal Retreat mining. Much of the T-reef and some of the F-reef were designed for this type of mining.

The rest of the orebody will be mined through sub-level open stoping methods along strike or transverse.

25.1.3.2.4 Surface Subsistence due to Mining

Surface subsidence as result of mining is highly unlikely.

25.1.3.2.5 Hydraulic Radius and Design Charts

The maximum unsupported stable span of the stope hanging wall and footwall was determined empirically using the two most widely accepted methods, namely Laubscher's caving chart, as described in Brown (2000) and otherwise known as the MRMR method, and the Matthews-Potvin N' stability number method, as described in Potvin (1988).

25.1.3.2.6 Designing within the Transition Zone

For the Waterberg Project, the stope spans were deliberately designed to lie within the Transition Zone in order to maximise extraction whilst reducing the chances of stress lockup in the back and possible pillar bursting. It must be understood that designing in the Transition Zone carries risk that is considerably higher than designs aimed for the stable zone because of high uncertainty around many variables. Success of this method will rely on strict adherence to recommended pre-cautionary measures.

25.1.3.2.7 Pillar Dimensions

Numerical and empirical analyses were employed to estimate required pillar dimensions for the various stoping methods. More rigorous analyses will be required during the next phase of the study.

25.1.3.2.8 Ground Control and Support Requirements

The geotechnical information acquired through this study is adequate to inform generic ground control guidelines. However, detailed layout and site-specific support design of Life of Mine (LoM) tunnels and large excavations will require geotechnical and geological studies through dedicated drilling and core logging very early or even prior to the Feasibility Study phase.

25.1.3.2.9 Ground Control and Support Requirements

The geotechnical information acquired through this study is adequate to inform generic ground control guidelines. However, detailed layout and site-specific support design of Life of Mine (LoM) tunnels and large excavations will require geotechnical and geological studies through dedicated drilling and core logging very early or even prior to the feasibility study phase.

25.1.3.2.10 Rock Mechanics Parameters for Mine Design

A set of Rock Mechanics Parameters to guide mine design was developed and presented.

25.1.3.3 Risks and Opportunities

Rock Mechanics related risks associated with the present study are presented in Balt (2016) together with connected recommended forward work. Following is a summary of the outcome from this risk analyses and forward work plan.

The identified risks are related to the following and refer mainly to what is required for the next phase of the study:

- Insufficient geotechnical investigations;
- Insufficient information with respect to ground water;
- Insufficient information on geological structure;
- Insufficient information with respect to the stress regime;
- Designing for the transition zone of stability charts; and
- Insufficient numerical modelling;

- Ground control measures and raise bore location test holes ;
- Seismicity; and
- Surface subsidence.

25.1.4 Mining Methods

The mine design and production schedules presented are deemed as reasonable for a PFS level of confidence.

Although, the BLR mining method is not widely utilized, it is the view of the project study team that the layouts and schedule rates areappropriate.

A number of potential optimization opportunities have been identified including reducing development waste and dilution.

25.1.4.1 Risks and Opportunities

The following mining related risks have been identified:

- Non-achievement of development advance rates resulting in schedule impacts.
- Under estimation of the unit costs for development.
- Design contingencies - The mine design of underground access infrastructure, other underground excavations and production areas should be prepared to higher level of confidence.
- Due to Step Room and Pillar and Blind Longitudinal Retreat being mining methods of limited use, this may lead to the non-achievement of tonnages and on-reef advance rates resulting in impact on production and unit costs.
- The selected strategy of initially targeting higher grade mining cuts has the potential of sterilizing lower grade resources.
- The design and scheduling of the additional tonnages at the tail of the production profile has been conducted using factored design parameters.
- Lack of local availability of appropriate skills for mining and engineering may lead to unavailability of skills to achieve development schedule and production targets resulting in cost and time impacts. This applies to the skills required for both the operation and maintenance of trackless mobile machinery.
- Fragmentation of blasted rock could have an impact on hauling cycle times, and therefore productivities, as well as the availability of the conveyor system.
- Congestion caused by the movement of underground traffic could result in longer cycle times which in turn could have an impact on production volumes.

The following are potential opportunities, which could have an impact on the project:

- Underground access methodologies and designs. This includes the positioning and orientation of main declines.
- Potential for further optimization of development end sizes for certain excavation types reducing dilution and waste development.
- Optimise the layouts of the SLOS — Longitudinal and SLOS — Transverse mining methods. Production ring designs and in-stope cleaning for lower inclinations would be specific aspects to be considered.
- Potential to optimize underground rock handling with the view of reducing hauling distances.

- Further optimization of the ventilation and refrigeration system.
- Investigate the potential to increase effective face time for specific production activities such as loading and hauling.
- The deposit remains open along strike and down dip. Once the mine is developed and built there are significant opportunities for additional exploration adjacent to workings. The deposit has been cut-off arbitrarily at 1,250 meters vertical depth.

25.1.5 Metallurgical Processing

It is the opinion of the qualified person responsible for Section 13 of the technical report, Mr Gordon Cunningham, that sufficient test work to support the Waterberg Platinum pre-feasibility has been undertaken.

Bench scale test work conducted, on each of the Waterberg lithologies as well as blends of the Waterberg ores, has demonstrated that a saleable concentrate containing at least 80 g/t (3E+Au) can be produced by applying a MF2 flowsheet which utilizes standard Southern African PGM reagents. No deleterious elements are expected whilst 3E+Au recoveries in excess of 80% is predicted.

25.1.5.1 Risks and Opportunities

The presence of coarse nuggets, most noticeably gold and palladium, has introduced some uncertainty in assaying, although adequate measures have been taken by the metallurgical testing facility and process team to verify and validate metal recoveries. Test work results have indicated that a fair degree of scatter is to be expected around the recovery estimates provided. Future feasibility level test work will aim to describe the metallurgical response variability (on specific ore types and ore blends) across the orebody, with specific reference to comminution and flotation variability.

Extensive metallurgical test work has been conducted on two different flowsheets, namely the MF1 and MF2 flowsheets, with encouraging results obtained from both. Test results have demonstrated that some of the ore types respond better to a particular configuration. Should the mining strategy, and practicality thereof, lend itself to delivering discrete ore types to the process plant then additional flexibility and optimisation opportunities could be realised. There is opportunity to take advantage of the lower capital cost and operating cost potential on the MF1 flowsheet, should F-Central material be mined discretely and processed through one of the concentrator plant modules proposed for the study.

The earthworks and civils engineering design assumes that fill material and aggregate for concrete can be readily obtained from the towns in the vicinity of the site. Should further investigation identify a source of material that can be quarried closer to the site this would result in capital cost savings.

25.1.6 Marketing and Contracts

The Waterberg project will produce a flotation concentrate from the processing plant, which is assumed to be sold, or toll treated into the local South African market.

Based on the project revenue calculations the major revenue contributor in excess of 85% is from the contained PGE's with the remainder being copper and nickel.

Production of up to 285 000 tonnes of concentrate per annum will be available at peak production. The concentrate will contain approximately 80 g/t 4E's plus copper at between 1% and 9.2% and nickel at between 1.1% and 5%. The concentrate does not contain any penalty elements such as chrome and is rich in sulphur, thus making it a desirable concentrate to blend with other high chrome concentrates.

No formal marketing studies have been conducted for this study nor have the local smelter and refinery operators been formally contacted to understand the appetite in the local industry to treat the concentrate to be produced from the project. Informal discussions have indicated interest in the product and based on the specifications it should be attractive.

Based upon industry data, it is expected that the payability for the concentrate sold to a local smelter operator will be up to 85% for the PGE's, 73% for contained copper and 68% for contained nickel. It is assumed that the option of taking payment advances will be exercised. Agreement against payment advances on concentrate sales ahead of the settlement dates is contained in the agreement.

25.1.6.1 Risks and Opportunities

The volume of concentrate to be produced from the Waterberg project will place a significant load on the available 'in country' smelting capacity without considerable expansion of the capacity. This may result in the local smelter capacity being unable to consume the Waterberg production, depending upon the status of the existing platinum industry and other expansion projects at the time. It is maybethat the concentrate production will be shared between a number of smelter, and this is the basis of the study.

The treatment terms and conditions have been assumed to be similar to the agreements published for PTM's Maseve project, but this may not be the case when negotiations have commenced for a formal off-take agreement.

The desirability of the concentrate being similar to the Merensky in terms of Sulphur content and having no Chrome represents and opportunity for a desirable off-take agreement.

There is a significant opportunity for the construction of a dedicated project specific smelting facility to process the production from Waterberg, which could also be a commercial facility toll treating concentrate."

25.1.7 Infrastructure

For the purposes of this PFS, a range of options were considered for the on-site and regional infrastructure.

The main infrastructure requirements for the Waterberg Project are access roads, residue disposal, water management, power supply and process plant to service and treat the targeted mine production.

It can be concluded that the availability of skilled labour resources, for both construction and operational phases, is limited and that the training and skills development program will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Waterberg Project.

25.1.7.1 Residue Storage Facility

The following conclusions were drawn from the study:

- Paste disposal is untested in the platinum industry and would pose a significant risk and require an extensive testing regime to consider implementing;
- Dry Stacking is a possible option and the potential water recoveries could make this option feasible, however the high capital and operational costs associated with dry stacking could make this potion infeasible compared to a conventional tailings dam;
- Cycloned tailings may provide a cost saving due to the higher rates of rise achievable, however test work is required prior to recommending this option;

- Conventional/thickened tailings is the safest option and well understood in the platinum industry and has been regarded as the preferred option for Waterberg

25.1.7.2 Risks and Opportunities

Some possible project risks associated with the current RDF design are as follows:

- A geotechnical investigation has not yet been undertaken on the soils underlying the RDF, therefore it is not known whether there could be problem soils in the area, such that an alternative site may be required;

- No stability analyses have been undertaken in this phase and are recommended in the next phase of the project. If stability is found to be an issue the configuration of the facility would need to be reassessed;

- A hydrology study has not been undertaken and will be required in order to determine any flood lines that may affect the facility;

- Construction rates are based on Epoch's past projects and are not specific to Waterberg. Acquiring construction rates from a contractor is required in the next phase of the project; and

- The facility is upstream of a river and homesteads, which means that the facility will be ranked as a high risk facility.

The possible opportunities associated with the current RDF design are as follows:

- The potential relaxation of the liner requirements may be possible based on a letter from the DWS which could reduce the costs associated with the liner;

- It may be possible to convert the RDF to a cyclone facility if the material characteristics are suitable. This would require further test work, however this could reduce the footprint area of the facility, thereby reducing costs associated with the liner; and

- Additional capacity is available if the need arises for a greater storage volume.

25.1.8 Economic Outcome

25.1.8.1 Waterberg PFS 2016

The Waterberg 2016 PFS presents the Mineral Reserve for the current Phase of the Project development. Further work and studies should be undertaken to bring the project to a Feasibility Study level. Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of the expansion will be evaluated at a later date, and the decision to expand can be deferred or brought forward as markets dictate and funding permits.

25.1.8.2 Capital and Operating Costs

Unit rates together with cost drivers form the basis for generating the Waterberg project operating cost estimates.

Activity based and first principle costing was utilised for the labour sub-model, lifecycle costing for all equipment, infrastructure and fleet; with the remainder being fixed/variable or zero-based costing.

The Operating cost model was constructed in such a way as to facilitate the reporting and analysis of the OPEX estimate model as a total cost profile and unit cost profile, with respective breakdowns which shall exclude reporting by fixed / variable and responsibility.

25.1.8.3 Capital and Operating Cost Risks and Opportunities

Some possible project risks associated with the current operating costs are as follows:

- Under budget due to:
    - Budget / Supplier prices outdated or escalated incorrectly
    - Exchange rate impacts imported equipment / consumables or portion thereof affecting price.
    - Quantity — Engineers miscalculated or under designed
    - Incomplete costing (leaving out items from budget)
    - Labour cost - can be higher than budgeted if location is remote; if market is competitive, undersupply of skilled labour
    - Increase in utilities cost — scarcity of water in area, shortage of power supply
    - Increase in fuel cost — remote location

Leading to a positive business case which is in actual fact not realise in practice

- Over budget due to:
    - Budget / Supplier prices outdated or escalated incorrectly
    - Exchange rate impacts imported equipment / consumables or portion thereof affecting price.
    - Quantity — Engineers miscalculated or under designed

Leading to a negative business case, resulting in an financial decision not being taken and an potential value adding opportunity lost

Some possible project opportunities associated with the current operating costs are as follows:

- Similar mines in area, client involved with similar operations,
    - Better benchmarking
    - Can negotiate price reductions
    - More contractor competition
- Management streamlining
- Implement proper maintenance philosophies to minimise breakdowns/failures.
- Create spare storage capacity for fuel and explosives to reduce trip frequency
- Standardisation of parts or equipment
- Economies of scale
- Improve operation efficiencies to reduce cost

- Introduce energy and water efficiency initiatives / incorporate into designs
- Local smelting options may be developed.

25.1.9 Environmental

Some possible project risks associated with the Environmental aspects of the PFS are as follows:

25.1.9.1 Risks

- Mining activities could affect local groundwater flow due to groundwater abstraction activities which could lower the water table, affecting local wells. This would require mitigation as part of the SLP.
- The natural landscape of the area will be significantly disrupted through the establishment of the mine. The visual and landscape impacts will be significant for the adjacent villages. The visual impacts of the underground access, plant, waste rock dumps and RDF will be significant and permanent. As a result of mining activity, vegetation will be cleared, large industrial structures will be built and vehicles and earth moving equipment will become familiar in the landscape. Thus, the aesthetics of the project area will change due to the mine and associated infrastructure.
- The establishment of a mine results in vegetation being cleared in the mine path and adjacent areas for secondary infrastructure. In this instance it will result in the removal of topsoil together will all associated vegetation.
- Similarly, any watercourse/drainage lines impacted by mining operations is likely to have a permanent and irreversible impact on the pre-existing hydrological function, although it is possible that final landform rehabilitation can replicate its basic function successfully, it will be difficult to do so.
- There is an inherent concern that villagers' sacred sites, some of which are located inside the mine's proposed area of influence (and especially on the mountains) might be disturbed. Respecting villagers and their traditional beliefs is also to value this privacy and concealment.
- Rural communities in South Africa place high importance on cultural heritage, including graves. The physical removal or relocation of graves is a sensitive impact. Social and Community

25.1.10 Social and Community

- The farms that are directly affected by the mine operations are Ketting 368LR Disseldorp 369LR, Goedetrouw 366 LR and Early Dawn 361LR. It is envisaged the most infrastructure and mine development will be in the Ketting farm.
- All the communities will get direct benefits in the development of the farm in terms of economic benefits and employment.
- At present, there is no economic benefit in these areas. Most of the community members are employed in cities far away from their homes. The area is rural and not developed at all.

25.1.10.1 Risks and Opportunities

Some possible project risks associated with the Social and Community are as follows:

- Securing land tenure by either a long lease or outright purchase of the farm Ketting that there are multiple owners of the land is poses a risk. New housing will be required.
- Unemployment and level of poverty is very high in the area, this can cause instability and this will require urgent intervention.

26. Recommendations

The QP's recommend that the Waterberg project advance to the DFS stage. The project financial model, including low capital cost per annual ounce of production and low operating costs provides the basis for further investment and refinement of the project design. The QP's recommend that based on the large scale PGM production profile of the project at 744,000 4E ounces per year that the project owners initiate discussions with smelters an investigate a standalone smelting option. The QP's also recommend that the owners initiate work towards an application for a Mining Right including the development of a Social and Labour Plan and environmental permits.

26.1 Recommended Work Programs

The QPs recommendations for the Waterberg Project and regional work are described below:

26.1.1 Geology

It is recommended that exploration drilling continues in order to advance the geological confidence in the deposit through infill drilling. This will provide more data for detailed logging and refined modelling. This is expected to confirm the geological continuity and allow the declaration of further Indicated Mineral Resources and the investigation for Measured resources for the DFS.

Given the results of the diamond drilling on the northern area and the extent of target areas generated by geophysical surveys, the completion of the planned exploration drilling is recommended north of the location of the current exploration programme. The objective of the exploration drilling would be to find the limit of the current deposit, confirm the understanding of the F Zone and T Zone and allow appropriate selection of the potential mining cut. This will improve geoscientific confidence in order to upgrade mineral resources to a Measured Category for consideration in the DFS. Hydrological Drilling.

26.1.2 Hydrological Drilling

The Hydrogeological evaluation will consist of the following:

Information from the Department of Water and Sanitation (DWS) Groundwater Database together with other available information from the Project area and information to be collected from a borehole census and water sampling programme, will be used to evaluate the groundwater resources available for the planned project. The evaluation will be done in 2 Phases as follows, viz:

26.1.2.1 Phase 1 - Target area identification

- Desk study collating available information within a 50 km radius
- Identify all existing users.
- Evaluation of all available data to determine the existing developed groundwater resources and the potential for further development in the area.
- Identify existing boreholes with poor water quality which are not suitable for domestic use but which could be utilised for plant supply
- Evaluate the geology and structural geology to determine target areas for new boreholes

26.1.2.2 Phase 2 - Drilling, testing and resource evaluation

- Conduct field mapping and geophysical surveys in target areas to pinpoint exploration drill sites (5 sites with 5 monitor boreholes)
- Drilling supervision of borehole and borehole design
- Testing of the newly drilled boreholes
- Water quality sampling and analysis of results
- Compile all info into a technical report giving the development potential and recommendations for wellfields

26.1.3 Geotechnical and Rock Engineering

26.1.3.1 Geotechnical

During the Definitive Feasibility Study, an appropriate Factor of Safety (FoS) will be determined according to international standards and our professional experience in similar ground conditions. The future site geotechnical investigation will allow getting all the necessary soil/rock parameters for a slope stability analysis to be conducted with the finite element and limit equilibrium methods.

Additional trial pits should be excavated at the exact positions of the proposed structures during the Definitive Feasibility Study at the next stage. A diamond drilled triple tubes borehole should be undertaken at each surface crusher up to a depth of 45 m or 10 m into medium hard rock sandstone or stronger (>25MPa). Appropriate soil and rock laboratory testing should be part of the geotechnical investigation at this stage, including falling head permeability test of the insitu material for the clay/geosynthetic liner of the tailing dam.

26.1.3.2 Rock Engineering

26.1.3.2.1 Site Investigation Work

To better inform design during the next stage of study it is recommended that a geotechnical drilling and logging programme be designed and implemented before the commencement of a feasibility study:

- Additional drilling focus is necessary at the T reef;
- Orientated core drilling and Geotechnical logging of joint properties to allow for joint stability analysis;
- Dedicated drilling and geotechnical core logging aimed at:
  - The portal locations;
  - The main decline routes;
  - The raise bore locations; and
  - Underground shaft infrastructure and other large chambers.
- Structural geology model (3D) is to be compiled showing prominent faults, dykes and other major structures. A better interpretation of the observed olivine-rich shear zones is required. This would be the responsibility of a suitably qualified Geologist;
- Geohydrological studies should be conducted to assess the potential for water inflow;
- Stress measurements to be conducted with specific focus on determining relevant K ratios. Core can be selected and sent for Acoustic Emissions (AE) testing.

26.1.3.2.2 Analysis and Numerical modelling

Basic assessments that are adequate for this level of study have been conducted but further numerical assessment in the following areas is required:

- The 3D block model should be updated with as much information as possible to cover at least all the areas where infrastructure might be located in order for appropriate design of these excavations and their ground support requirements;
- Structure related stability assessments (kinematic analysis at key LoM locations) and impact on support requirements. This relies on dedicated and directed orientated core drilling and geotechnical logging;
- Re assess the BLR and SLOS layouts using the updated geotechnical understanding;
- Case histories of design with the transitional zone. Focus on the issue of stand-up times and the risk of premature failure; and
- Additional assessment raise bore stability based on dedicated center line boreholes.

26.1.3.2.3 Operational systems

In stope ground control strategies, including monitoring of ground movements and stope evacuation procedures, require more intensive investigation.

26.1.4 Mining

For the Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- The mine design of underground access infrastructure, other underground excavations and production areas should be prepared to higher level of confidence;
- Scheduling rates for development and production should be revisited to ensure that the rates planned remain realistic and achievable.
- Compile a detailed Bill of Quantities of the mine design and involve relevant mining contracting companies so that accurate cost estimates can be prepared.
- Conduct a simulation exercise that considers all underground logistics. It is recommended that this be done using an appropriate software package.
- Review the risks mentioned so that where possible adequate mitigating factors can be incorporated into the mine design and schedule.
- Optimization of dilution and waste development for ventilation.

It is recommended that the opportunities mentioned above be investigated further. This could be done prior to the next phase of the study or at least during the next study phase.

26.1.5 Metallurgical Processing

It is recommended that the concentrator plant be advanced to a Definitive Feasibility Study phase.

The following metallurgical test work is recommended for the next phase of the project:

- Flotation test work using water from the envisaged raw water sources to ensure the flotation performance is not negatively affected.
- Testing of the MF2 circuit using an Oxalic acid and Thiourea reagent scheme
- Comminution variability test work on the individual ore types

- Comminution variability test work on various possible mine blends
- Flotation open circuit batch variability test work on the individual ore types
- Flotation open circuit batch variability test work on various possible mine blends
- Concentrate thickening and filtration test work

Further geotechnical investigations of the plant site is recommended to determine the local founding conditions.

A more detailed topographical survey of the plant area is also recommended.

26.1.6 Marketing Studies and Contracts

It is recommended that the local smelter operators be formally approached to better understand the appetite to consume the significant concentrate production once the mine is at steady state.

In addition, during the Definitive Feasibility Study, it is recommended that a Scoping Study be completed into the potential for the inclusion of a Waterberg Project Smelter on site. The product from this smelter could be a furnace matte or a convertor matte, which could be treated locally or exported for refining.

26.1.7 Infrastructure

26.1.7.1 Residue Disposal Facility

For the Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- A geotechnical investigation of the RDF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials;
- The potential relaxation of the liner requirements may be possible based on a letter from the DWS which could reduce the costs associated with the liner;
- It may be possible to convert the RDF to a cyclone facility if the material characteristics are suitable. This would require further test work, however this could reduce the footprint area of the facility, thereby reducing costs associated with the liner
- A seepage analysis and slope stability study be undertaken to confirm the seepage regimes through the RDF as well as to confirm the RDF stability. The results of these analyses could impact greatly on the geometry of the RDF walls and ultimate height of the facility;
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample;
- Possible further optimisation of the RDF preparatory works in terms of layout, footprint extent, etc. including any changes to the mine plan;
- Review the construction rates with a contractor to price the facility with representative rates;
- Compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy; and
- A hydrological study of potential flood lines near the RDF

26.1.7.2 Bulk Water Supply

Due to the scarcity of water in the area, it will be critical to conduct more detailed hydrogeological investigations in order to identify potential groundwater resources that can be developed for mine supply and to predict the mine inflows and impact zone accurately. This will also be important to determine external bulk water requirements and the timing thereof. These hydrogeological investigations should include a numerical model, which will also assist the mine with monitoring and water management during the life of mine.

26.1.7.3 Bulk Power Supply

It is recommended that the existing 22kV lines be upgraded to service the project constructionperiod whilst awaiting the permanent 132kV line and permanent supply infrastructure construction.

26.1.8 Environmental

PTM has a programme of work in place to comply with the necessary environmental, social and community requirements.

Key work should continue to include:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA;
- Public Participation Process (PPP) in accordance with the NEMA Guidelines;
- Specialist investigations in support of the ESIA;
- Integrated Water Use License Application (IWULA) in compliance with the National Water Act (NWA); and
- Integrated Waste Management License in compliance with the National Environmental Management Waste Act (NEMWA).

26.1.9 Social and Community

The community and Social impact studies that are underway must be compiled as soon as possible. The compilation of the SLP should also be conducted during the DFS. This will inform which projects will form part of the local economic Development (LED) and the Human Resources Development (HRD), which are components of the social labor Plan (SLP) security and land tenure.

The first priority is to commence negotiations for the purchasing of the farm or alternatively along term lease agreement is recommended.

These procedures have to be compiled before the mining Right is obtained or granted.

27. References

- Bieniawski, Z.T. (1974). *Estimating the strength of rock materials* . SAIMM, 312-320.
- Barton, N., Lien, R., and Lunde, J. (1974). *Engineering classification of rock masses for the design of tunnel support* , Rock Mech. 6 (1974), pp. 188—236.
- Brown, E. (2000). *Block Caving Geomechanics* . Queensland: Figure 2 4.
- Hutchinson, J.D. and Diederichs, M.S. (1996). *Cablebolting in Underground Mines* , Bitech Publishers, Richmond, BC, 1996.
- Jager, A. J., & Ryder, J. A. (1999). *A Handbook on Rock Engineering Practice for Tabular Hard Rock Mines* . Johannesburg: SIMRAC.
- Laubscher, D.H. (1990). *A geomechanics classification system for the rating of rock mass in mine design* . Johannesburg: J.S. Afr. Inst. Min. Metall., vol. 90, no. 10. pp.257-273.
- Mathews, K.E., Hoek, E., Wyllie D.C., and Stewart, S.B.V. (1981). *Prediction of stable excavation spans at depths below 1000m in hard rock mines* , CANMET Report, DSS Serial No. OSQ80-00081, 1981.
- NGI (2015). *Using the Q-system Rock mass classification and support design* , NGI, Oslo, May 2015.
- McCracken, A., & Stacey, T. R. (1989). *Geotechnical risk assesment for large diameter raise-bored shafts* . Trans. Instn Min. Metall. (Sect. A Min. industry), A145 - A150.
- Nickson S. D. (1992). *Cable support guidelines for underground hard rock mine operations* , M.Sc. thesis, University of British Columbia, 1992.
- Potvin Y. (1988). *Empirical open stope design in Canada* , Ph.D. thesis, University of British Columbia, 1988.
- RocScience. (n.d.). *Rock Mass Classification* . Rocscience.
- Suorineni, F.T., Kaiser, P. K. and Hudson, J. G (2001). *Narrow-vein Mining Experiences at the Ashanti Mine: A Case Study.* Symposium on Narrow-vein Deposits, Val d'Or, Quebec, October 2-4, 2001
- WorleyParsons Reports
  - Balt, K. (2016). *Rock Mechanics study for the Waterberg PFS* . PFS Report, Johannesburg, Advision.
  - Louw, G. (2016). *Ventilation Shaft Raise Bore Analysis* . Internal Report Johannesburg, Advision.
  - Louw, G., & Shangase, N. (2016). *Geotechnical logging report.* Internal Report, Doc No: *C00458-1000-MG-REP-0001* , Johannesburg, Advision.
  - Shangase, N. and Thompson J. (2015). *Geotechnical review of rock mass conditions towards the pre-feasibility study* . Internal report, Doc No: C00333-000-GM-REP-0001, Johannesburg, WorleyParsonsRSA.
- Metallurgical Test Work Reports
  - Ramlall, N., Mosinyi, D. and Moropa, T. (2013). Scoping Test work on PGM Samples from the Waterberg Platinum Project. Mineralogical Report No: 12/346 Rev. 2. SGS (Booysens).

- JOGMEC Metals Mining Technology Department (2014). Results of the Flotation Tests on F-zone from the Waterberg Deposit. Hand-out.
- Sindane, Z. and Mothupi, T. (2015). Flotation Concentrate Production for Initial Off-take Discussions on Samples from the Waterberg Deposit in the Bushveld Complex. Report No: To be issued. MINTEK, Johannesburg.
- Moodley, T.L. (2016). Concentrator Flowsheet Development Work for the PTM Waterberg Project on the Northern Limb of the Bushveld Igneous Complex, South Africa, Phase 1.. Report No: To be issued. MINTEK, Johannesburg.
- Moodley, T.L. and Nembudani, P. (2016). Waterberg Project — Phase 2: MF-1 Circuit Collector Optimisation Work. Report No: To be issued. MINTEK, Johannesburg.
- Moodley, T.L. (2016). Optimisation Work on the Waterberg Super F North Sample as Part of the Waterberg Pre-Feasibility Study — Phase 3. Report No: To be issued. MINTEK, Johannesburg.
- Moodley, T.L. (2016). Optimisation Work on the Waterberg T-zone South Sample as Part of the Waterberg Pre-Feasibility Study — Phase 4. Report No: To be issued. MINTEK, Johannesburg.
- Moodley, T.L. (2016). Extension of Optimisation Work on the Waterberg T-zone South and F-Boundary Samples as Part of the Waterberg Pre-Feasibility Study — Phase 4B. Report No: To be issued. MINTEK, Johannesburg.
- VIETTI Slurrytec. (2015). Waterberg Tailings Trade Off Study, Thickening Test Work Report. Report No: DRA-12-8435 R01 Rev 1.
- VIETTI Slurrytec. (2015). Waterberg Tailings Trade Off Study, Filtration Test Work Report. Report No: DRA-12-8435 R02 Rev 0.
- PATERSON & COOKE. (2015). PGM Waterberg Pre-Feasibility Study, Slurry Test Report. Report No: DRA-12-8435-00-TW-REP-0001 Rev 1.

28. Date and Signature Page

This technical report entitled "Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study" with the effective date of 17 October 2016 was prepared on behalf of Platinum Group Metals (RSA) (Pty) Ltd by the following Qualified Persons and signed.

Dated in Johannesburg, this 19 th day of October, 2016.

"Charles Johannes Muller"

CHARLES JOHANNES MULLER
B.Sc (Hons), Pr. Sci. Nat

"Robert Leon Goosen"

ROBERT LEON GOOSEN
B. Eng. (Mining Engineering), Pr. Eng

"Gordon Ian Cunningham"

GORDON IAN CUNNINGHAM
B. Eng. (Chemical), Pr. Eng., FSAIMM

CONSENT OF EXPERT

The undersigned, on his own behalf and on behalf of Turnberry Projects (Pty) Ltd., hereby consents to the references to, the information derived from, and the incorporation by reference of the report titled "Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study", dated October 19, 2016 and to the references, as applicable, to the undersigned's name and to Turnberry Projects (Pty) Ltd. included in or incorporated by reference in the Registration Statement on Form F-10 filed by Platinum Group Metals Ltd. (File No. 333-213985), and any amendments or supplements thereto and any registration statements filed pursuant to Rule 429 under the United States Securities Act of 1933, as amended.

"Gordon Cunningham"

Gordon Cunningham
Turnberry Projects (Pty) Ltd.
Date: October 19, 2016

CONSENT OF EXPERT

The undersigned, on his own behalf and on behalf of Advisian/WorleyParsons Group, hereby consents to the references to, the information derived from, and the incorporation by reference of the report titled "Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study", dated October 19, 2016 and to the references, as applicable, to the undersigned's name and to Advisian/WorleyParsons Group included in or incorporated by reference in the Registration Statement on Form F-10 filed by Platinum Group Metals Ltd. (File No. 333-213985), and any amendments or supplements thereto and any registration statements filed pursuant to Rule 429 under the United States Securities Act of 1933, as amended.

"Robert Goosen"

Robert Goosen
Advisian/WorleyParsons Group
Date: October 19, 2016

Exhibit 99.6

CONSENT OF EXPERT

The undersigned, on his own behalf and on behalf of CJM Consulting (Pty) Ltd., hereby consents to the references to, the information derived from, and the incorporation by reference of the report titled "Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study", dated October 19, 2016 and to the references, as applicable, to the undersigned's name and to CJM Consulting (Pty) Ltd. included in or incorporated by reference in the Registration Statement on Form F-10 filed by Platinum Group Metals Ltd. (File No. 333-213985), and any amendments or supplements thereto and any registration statements filed pursuant to Rule 429 under the United States Securities Act of 1933, as amended.

"Charles Muller"

Charles Muller
CJM Consulting (Pty) Ltd.
Date: October 19, 2016

Exhibit 99.7

CONSENT OF EXPERT

The undersigned hereby consents to all references to him as a non-independent qualified person included in or incorporated by reference in the Registration Statement on Form F-10 filed by Platinum Group Metals Ltd. (File No. 333-213985), and any amendments or supplements thereto and any registration statements filed pursuant to Rule 429 under the United States Securities Act of 1933, as amended.

"R. Michael Jones"

R. Michael Jones
Date: October 19, 2016

Exhibit 99.8

Certificate of Qualified Person

I, Gordon Ian Cunningham, B. Eng. (Chemical), Pr. Eng., FSAIMM, residing at Sixth Avenue, Melville, South Africa hereby certify that:

1. I am currently employed as a Director and Principal Metallurgical Engineer by Turnberry Projects (Pty) Ltd. of PO Box 2199 Rivonia, Sandton, 2128, South Africa.

2. I am a member in good standing of the Engineering Council of South Africa and am registered as a Professional Engineer — Registration No. 920082. I am a Fellow in good standing of the South African Institute of Mining and Metallurgy — Membership No. 19584.

3. This certificate applies to the technical report titled "Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study" dated effective 17 October, 2016 (the "Report").

4. I graduated from the University of Queensland with a B. Eng. (Chemical) in 1975. I have worked as a Metallurgist in production for a total of 20 years since my graduation. I have worked as a Consulting Metallurgist for 5 years since graduation and have been working for Turnberry Projects for 16 years as a Project and Principal Engineer and Director, primarily associated with mining and metallurgy projects.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of the Instrument.

6. Prior to co-authoring this Report in respect of the Waterberg Project, I have had minimal prior involvement with the Waterberg Project. I was involved with a review of the preliminary economic assessment prepared in 2014 (the "PEA") in respect of the Waterberg Project but did not serve as a qualified person for the PEA.

7. I most recently conducted a personal inspection of the Waterberg Project on 13 October 2016 for a period of one day.

8. I am the co-author of this Report and am responsible for Sections 13, 17, 19, 20, 21 and 22 and am jointly responsible for Sections 1, 2, 3, 18, 24, 25, 26 and 27.

9. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission of which would make the Report misleading.

10. I am independent of Platinum Group Metals Ltd. as described in section 1.5 of the Instrument.

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Director: G.I.Cunningham

11. I have read the Instrument and confirm that the parts of the Report that I am responsible for have been prepared in compliance with the Instrument.

12. I do not have nor do I expect to receive a direct or indirect interest in the mineral properties of Platinum Group Metals Ltd. and I do not beneficially own, directly or indirectly, any securities of Platinum Group Metals Ltd. or any associate or affiliate of such company.

13. At the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Johannesburg, South Africa, on October 19, 2016.

"Gordon Ian Cunningham"

Gordon Ian Cunningham
Director
Turnberry Projects (Pty) Ltd

Exhibit 99.9

Certificate of Qualified Person

I, Robert Leon Goosen, B.Eng. (Mining Engineering), Pr. Eng., residing at 448 Rooibekkie Crescent, Featherbrooke Estate, Krugersdorp, South Africa hereby certify that:

1. I am currently a Principal Consultant — Mining Engineering with the firm of Advisian/WorleyParsons Group of 39 Melrose Boulevard, Melrose Arch, 2076, Johannesburg, South Africa.

2. I am a practising mining engineering consultant and a registered professional engineer with the Engineering Council of South Africa (ECSA). I am also a member of the Association of Mine Managers of South Africa (AMMSA).

3. This certificate applies to the technical report titled “Independent Technical Report for the Waterberg Project including Resource Update and Pre-Feasibility Study” dated effective October 17, 2016 (the “Report”).

4. I graduated from the University of Pretoria with a B.Eng. (Mining Engineering) degree in 1993. I obtained my Mine Manager’s Certificate of Competency in 1996. I have practiced my profession since 1993. I have more than twenty years of mining experience including +15 years in mining production and project execution, the majority of this with hard rock precious metals mines in South Africa. I have been working for Advisian/WorleyParsons Group for 5 years as a consulting mining engineer.

5. I have read the definition of “qualified person” set out in National Instrument 43-101 (the “Instrument”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of the Instrument.

6. Prior to co-authoring this Report in respect of the Waterberg Project, I have had no prior involvement with the Waterberg Project.

7. I have visited the Waterberg Project for personal inspection during the project phase on 13 October 2016 for a period of one day.

8. I am the co-author of this report and am responsible for reviewing Sections 4, 5, 15, 16 and 23 and am jointly responsible for sections 1, 2, 3, 6, 18, 24, 25, 26 and 27.

9. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission of which would make the Report misleading.

10. I am independent of Platinum Group Metals Ltd. as described in section 1.5 of the Instrument.

11. I have read the Instrument and confirm that the parts of the Report that I am responsible for have been prepared in compliance with the Instrument.

12. I do not have nor do I expect to receive a direct or indirect interest in the mineral properties of Platinum Group Metals Ltd. and I do not beneficially own, directly or indirectly, any securities of Platinum Group Metals Ltd. or any associate or affiliate of such company.

13. At the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Johannesburg, South Africa, on October 19, 2016.

"Robert Leon Goosen"

Robert Leon Goosen
Principal Consultant — Mining Engineering
Advisian/WorleyParsons Group

Certificate of Qualified Person

As an author of the technical report titled "Independent Technical Report on the Waterberg Project including Mineral Resource Update and Pre-Feasibility Study" dated effective 17 October, 2016 (the "Report") I, Charles Johannes Muller, Pr.Sci. Nat residing at Roodepoort, South Africa hereby certify:

1. I am a senior Mineral Resource consultant with the firm of CJM Consulting of Ruimsig Office Estate, 199 Hole-in-one Road, Ruimsig, and Roodepoort, South Africa.

2. I am a practicing Mineral Resource consultant and a registered professional scientist with the South African Council for Natural Scientific Professions, Pr. Sci. Nat. Reg. No. 400201/04.

3. I graduated with a B.Sc. (Geology) degree from the Rand Afrikaans University — Johannesburg in 1988. In addition, I have obtained a B.Sc. Hons (Geology) from Rand Afrikaans University in 1994 and attended courses in geostatistics through the University of the Witwatersrand.

4. I have worked as a Mineral Resource geologist for 22 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of the Instrument.

6. I have previously authored the technical report titled "An Independent Technical Report on the Waterberg Project Located in the Bushveld Igneous Complex, South Africa" dated effective July 20, 2015 and the technical report titled "Mineral Resource Update on the Waterberg Project Located in the Bushveld Igneous Complex, South Africa" dated effective April 29, 2016.

7. I have last visited the Waterberg Project for personal inspection on 29 March, 2016 for a period of two days.

8. I am the co-author of this report and am responsible for Sections 7, 8, 9, 10, 11, 12 and 14, and am jointly responsible for sections 1, 2, 3, 6, 24, 25, 26 and 27.

9. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission of which would make the Report misleading.

10. I am independent of Platinum Group Metals Ltd. as described in section 1.5 of the Instrument.

11. I have read the Instrument and confirm that the parts of the Report that I am responsible for have been prepared in compliance with the Instrument.

12. I do not have nor do I expect to receive a direct or indirect interest in the mineral properties of Platinum Group Metals Ltd. and I do not beneficially own, directly or indirectly, any securities of Platinum Group Metals Ltd. or any associate or affiliate of such company.

13. At the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Report not misleading.

14. Dated at Roodepoort, South Africa, on 19 October 2016.

"Charles J Muller"

CHARLES J MULLER, Pr. Sci. Nat.
DIRECTOR — MINERAL RESOURCES
CJM CONSULTING (Pty) Ltd