Independent Technical Report

Waterberg Project Definitive Feasibility Study and Mineral Resource Update

Bushveld Complex, South Africa

Effective Date of Resource: 04 September 2019
Effective Date of Reserve: 04 September 2019
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TITLE PAGE

Report Title: ***Independent Technical Report – Waterberg Project Mineral Resource Update and Definitive Feasibiity Study***

Property: **Waterberg Project**

Location: **Bushveld Complex, South Africa**

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Effective Date of Mineral Reserve: **04 September 2019**

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IMPORTANT NOTICE

This report was prepared as a Technical Report, in accordance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects, for Waterberg JV Resources (Pty) Ltd.

The Technical Report is based on information and data supplied to the Report Authors by Waterberg Joint Venture (JV) Resources (Pty) Ltd. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of the Report Authors, based on: i) information available at the time of preparation of the Report, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report.

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DATE AND SIGNATURE PAGE

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FREQUENTLY USED ACRONYMS, ABBREVIATIONS, DEFINITIONS AND UNITS OF MEASURE

Acronyms

Following are acronyms and abbreviations used in the Waterberg Project Technical Report.

3D	three dimensional
4E	platinum, palladium, rhodium and gold
A	
Ag	silver
Ai	abrasion index
Al	aluminium
AMEC	AMEC GRD SA (Netherlands)
As	arsenic
Au	gold
B	
Ba	barium
BAC	bulk-air cooler
BBE	Bluhm Burton Engineering
BBWi	bond ball work index
BE	Bateleur Environmental & Monitoring Services
BEE	Black Economic Empowerment
BOQ	bill of quantity
BRWi	bond rod work index
C	
Ca	calcium
CapEx	capital expenditure
Cd	cadmium
Ce	cerium
CIM	Canadian Institute of Mining
CJM	CJM Consulting (South Africa) Pty Limited
Co	cobalt
Cr	chromium
CRM	certified reference material
Cs	caesium
Cu	copper
CWi	bond crushability work index
D	
DBM	drill-blast-muck
DFS	Definitive Feasibility Study
DMR	Department of Mineral Resources
DWi	drop weight index
E	
EA	Environmental Authorisation
EBIT	earnings before interest and taxes
EIA	Environmental Impact Assessment
EMP	Environmental Management Plan
EMPr	Environmental Management Programme
Mo	molybdenum
MASL	metres above sea level
MPRDA	Mineral and Petroleum Resources Development Act
MPTO	Mineral and Petroleum Titles Office
MSO	Mineable Shape Optimiser
MTO	material takeoff
N	
Nb	niobium
ND	not determined
NEMA	National Environmental Management Act
Ni	nickel
O	
OK	ordinary kriging
OpEx	operating expenditure
P	
P&G	preliminary and general
Pb	lead
Pd	palladium
PEA	Preliminary Economic Assessment
PFS	prefeasibility study
PGE	platinum group element
PGM	platinum group metals
PLC	power-line communication
PP plot	probability plot
PR	prospecting right
Pt	platinum
PTM RSA	Platinum Group Metals (RSA) (Pty) Ltd
PTM	Platinum Group Metals Ltd.
PTML	Platinum Group Metals (Pty) Ltd (Canada)
Q	
QA/QC	quality assurance and quality control
QP	qualified person
R	
Rb	rubidium
Rh	rhodium
ROM	run of mine
RQD	rock quality designation
RSA	Republic of South Africa
Ru	ruthenium
RWD	return water dam
S	
S	sulphur

EPCM	engineering, procurement, and construction management
Epoch	Epoch Resources (Pty) Ltd.
ESHIA	Environmental, Social, and Health Impact Assessment
F	
Fe	iron
FRSC	fibre-reinforced shotcrete
FZ_IFW	F Zone Immediate Footwall (0-5 m)
G	
G&A	general and administrative
Ga	gallium
GCL	Geosynthetic Clay Liner
Ge	germanium
GIS	geographic information system
H	
H	high / height
HLS	heavy liquid separation
I	
ICP	inductively coupled plasma
ICP/MS	inductively coupled plasma / mass spectrometry
ICP/OES	inductively coupled plasma / optical emission spectrometry
IEC	International Electrotechnical Commission
IMPLATS or Impala	Impala Platinum Holdings Limited
In	indium
Ir	iridium
ISO	International Standards Organization
ITH	in-the-hole
J	
Ja	joint alteration number
Jn	joint set number
JOGMEC	Japanese Oil and Metals National Corporation
Jr	joint roughness number
JV	joint venture
K	
K	potassium
L	
LCT	leachate concentration test
LHD	load haul dump
Li	lithium
LOM	life of mine
LPA	lower pegmatoidal anorthosite
LPP	lower pegmatoidal pyroxenite
LSLOS	longitudinal sublevel open stoping
M	
SAHRA	South African Heritage Resource Agency
SANAS	South African National Accreditation System
SAMREC	South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (2007)
Sb	antimony
SC	mesh plus shotcrete
SCADA	supervisory control and data acquisition
SD	standard deviation
Se	selenium
SG	specific gravity
SHEQ	safety, health, environmental, and quality
SIB	stay-in-business
SIBX	sodium isobutyl xanthate
SiO_2	silicon dioxide
SK	simple kriging
SLP	social and labour plan
SMC	SAG mill comminution
Sr	strontium
SS	split set
SSBS	Sustainable Slurry and Backfill Solutions
Stantec	Stantec – Mining
SWD	stormwater dam
T	
Ta	tantalum
TD	tailings dam
Th	thorium
Ti	titanium
Tl	thallium
TSF	tailings storage facility
U	
U	uranium
UCS	uniaxial compressive strength
UGR	upgrade ratio
UPA	Upper Pegmatiodal Anorthosite
US$	United States dollar
UTM	Universal Transverse Mercator
V	
V	vanadium
VFD	variable frequency drive
VIR	value-investment ratio
VOIP	voice over internet protocol
W	
W	wide
w/w	weight/weight

Ma	mega annum – a million years
MF1	mill-flotation circuit, single stage milling followed by flotation
MF2	mill-flotation-mill flotation circuit, two stage milling followed by a twin-stage flotation circuit
MgO	magnesium oxide
MSHA	Mine Safety and Health Administration
M&I	measured and indicated
Mn	manganese
Mnombo	Mnombo Wethu Consultants (Pty) Ltd.
Waterberg JV Resources	Waterberg JV Resources (Pty) Ltd
WBGT	wet-bulb globe temperature
WBS	work breakdown structure
WML	Waste Management License
WUL	Water Use License
Y	
Y	yttrium
Z	
ZAR	South African rand
Zn	zinc

Units of Measure

Following are units of measure used for the Waterberg Project.

º	degrees
ºC	degrees Celsius
ºF	degrees Fahrenheit
cm	centimetre
dtph	dry tonnes per hour
g/t	grams per tonne
ha	hectare
kgm^3	kilogram per cubic metre
km	kilometre
km^2	square kilometres
ktpa	kilo tonnes per annum
ktpm	kilo tonnes per month
kV	kilovolt
kVA	kilovolt-ampere
kVAhr	kilovolt-ampere hour
kW	kilowatt
kWhr	kilowatt hour
L/min	litres per minute
m	metre
m^3/s	cubic metres per second
Moz	million ounces
MPa	megapascal
MVA	megavolt amperes
MW	megawatt
MW_R	megawatt refrigeration
MWh	megawatt hour
Ø	diameter
ppb	parts per billion
ppm	parts per million
t	tonnes
t/m^3	tonnes per cubic metre
tpa	tonnes per annum
tph	tonnes per hour
tpm	tonnes per month

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1 SUMMARY

1.1 Introduction

This report was compiled for Waterberg Joint Venture (JV) Resources (Pty) Ltd. (Waterberg JV Resources), a company owned by Platinum Group Metals Ltd. (PTM), Impala Platinum (IMPLATS), Japan Oil, Gas and Metals National Corporation ("JOGMEC"), Hanwa Co. Ltd. ("Hanwa") and Mnobo Wethu Consultants (Pty) Ltd. ("Mnobo"). PTM is listed on the Toronto stock exchange under the symbol "PTM" and on the New York Stock Exchange under the symbol "PLG.A."

The purpose of this report is to provide an update to the Mineral Resource estimate, update to the Mineral Reserve, and publish the results of a definitive feasibility study (DFS) for the Waterberg Project. The Waterberg Project is the development of a platinum group metals (PGM) mine and Concentrator Plant in the Province of Limpopo, South Africa.

This report was prepared in accordance with disclosure and reporting requirements set forth in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), Companion Policy 43-101CP to NI 43-101, and Form 43-101F1 of NI 43-101.

The estimated Mineral Resources for the Waterberg Project at a 2.5 g/t platinum (Pt), palladium (Pd), rhodium (Rh), and gold (Au) (4E) cutoff grade include a combined 242.4 million tonnes at an average grade of 3.38 g/t 4E, 0.10% copper (Cu) and 0.18% nickel (Ni) in the measured and indicated (M&I) categories, and an additional 66.7 million tonnes at an average grade of 3.27 g/t 4E, 0.11% Cu, and 0.15% Ni in the inferred category.

The estimated Mineral Reserve for the Waterberg Project at a 2.5 g/t 4E cutoff grade includes a combined 187.5 million tonnes at an average grade of 3.24 g/t 4E, 0.09% Cu, and 0.18% Ni in the proven and probable categories. The estimated Mineral Reserves contains a total of 19.5 million ounces of Pd, Pt, Rh, and Au.

The key outcome of the DFS is the development of one of the largest and lowest cash cost underground PGM mines globally. The shallow, decline-accessed mine will be fully mechanized and produce approximately 4.8 million tonnes of ore and 420,000 combined ounces of Pd, Pt, Rh, and Au in concentrate per year at steady state. The mine will produce for approximately 45 years. Additional outcomes include:

- Estimated project capital of approximately R13.1 billion [United States dollar (US$)874 million] plus R3.5 billion in capitalized operating costs to achieve 70% of steady-state production.
- Peak funding of R9.26 billion (US$617 million).
- Payback period of approximately 11.4 years at 3-year average prices and 8.4 years at spot prices.
- After tax net present value (NPV) of R5.62 billion (US$333 million) at an 8% discount rate [three year average price US$931 per oz Pt, US$1 055 per oz Pd, US$1 930 per oz Rh, US$1 318 per

oz Au, US$2.87 per pound Cu and US$5.56 per pound Ni, US$/South African Rand (ZAR) 15.95].

- After tax NPV of R14.7 billion (US$982 million) at an 8% discount rate (spot prices 04 September 2019 - US$980 per oz Pt, US$1 546 per oz Pd, US$5 036 per oz Rh, US$1 548 per oz Au, US$2.56 per pound Cu and US$8.10 per pound Ni, US$/ZAR 15.00).
- After tax internal rate of return (IRR) of 13.3% (three year trailing average price).
- After tax IRR of 20.7% (Spot Prices 04 September 2019).

1.2 Property Description and Location

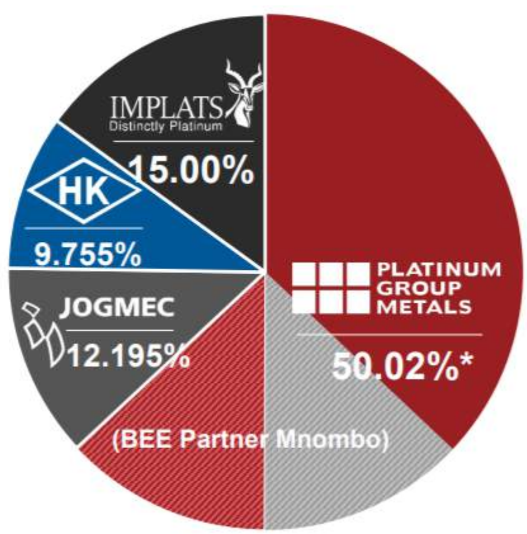
1.2.1 Property and Title

The Waterberg Project is located 85 km north of the town of Mokopane in the province of Limpopo, South Africa, approximately 330 km NNE from Johannesburg. The total project area, active prospecting rights (PRs), and mining right application area covers a total area of 99 244 hectare (ha). Elevation ranges from approximately 880 to 1 365 metres (m) above sea level.

1.2.2 Holdings Structure

Platinum Group Metals (RSA) (Pty) Ltd (PTM RSA) is the operator of the Waterberg Project, with JV partners being Japanese Oil, Gas and Metals National Corporation (JOGMEC), Hanwa Co. (Hanwa), Impala Platinum Holdings Ltd (IMPLATS) and Mnombo Wethu Consultants (Pty) Ltd. (Mnombo). Figure 1-1 shows the holdings of the Waterberg Project.

Figure 1-1: Waterberg Project Holdings



1.3 Geological Setting 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 66 000 km^2 in extent, of which about 55% is covered by younger formations. The Bushveld Complex hosts several layers rich in PGM, chromium (Cr) and vanadium (V), and constitutes the world's largest known Mineral Resources of these metals.

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

PGM mineralisation within the Bushveld package underlying Waterberg is hosted in two main layers: T Zone and 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 mineralised layers, three potential economical layers were identified, TZ, T1, and T0 - 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 (harzburgite) and FP (pyroxenite) layers. The FH layer has significantly higher volumes of olivine in contrast with the lower lying FP layer, which is predominately pyroxenite.

1.4 Deposit Types

The mineralised layers of the Waterberg Project meet some the criteria for Platreef-type deposits, where the mineralisation is hosted by sulphides that are magmatic in origin. The mineralised layers can be relatively thick, often greater than 10 m.

The other criteria relating to the Platreef have yet to be demonstrated. Consequently, this mineralisation is deemed to be similar, i.e. Platreef-like, but its stratigraphic position, geochemical and lithological profiles suggest a type of mineralisation not previously recognised in the Bushveld Complex.

1.5 Exploration Data / Information

The Waterberg Project is an advanced project that has undergone preliminary economic evaluations, a prefeasibility study (PFS) and resulted in this DFS. Drilling to date has given the confidence to classify Mineral Resources as inferred, indicated, and measured.

1.6 Drilling

The data from which the structure of the mineralised horizons were modelled and grade values estimated were derived from a total of 362 293 m of diamond drilling. This report updates the

Mineral Resource Estimate using this dataset. The drill hole dataset consists of 441 drill holes and 583 deflections at the date of drill data cutoff (01 December 2018).

The management of the drilling programmes, logging, and sampling were undertaken from multiple facilities: one at the town of Marken in Limpopo Province, South Africa, and the other on the farm Goedetrouw 366LR within the PR area, or at an exploration camp on the adjacent farm Harriet's Wish.

1.7 Sample Preparation, Analyses, and Security

The sampling methodology concurs with Waterberg JV Resources' 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 enough shoulder sampling to ensure the entire economic zones are assayed.

Waterberg JV Resources instituted a complete quality assurance / quality control (QA/QC) programme, including the insertion of blanks and certified reference materials as well as referee analyses. The programme is being followed and is to industry standard. The data is as a result, considered reliable in the opinion of the qualified person (QP).

1.8 Data Verification

Printed logs for 90% of the holes were checked with the drilled core. The depths of mineralisation, 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. The average specific gravity (SG) values were generated for each individual lithological type and missing SG values were inserted according to the lithological unit. Assay certificates were checked on a test basis. The data was reviewed for statistical anomalies.

The individuals in Waterberg JV Resources' senior management and certain directors of the company, who completed the tests and designed the processes, are non-independent mining or geological experts. The QP's opinion is that the data is adequate for use in Mineral Resource Estimation.

1.9 Mineral Processing and Metallurgical Testing

Metallurgical testing of the F Zone and T Zone on selected drill core samples was completed at accredited metallurgical laboratories in South Africa with all analyses being performed with appropriate QA/QC oversight. The economic minerals will be recovered by flotation techniques into a flotation concentrate suitable as feed stock to a smelter and followed by further downstream processing at a precious metals refinery, typical of the PGM industry.

The PFS programme selected the most appropriate metallurgical process for the optimized recovery of the 4E elements and the associate base metals and this was confirmed during the DFS variability and production blend evaluations.

The ore is hard and is not amenable to semi-autogenous milling; therefore, a three-stage crushing followed by two-stage ball milling circuit was selected for comminution.

The testwork programme was used to develop a grade-recovery relationship targeting 80 g/t 4E in the flotation concentrate as feed to a smelter. The concentrate is expected to contain 2.5% Cu and 2.7% Ni in addition to the contained 4E elements (Pt, Pd, Rh, and Au). The grade recovery relationship was developed for each of the six economic metals with 4Es at 81%, Cu at 82%, and Ni at 48% for the first 13 years of production with the corresponding life of mine recoveries being 79%, 83%, and 48%, respectively.

1.10 Mineral Resource Estimates

This report documents the Mineral Resource Estimate - effective date: 04 September 2019. Infill drilling over portions of the Waterberg Project area and new estimation methodology made it possible to estimate a new Mineral Resource Estimate and upgrade portions of the Mineral Resource to the measured category. All the JV partners were involved in the development of the latest Mineral Resource Model, appropriate cutoff grades, economic parameters, and Mineral Resource Model criteria. It was determined in relation to basic working costs and in consideration of the overall resource envelope for the deposit, that at a 2.0 g/t cutoff grade, the deposit has a reasonable prospect of economic extraction. The Mineral Resource Statement is summarised in Table 1-1. For purposes of the DFS, sensitivity analysis and comparison to the 2016 PFS, which utilised a 2.5 g/t Pt, Pd, Rh, Au for the (4E) cutoff grade, a Mineral Resource Estimate at a 2.5 g/t cutoff grade is the preferred scenario as shown in Table 1-2.

Table 1-1: Summary of Mineral Resource Estimate Effective 04 September 2019 on a 100% Project Basis at 2.0 g/t Cutoff

Total T Zone at 2.0 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	4 892 193	1.12	2.01	0.04	0.85	4.02	0.16	0.08	19 667	0.632
Indicated	2.0	21 479 925	1.23	2.09	0.03	0.78	4.13	0.19	0.09	88 712	2.852
M+I	**2.0**	**26 372 118**	**1.21**	**2.08**	**0.03**	**0.79**	**4.11**	**0.18**	**0.09**	**108 379**	**3.484**
Inferred	2.0	25 029 695	1.17	1.84	0.03	0.60	3.64	0.14	0.07	91 108	2.929

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.9	50.0	1.0	21.1
Indicated	29.8	50.6	0.7	18.9
M+I	**29.5**	**50.6**	**0.7**	**19.2**
Inferred	32.1	50.5	0.8	16.6

F Zone at 2.0 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	75 332 513	0.82	2.00	0.05	0.14	3.01	0.08	0.19	226 833	7.293
Indicated	2.0	273 272 480	0.80	1.85	0.04	0.14	2.83	0.07	0.18	772 103	24.824
M+I	**2.0**	**348 604 993**	**0.80**	**1.88**	**0.04**	**0.14**	**2.87**	**0.08**	**0.18**	**998 936**	**32.117**
Inferred	2.0	121 535 227	0.70	1.62	0.04	0.13	2.50	0.07	0.16	303 722	9.765

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.2	66.4	1.7	4.7
Indicated	28.3	65.4	1.4	4.9
M+I	**28.0**	**65.7**	**1.4**	**4.9**
Inferred	28.1	65.1	1.6	5.2

Waterberg Aggregate Total 2.0 g/t Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	80 224 706	0.84	2.00	0.05	0.18	3.07	0.08	0.18	246 500	7.925
Indicated	2.0	294 752 405	0.83	1.87	0.04	0.19	2.92	0.08	0.17	860 815	27.676
M+I	**2.0**	**374 977 111**	**0.83**	**1.90**	**0.04**	**0.19**	**2.96**	**0.08**	**0.18**	**1 107 315**	**35.601**
Inferred	2.0	146 564 922	0.78	1.66	0.04	0.21	2.69	0.08	0.15	394 830	12.694

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.3	65.1	1.6	6.0
Indicated	28.4	63.9	1.3	6.4
M+I	**28.1**	**64.3**	**1.3**	**6.3**
Inferred	29.0	61.7	1.5	7.8

Notes:

- 4E = Platinum Group Elements (PGE) (Pt + Pd + Rh) and Au.
- The cutoffs for Mineral Resources were established by a QP 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 entity.
- Conversion factor used – kg to oz = 32.15076.
- Numbers may not add due to rounding.
- A 5% and 7% geological loss were applied to the measured / indicated and inferred Mineral Resource categories, respectively.

Table 1-2: Summary of Mineral Resource Estimate effective 04 September 2019 on a 100% Project Basis at 2.5 g/t (4E) Cutoff

T Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	4 443 483	1.17	2.12	0.05	0.87	4.20	0.15	0.08	18 663	0.600
Indicated	2.5	17 026 142	1.37	2.34	0.03	0.88	4.61	0.20	0.09	78 491	2.524
M+I	**2.5**	**21 469 625**	**1.34**	**2.29**	**0.03**	**0.88**	**4.53**	**0.19**	**0.09**	**97 154**	**3.124**
Inferred	2.5	21 829 698	1.15	1.92	0.03	0.76	3.86	0.20	0.10	84 263	2.709

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.8	50.4	1.2	20.6
Indicated	29.7	50.7	0.6	19.0
M+I	**29.5**	**50.4**	**0.7**	**19.4**
Inferred	29.8	49.7	0.8	19.7

F Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	54 072 600	0.95	2.20	0.05	0.16	3.36	0.09	0.20	181 704	5.842
Indicated	2.5	166 895 635	0.95	2.09	0.05	0.15	3.24	0.09	0.19	540 691	17.384
M+I	**2.5**	**220 968 235**	**0.95**	**2.12**	**0.05**	**0.15**	**3.27**	**0.09**	**0.19**	**722 395**	**23.226**
Inferred	2.5	44 836 851	0.87	1.92	0.05	0.14	2.98	0.06	0.17	133 705	4.299

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.3	65.4	1.5	4.8
Indicated	29.3	64.4	1.6	4.7
M+I	**29.1**	**64.8**	**1.5**	**4.6**
Inferred	29.2	64.4	1.7	4.7

Waterberg Aggregate Total 2.5 g/t Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	58 516 083	0.97	2.19	0.05	0.21	3.42	0.09	0.19	200 367	6.442
Indicated	2.5	183 921 777	0.99	2.11	0.05	0.22	3.37	0.10	0.18	619 182	19.908
M+I	**2.5**	**242 437 860**	**0.98**	**2.13**	**0.05**	**0.22**	**3.38**	**0.10**	**0.18**	**819 549**	**26.350**
Inferred	2.5	66 666 549	0.96	1.92	0.04	0.34	3.27	0.11	0.15	217 968	7.008

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.2	64.0	1.5	6.3
Indicated	29.4	62.6	1.5	6.5
M+I	**29.1**	**63.0**	**1.5**	**6.4**
Inferred	29.5	58.9	1.2	10.4

Notes:

- 4E = PGE (Pt + Pd + Rh) and Au.
- The cutoffs for Mineral Resources were established by a QP 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 entity.
- Conversion factor used – kg to oz = 32.15076.
- Numbers may not add due to rounding.
- A 5% and 7% geological loss were applied to the measured/indicated and inferred Mineral Resource categories, respectively.

Following are the parameters for the Mineral Resources.

- Mineral Resources are classified in accordance with the South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC) 2016 standards. Certain differences exist with the "Canadian Institute of Mining (CIM) Standards on Mineral Resources and Mineral Reserves;" however, in this case the company and QP believe the differences are not material and the standards may be considered the same. Inferred Mineral Resources have a high degree of uncertainty.
- Mineral Resources are provided on a 100% project basis. Inferred and indicated categories are separate. The estimates have an effective date of 04 September 2019.
- A cutoff grade of 2.0 g/t and 2.5 g/t 4E is applied to the selected Base Case Mineral Resources.
- Cutoff grade for the T Zone and the F Zone 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 cutoff considerations. The upper and lower bound metal prices used in the determination of cutoff grade for resources estimated are as follows: US$983/oz-US$953/oz Pt, US$993/oz-US$750/oz Pd, US$1 325/oz-US$1 231/oz Au, US$1 923US/oz-US$972/oz Rh, US$6.08/lb-US$4.77/lb Ni, US$3.08/lb-US$2.54/lb Cu, and US$/ZAR15-US$/ZAR12. These metal prices are based on the estimated 3-year trailing average prices and the spot prices at the time of commencement of the Mineral Resource Estimate modelling. The lower cutoff was tested against the higher metal price in the range and the higher cutoff was tested against the lower price in the range.

The objective of the cutoff grade estimation was to establish a minimum grade for working break even. Following the PFS, the following factors were used for the calculation of cutoff at 2.0 g/t 4E at higher potential prices and 2.5 g/t 4E at more conservative lower prices listed above.

- Working cost mining of US$25.00, R379 per tonne, life-of-mine (LOM) average total operating costs (OpEx) US$38 574 Rand average LOM.
- 80 g/t concentrate, 82% recoveries of the PGMs, 88% of the Cu and 49% of the Ni.
- 85% payability of the PGMs from a third-party smelter, 73% for Cu and 68% for Ni.

These costs recoveries and pay abilities were updated in the DFS for the consideration of Mineral Reserves.

- Charles Muller of CJM Consulting (South Africa) Pty Limited (CJM) completed the Mineral Resource Estimate.
- Mineral Resources were estimated using ordinary kriging (OK) and simple kriging (SK) methods in Datamine Studio3 from 441 mother holes and 583 deflections in mineralisation. A process of geological modelling and creation of grade shells using indicating kriging (IK) was completed in the estimation process.

- The estimation of Mineral Resources considered environmental, permitting, legal, title, taxation, socioeconomic, marketing, and political factors. The Mineral Resources may be materially affected by metals prices, exchange rates, labour costs, electricity supply issues, or many other factors detailed in the company's annual information form.
- Estimated grades and quantities for byproducts are included in recoverable metals and estimates in the DFS. Cu and Ni are the value byproducts recoverable by flotation and for M&I Mineral Resources are estimated at 0.18% Cu and 0.09% Ni in the T Zone and 0.08% Cu and 0.18% Ni in the F Zone.

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

1.11 Mineral Reserve Estimates

The effective date for the Mineral Reserve estimate contained in this report is 04 September 2019.

The Waterberg Project Mineral Reserve Estimate was based on the M&I Mineral Resource material contained in the T Zone and Super F Zone (F Zone) resource block models. The F Zone is comprised of the five sub-zones listed below.

- Super F-South Zone (F-South)
- Super F-Central Zone (F-Central)
- Super F-North Zone (F-North)
- Super F-Boundary North Zone (F-Boundary North)
- Super F-Boundary South Zone (F-Boundary South)

A 2.5 g/t 4E stope cutoff grade was used for mine planning for both the T Zone and F Zone.

The mine design is based on using the sublevel longhole stoping mining method with paste backfill. Sublevel intervals and stope dimensions were established from evaluating mineral resource geometry and continuity, geomechanical study design parameters, and optimizing production rate and resource extraction. Individual stope mining shapes were created using mineable shape optimizer (MSO) software. Stope sill development designs were prepared for all stopes and the Mineral Resources contained in development has been separated from the stopes. The *in situ* Mineral Resource contained in the stope shapes and development designs were extracted from the resource models and include all planned dilution. Modifying factors applied to the *in situ* Mineral Resource include geological losses, external overbreak dilution, and mining losses.

The reference point for the estimated Mineral Reserves is delivery of run-of-mine (ROM) ore to the processing plant.

The estimated proven, probable, and total Waterberg Project Mineral Reserves at 2.5 g/t 4E cutoff effective as of 04 September 2019 are summarized in Table 1-3, Table 1-4, and Table 1-5.

Table 1-3: Proven Mineral Reserve Estimate at 2.5 g/t 4E Cutoff effective 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**3 963 694**	**1.02**	**1.84**	**0.04**	**0.73**	**3.63**	**0.13**	**0.07**	**14 404**	**0.463**
F-Central	17 411 606	0.94	2.18	0.05	0.14	3.31	0.07	0.18	57 738	1.856
F-South	0	0	0	0	0	0	0	0	0	0.000
F-North	16 637 670	0.85	2.03	0.05	0.16	3.09	0.10	0.20	51 378	1.652
F-Boundary North	4 975 853	0.97	2.00	0.05	0.16	3.18	0.10	0.22	15 847	0.509
F-Boundary South	5 294 116	1.04	2.32	0.05	0.18	3.59	0.08	0.19	19 020	0.611
F Zone Total	**44 319 244**	**0.92**	**2.12**	**0.05**	**0.16**	**3.25**	**0.09**	**0.20**	**143 982**	**4.629**
Waterberg Total	**48 282 938**	**0.93**	**2.10**	**0.05**	**0.20**	**3.28**	**0.09**	**0.19**	**158 387**	**5.092**

Table 1-4: Probable Mineral Reserve Estimate at 2.5 g/t 4E Cutoff effective 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**12 936 870**	**1.23**	**2.10**	**0.02**	**0.82**	**4.17**	**0.19**	**0.09**	**53 987**	**1.736**
F-Central	52 719 731	0.86	1.97	0.05	0.14	3.02	0.07	0.18	158 611	5.099
F-South	15 653 961	1.06	2.03	0.05	0.15	3.29	0.04	0.13	51 411	1.653
F-North	36 984 230	0.90	2.12	0.05	0.16	3.23	0.09	0.20	119 450	3.840
F-Boundary North	13 312 581	0.98	1.91	0.05	0.17	3.11	0.10	0.23	41 369	1.330
F-Boundary South	7 616 744	0.92	1.89	0.04	0.13	2.98	0.06	0.18	22 737	0.731
F Zone Total	**126 287 248**	**0.91**	**2.01**	**0.05**	**0.15**	**3.12**	**0.08**	**0.18**	**393 578**	**12.654**
Waterberg Total	**139 224 118**	**0.94**	**2.02**	**0.05**	**0.21**	**3.22**	**0.09**	**0.18**	**447 564**	**14.390**

Table 1-5: Total Estimated Proven and Probable Mineral Reserve at 2.5 g/t Cutoff effective as of 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**16 900 564**	**1.18**	**2.04**	**0.03**	**0.80**	**4.05**	**0.18**	**0.09**	**68 391**	**2.199**
F-Central	70 131 337	0.88	2.02	0.05	0.14	3.09	0.07	0.18	216 349	6.956
F-South	15 653 961	1.06	2.03	0.05	0.15	3.29	0.04	0.13	51 411	1.653
F-North	53 621 900	0.88	2.09	0.05	0.16	3.18	0.10	0.20	170 828	5.492
F-Boundary North	18 288 434	0.98	1.93	0.05	0.17	3.13	0.10	0.23	57 216	1.840
F-Boundary South	12 910 859	0.97	2.06	0.05	0.15	3.23	0.07	0.19	41 756	1.342
F Zone Total	**170 606 492**	**0.91**	**2.04**	**0.05**	**0.15**	**3.15**	**0.08**	**0.19**	**537 560**	**17.283**
Waterberg Total	**187 507 056**	**0.94**	**2.04**	**0.05**	**0.21**	**3.24**	**0.09**	**0.18**	**605 951**	**19.482**

Notes:

- A stope cutoff grade of 2.5 g/t 4E was used for mine planning for the mineral reserves estimate
- Tonnage and grade estimates include planned dilution, geological losses, external overbreak dilution, and mining losses
- Metal prices assumed for cutoff grade estimates were: Pt = $US 960/oz, Pd = $US 993/oz, Rh = $US 1 923/oz, Au = $US 1 325/oz, Cu = $US 6 795/tonne, Ni = $US 13 395/tonne and ZAR:$US 12.04
- 4E = PGE (Pt + Pd + Rh) and Au.
- Numbers may not add due to rounding.

1.12 Mining Methods

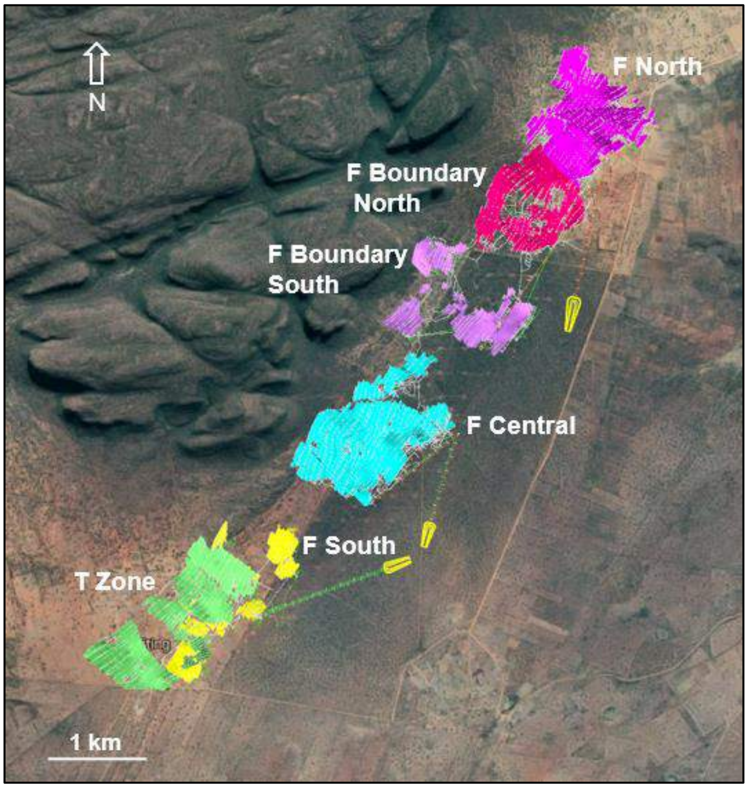
The Waterberg Project will be a 400 000 tpm (400 ktpm) mechanized underground mining operation accessed via declines. The mine design is based on using the Sublevel Longhole Stoping mining method (Longhole) and backfilling the mined voids with paste backfill.

The Waterberg Project was divided into the following three mining complexes.

- The South Complex that includes T Zone and F-South
- The Central Complex that includes F-Central
- The North Complex that includes F-North, F-Boundary North, and F-Boundary South

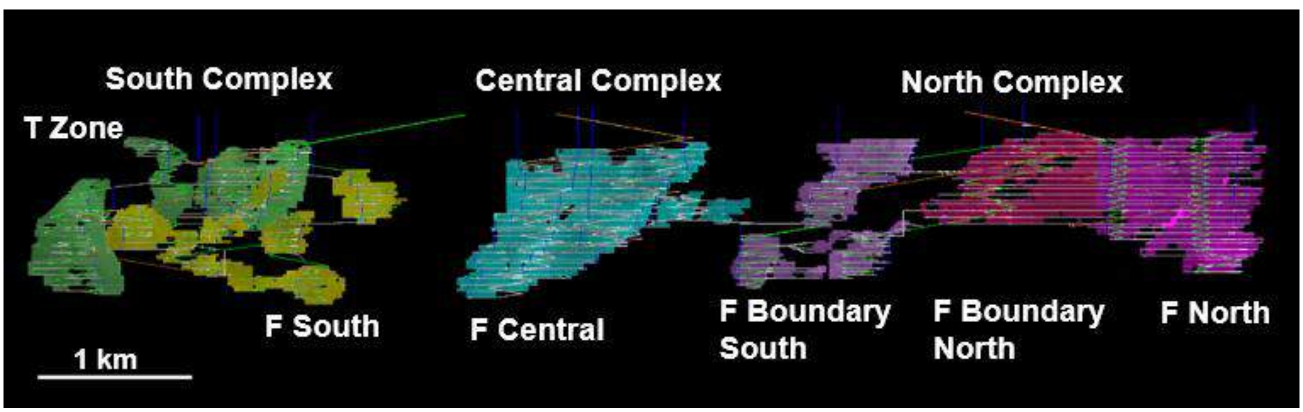
A plan view with the production areas projected to surface is shown in Figure 1-2 and a longitudinal view of the zones, looking approximately northwest (looking from the footwall), is shown in Figure 1-3.

Figure 1-2: Surface Plan View Showing Mineral Resource Extents



Source: Background – Google Maps

Figure 1-3: Longitudinal View of Waterberg Complexes (Looking Northwest)



There will be a box cut and portal at each complex, each with twin declines (service decline and conveyor decline) developed to access and service the complex for the LOM.

1.12.1 Geomechanical

Geomechanics core logging and laboratory test data from the PFS and additional data collected as part of this DFS were combined in a database and used to develop a geomechanical model and for use in rock mass classifications systems to develop rock mechanics parameters for the mine design. The analysis utilised several common empirical models and was validated with numerical modelling in several instances.

Support requirements for development headings were developed and are in line with both empirical calculation methods and common support types. Generally, primary ground support will consist of patterned rock bolts and screen, with application of shotcrete in areas deeper in the mine.

A numerical modelling exercise was undertaken to evaluate the evolution of rock mass damage and paste backfill performance as mining progresses. The principal findings of the modelling exercise are listed below.

- No requirement exists for substantial designed regional ore pillars.
- No major rock mass damage (stopes and rock pillars) was developed above around 300 m below surface. Moderate to major rock mass damage developed in stope abutments and secondary stope cores towards end of the sequence, especially below 1 000 m.
- Paste backfill dilution in wider parts of the ore body is expected, principally affecting secondary transverse stopes. In general, paste backfill dilution is anticipated to increase with depth and towards completion of the mining level and has been reflected in the dilution estimates

Backfill stability was assessed primarily using empirical-analytical methods with developed backfill strength requirements validated by benchmarking and limited three-dimensional (3D) finite element modelling.

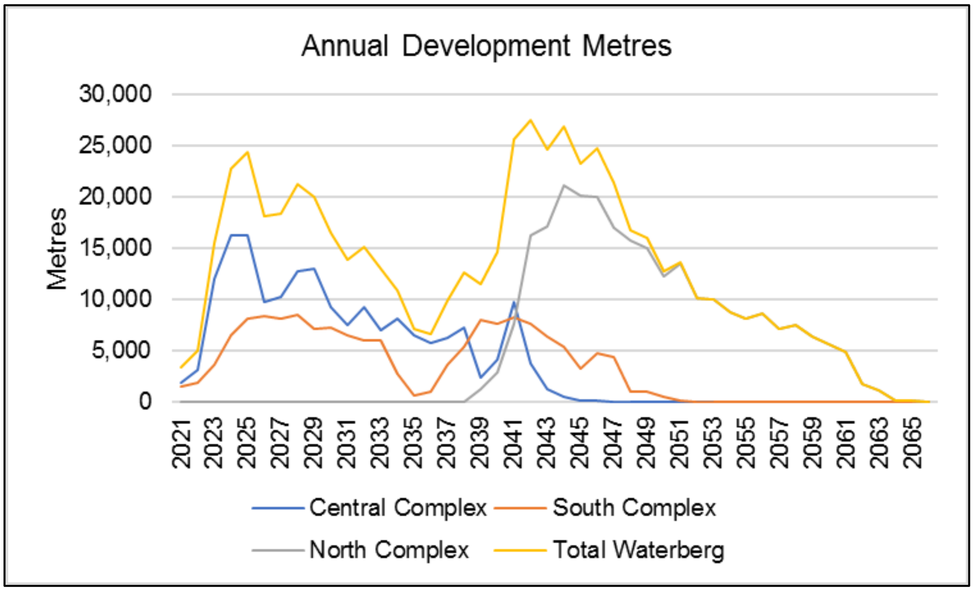
1.12.2 Mine Development

All decline and lateral excavations will be developed using drill and blast methods and mechanized diesel-powered mobile equipment. A summary of the development totals by complex is included in Table 1-6 and the development profile is shown in Figure 1-4.

Table 1-6: Development Quantities by Complex

Item	Central Complex (m)	South Complex (m)	North Complex (m)	Waterberg Total (m)
Decline	22 316	37 197	33 398	**92 911**
Lateral Sublevel and Infrastructure	160 963	112 766	225 750	**499 479**
Total	**183 279**	**149 963**	**259 148**	**592 390**

Figure 1-4: Lateral Development Profile



1.12.3 Production

Mining blocks will be established at 100 m vertical intervals and will consist of two sublevels spaced at 40 m (40 m stope height) and one sublevel spaced at 20 m (20 m uppers stope that will be mined beneath the backfilled stopes in the block above). Individual stopes will be 20 m along strike and a combination of transverse and longitudinal approaches will be used to accommodate the varying ore body thickness. Within each mining block, stopes have been sequenced and there will be multiple stopes in the active stope cycle. To achieve the production profile, there will be multiple mining blocks in production simultaneously.

The production plan focuses on optimizing the ramp-up period and maximizing productivity. Each complex was scheduled independently as a stand-alone operation. The breakdown of tonnes and grade recovered by mining approach and zone is summarised in Table 1-7.

Initial production will come from the simultaneous operation of the Central and South Complexes, with the North Complex phased in once production in the Central and South Complexes begins to ramp down. There will be approximately five years of ramp up from the start of the decline development in 2021 to achieve sustainable 70% of steady-state production in January 2026. Steady-state production of 400 ktpm will be achieved in Q1 2027 with 300 ktpm from the Central Complex and 100 ktpm from the South Complex. Later in the mine life, the North Complex will ramp up to maintain 400 ktpm production. The ramp-up and steady-state production tonnage profiles are shown in Figure 1-5 and Figure 1-6.

Table 1-7: Life-of-Mine Production Summary

	T Zone	F-Central	F-South	F-North	F-Boundary North	F-Boundary South
Ore Tonnes – Stope Total	15 610 201	65 326 918	14 482 019	50 274 701	16 888 572	11 922 776
Ore Tonnes – Transverse	1 689 200	46 538 873	2 302 529	38 755 421	7 318 698	508 303
Ore Tonnes – Longitudinal	13 921 001	18 788 045	12 179 491	11 519 279	9 569 874	11 414 473
Ore Tonnes – Development	1 290 363	4 804 419	1 171 942	3 347 199	1 399 862	988 084
Ore Tonnes – Total	**16 900 564**	**70 131 337**	**15 653 961**	**53 621 900**	**18 288 434**	**12 910 859**
Grade 4E (g/t)	4.05	3.09	3.29	3.18	3.13	3.23
Grade Pt (g/t)	1.18	0.88	1.06	0.88	0.98	0.97
Grade Pd (g/t)	2.04	2.02	2.03	2.09	1.93	2.06
Grade Rh (g/t)	0.03	0.05	0.05	0.05	0.05	0.05
Grade Au (g/t)	0.80	0.14	0.15	0.16	0.17	0.15
Grade Cu (%)	0.18	0.07	0.04	0.10	0.10	0.07
Grade Ni (%)	0.09	0.18	0.13	0.20	0.23	0.19

Notes:
- 4E = PGE (Pt + Pd + Rh) and Au.
- Totals may not add due to rounding.

Figure 1-5: Production Tonnage by Month during Ramp-up

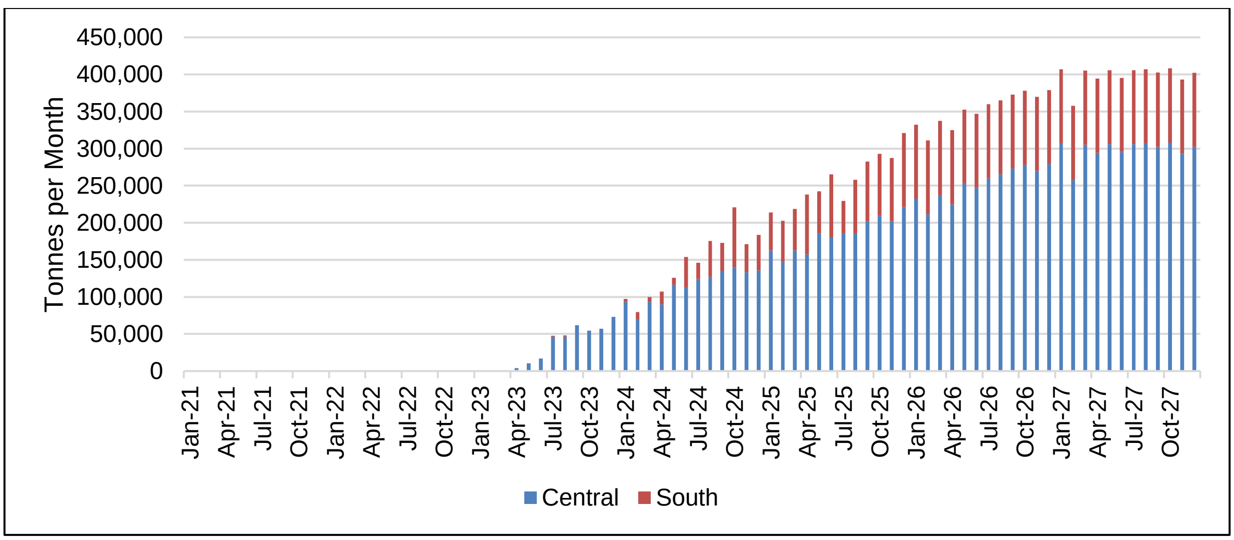
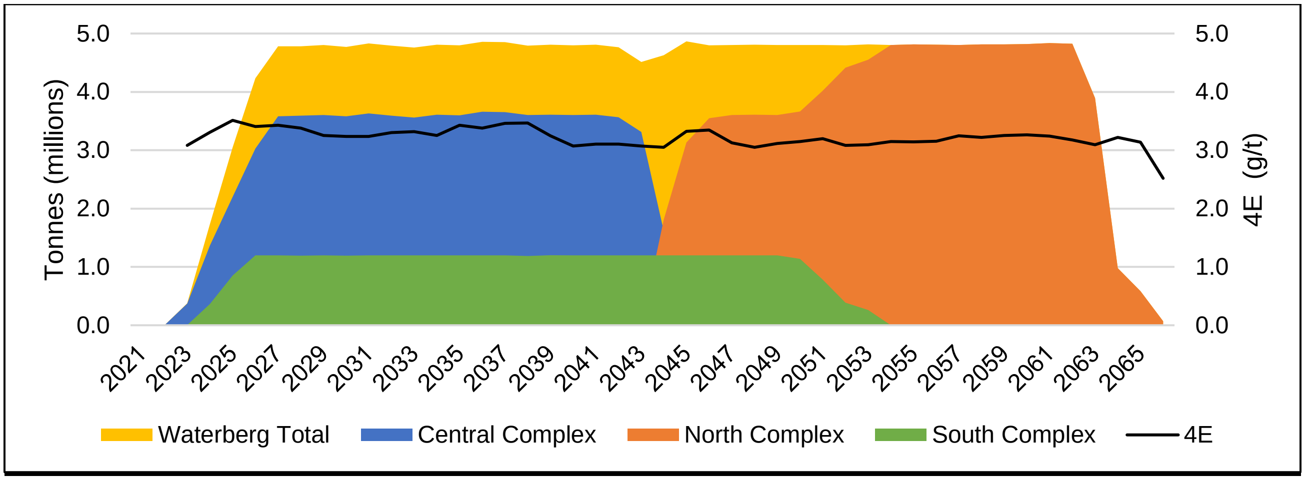


Figure 1-6: Annual Production Tonnage Profile



1.12.4 Ventilation and Mine Air Refrigeration

The underground mobile equipment will be diesel powered. The required ventilation flow will be 1 124 cubic metres per second (m^3/s), 688 m^3/s, and 1 229 m^3/s for the Central, South, and North Complexes, respectively.

Ventilation to each complex will be provided by surface fresh air and return air ventilation raises and the portals / declines. The ventilation systems will be a "pull" system with large surface fans located at the exhaust raises. Ventilation in the conveyor declines will have fresh air pulled from the portals and exhausted without being used to ventilate other mine workings.

The underground heat loads will be countered by a combination of refrigerated air and uncooled air. The cooling requirement will be 20 MW_R, 10 MW_R, and 20 MW_R for the Central, South, and North Complexes, respectively. Mine air cooling will not be required until mining depths reach 700 m below surface in 2030.

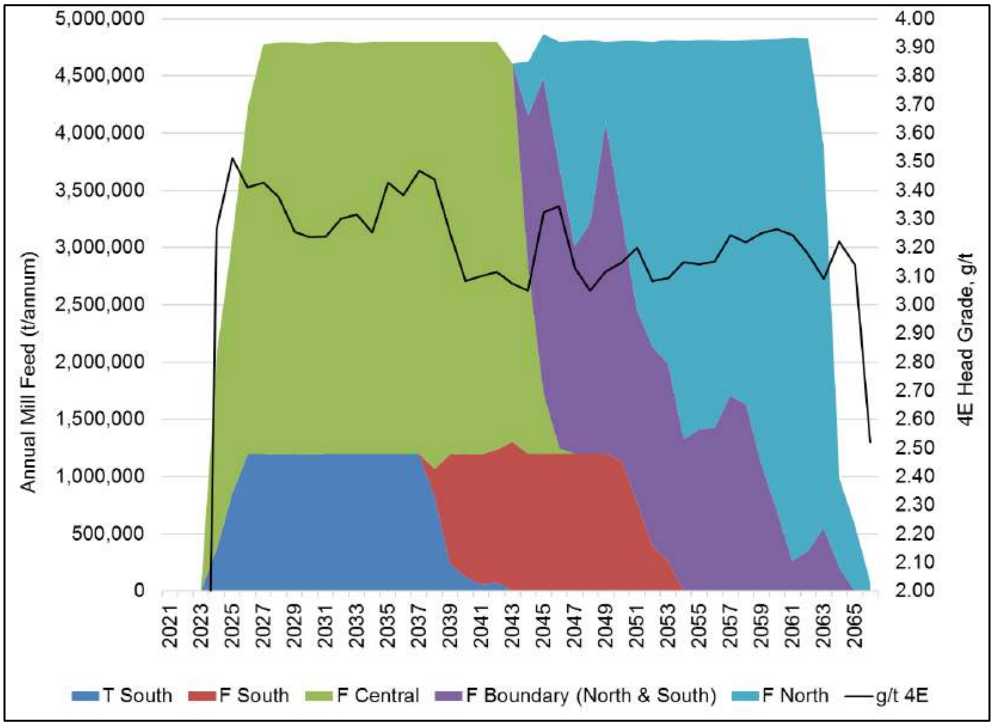
1.13 Recovery Methods

The process design for the Waterberg Concentrator Plant was developed based on the extensive metallurgical test work results and previous studies. The testwork programme developed during the PFS and the DFS identified that the mill-float-mill-float (MF2) configuration following three stage crushing is the most appropriate recovery technique for the PGE and the base metals for the F Zone and the T Zone ores. The plant design makes provision for the controlled blending of the two ore types in the crushing circuit. The blending of the ores does not require a conceptual change to the MF2 flowsheet, but the controlled blending is considered advantageous in providing a consistent feed composition to the process. Further optimisation of the reagent addition during operation to achieve the optimal concentrate grade and recovery can be completed.

The flotation concentrator will produce a concentrate containing 80 g/t 4E with a mass pull of approximately 3.1%. The concentrator was designed to process 4.8 Mtpa (400 ktpm) of ROM and will produce 155 ktpa of concentrate to be shipped by road to a smelter. The concentrate will contain 12% moisture while the tailings will be directed to either the backfill plant for placing as cemented fill underground or to the surface tailings storage facility (TSF).

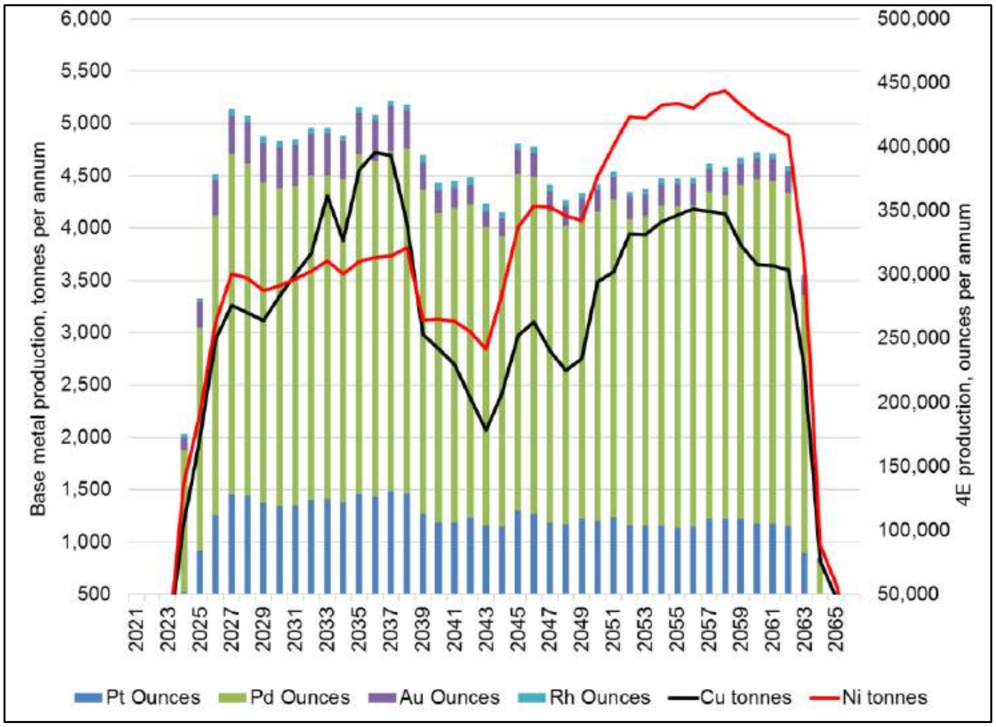
The plant production rate is aligned with mine production and plant production will commence in January 2024 with ramp-up continuing until steady state is reached December 2026 as indicated in Figure 1-7.

Figure 1-7: **Annual Mill Feed Profile Summary**



The concentrate production and contained 4E elements approaching 425 000 ounces per annum is indicated in Figure 1-8 along with anticipated the base metal content in tpa.

Figure 1-8: Annual Metal Production Summary



1.14 Project Infrastructure

The Waterberg Project is located in a rural area with limited existing infrastructure apart from gravel roads, drill hole water, and 22 kV rural power distribution with limited capacity. Upgrading is planned for all existing infrastructure, including the upgrading of 34 km of the gravel roads to the N11 national road.

In addition to three mining complexes and one processing facility, the Waterberg Project infrastructure required for a successful operation will include the construction of a new 132 kV electrical supply from the ESKOM Burotho 400/132 kV main transmission station 74 km south of the site. The development and equipping of a local well field spread over 20 km to provide water.

At the site, a lined TSF, ore stockpile and waste rock storage facilities, backfill preparation and distribution system, and the necessary surface infrastructure to support mining and processing operations will be constructed.

The project will require 90 mega volt amps (MVA) of electrical power and 6.2 ML/day of industrial water.

1.15 Market Studies and Contracts

One of the JV partners of the Waterberg Project is IMPLATS; therefore, no formal marketing study was commissioned for the DFS.

Metal price movements for the economic metals associated with the project (Pt, Pd, Rh, Au, Ni, and Cu) were reviewed for the preceding three years and show that there was a significant change in the market for the major contributors to income generation. The metal prices for the period to 04 September 2019 normalised to 01 July 2019 are detailed in Table 1-8.

Table 1-8: Pricing for all Economic Metals

Period	Pd	Pt	Au	Ni	Cu	Rh
	US$/oz	US$/oz	US$/oz	US$/tonne	US$/tonne	US$/oz
Three-year Trailing	$ 1 055	$ 931	$ 1 318	$ 12 248	$ 6 333	$ 1 930
Two-year Trailing	$ 1 174	$ 891	$ 1 322	$ 13 034	$ 6 530	$ 2 427
One-year Trailing	$ 1 338	$ 841	$ 1 318	$ 12 666	$ 6 146	$ 2 942
04 September 2019 Spot	$ 1 546	$ 980	$ 1 548	$ 17 855	$ 5 646	$ 5 036

Source – 'Johnson Matthey Metal Prices' BMO

Considering these metal prices and the production profile for the Waterberg Project, contributors to income are summarized in Table 1-9. The first 13 years of the production profile is treating about 25% from the T Zone with a different prill spilt to the F Zone ore.

Table 1-9: Economic PGEs and Base Metals for first 13 Years and Life of Mine

Metal	Approximate Percent of Revenue (3-year trailing price to September2019)		Approximate Percent of Revenue (04 September 2019 Spot Price)	
	First 13 years	LOM	First 13 years	LOM
Pd	54.3%	55.8%	59.4%	60.6%
Pt	23.2%	22.1%	18.2%	17.2%
Au	8.3%	6.1%	7.3%	5.3%
Ni	8.7%	10.5%	9.5%	11.3%
Cu	4.1%	4.0%	2.7%	2.6%
Rh	1.5%	1.5%	2.9%	3.0%

No off-take agreement was negotiated for the concentrate but IMPLATS has right of first refusal to develop the Waterberg Project and further treat the concentrate produced. It is anticipated that the payability for the contained metal in concentrate will be 85% for all 4E elements, 73% for Cu, and 68% for Ni. These net-smelter-return factors are fully inclusive of all smelting and refining costs, apart from delivery to the smelter.

It is anticipated that the metal pipeline between delivery of concentrate and payment will be 12 weeks. The Project finances are based on prefunding of the concentrate with an 85% value payment received in Month 1 and the 15% balance paid after the 3 months, incurring an interest charge (as defined in Section 21).

The concentrate from Waterberg Project will be very low in chromitite, which will make this material attractive for blending with other concentrates; however, the contained iron (Fe) and sulphur (S) with high base metals may require further optimization of the smelting and base metal refining protocols. No penalties are expected to be placed upon the concentrate.

1.16 Environmental Studies, Permitting, and Social or Community Impact

In consultation with the community, the mine footprint was planned to exclude areas significant to the community, including prime grazing areas.

Table 1-10 shows key environmental and social licenses and permit applications are required for the Waterberg Project.

Table 1-10: Status of Environmental Licenses and Permits Required for the Waterberg Project

License / Permit Application	Authority	Reference Number	Status
Mining Right (with Social and Labour Plan (SLP)	Department of Mineral Resources (DMR)	LP 30/5/1/2/2 /2/10161MR	Submitted
Environmental Authorisation (EA) [includes Environmental Impact Assessment (EIA) and Environmental Management Programme (EMPr) and Closure Plan]	DMR	LP 30/5/1/2/2 /2/10161EM	Submitted
Waste Management Licence	DMR	LP 30/5/1/2/2 /2/10161MR	Submitted
Water Use Licence	DWA	Imminent Application	Imminent Application
Heritage Resources Consent for Development	South African Heritage Resource Agency (SAHRA)	LP 30/5/1/2/2 /2/10161MR – 12878	Submitted

From an environmental and social perspective, the greatest impacts from mining are anticipated in the eastern (plant footprint) and south-east-central areas of the proposed mining right area. This area is where surface infrastructure is planned as this is the shallowest access for underground mining and is topographically relatively flat. The findings of the Environmental Assessment Practitioner and specialists' assessments have shown that the Waterberg Project may result in both negative and positive impacts to the environment; however, adequate mitigation measures are included into the EMPr to reduce the significance of the identified negative impacts.

The SLP forms part of the mining right in South Africa. It is a commitment to sustainable social development and was submitted, as required, with the mining right application. Local landowners, land users, and communities were consulted and updated from the prospecting stage and are well aware of the project plans. Land use agreements are currently being concluded with the Goedetrouw Community, the Ketting Community, and individual property owners on the farms traversed by the proposed water pipeline and powerlines.

Specific training needs were identified and a detailed training programme is being developed with an internationally recognised organisation to provide the structure and services required for the initial and ongoing needs of the Waterberg Project.

1.17 Capital and Operating Costs

Capital costs to 70% of steady-state production are estimated predominantly in ZAR, with all cost estimates expressed in ZAR real July 2019 terms. Modelled costs are converted to US$ at a long-term real exchange rate of 15.00 (ZAR/US$). The real escalation of costs (in ZAR terms) is estimated to be offset, over time, by the future devaluation of the ZAR against the US$. Estimated capital expenditure is R13 105 M for the Waterberg Project plus R3 453 M for capitalized operating costs to achieve the 70% of steady-state production as detailed in Table 1-11.

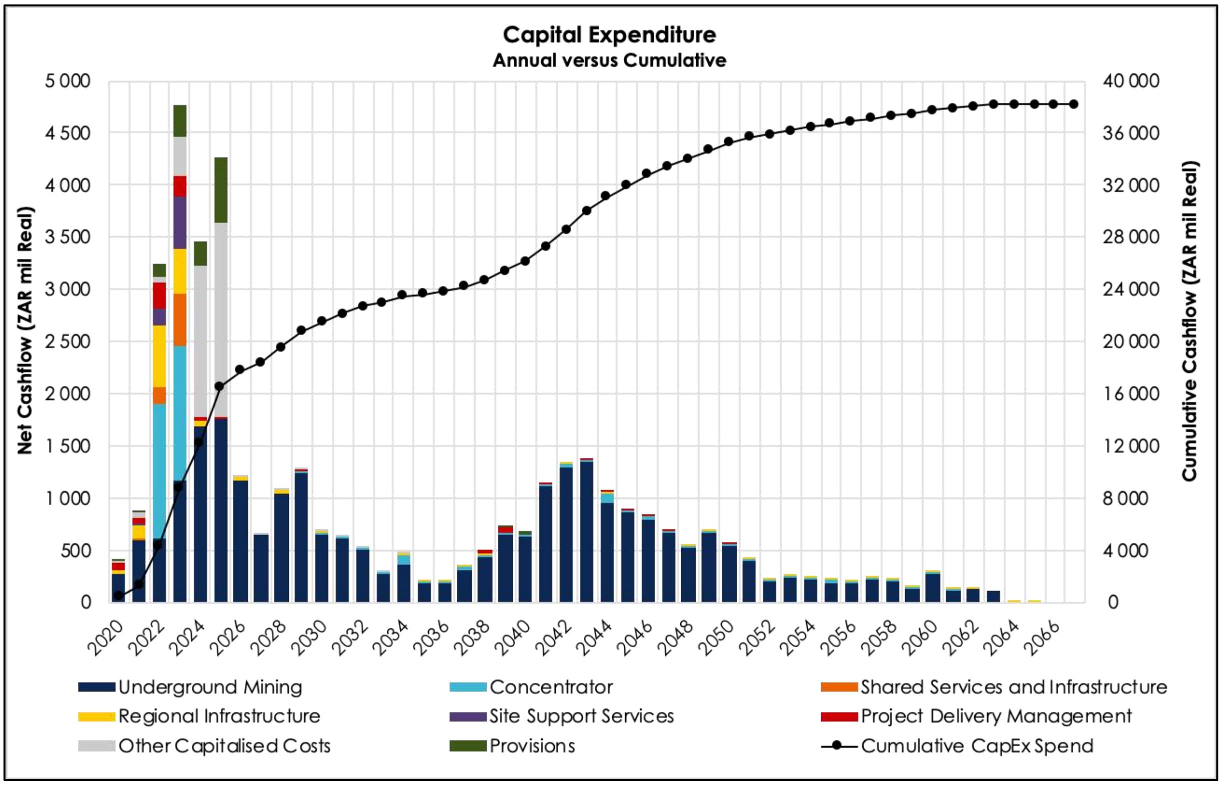
Table 1-11: Waterberg Project Capital Cost

Cost Area	ZAR Total (ZAR M)	USD Total (US$ M)
Underground Mining	R6 097	$406
Concentrator	R2 580	$172
Shared Services and Infrastructure	R682	$45
Regional Infrastructure	R1 229	$82
Site Support Services	R234	$16
Project Delivery Management	R654	$44
Other Capitalised Costs	R331	$22
Contingency	R1 298	$87
Total Project Capital (excluding Capitalised OpEx)	**R13 105**	**$874**
Capitalised Operating Costs	R3 453	$230
Total Project Capital (including Capitalised OpEx)	**R16 559**	**$1 104**

The SIB expenditure covers all expenditure of a capital nature following the achievement of 70% of the steady-state production. This includes all ongoing underground waste development, construction of the North Complex, and the required infrastructure plus mobile equipment replacement and other items of a capital nature associated with the concentrator and general mine infrastructure. The total stay-in-business (SIB) contingency is R21.6 billion spread over the more than 40 years of mine life.

The overall life of mine capital expenditure profile for the Project is shown in Figure 1-9.

Figure 1-9: Capital Expenditure Profile for Life of Mine



The LOM operating costs following achievement of 70% of steady-state production and excluding SIB expenditure is summarised in Table 1-12.

Table 1-12: Waterberg Project Operating Cost

Cost Area	LOM Average (ZAR/t milled)	LOM Average (US$/t milled)
Mining	R345	$23.01
Milling and Processing	R132	$8.79
Engineering and Infrastructure	R116	$7.76
General and Administration	R19	$1.25
Total On-site Operating Costs	**R612**	**$40.80**

The cash cost per 4E ounce is estimated at US$640 (spot prices) and US$554 (three-year trailing prices), respectively. The cash cost includes the smelter discount as a cost, as well as byproduct credits from Cu and Ni sales; therefore, the indicated cash costs are dependent on the prevailing metal price assumptions as detailed in Table 1-13.

Table 1-13: Waterberg Project Cash and All-In-Cost

Metric	Spot Prices (US$ / 4E oz)	Three-year Trailing Prices (US$ / 4E oz)
On-site Operating Costs	$487	$456
Smelting, Refining, and Transport Costs	$302	$227
Royalties and Production Taxes	$88	$54
Less Byproduct Base Metal Credits	$(236)	$(184)
Total Cash Cost	**$640**	**$554**
Sustaining Capital	$94	$88
Total All-in Sustaining Cost	**$734**	**$642**
Project Capital	$34	$32
Total All-in Cost	**$767**	**$674**

1.18 Economic Analysis

Key features of the Waterberg Project are listed below.

- The Waterberg Project capital expenditure (CapEx) (exclusive of sustaining capital) is estimated at R16 559 M (US$1 104 M). The Waterberg Project CapEx includes capitalised operating costs of R3 453 M up to 70% of steady-state production.
- The LOM average OpEx unit cost (exclusive of capitalised OpEx) is estimated at R612 / t milled.
- The Waterberg Project produces a positive business case in both the spot and three-year trailing average metal price scenarios. At spot prices, the Waterberg Project yields a post-tax $NPV_{8.0\%}$ of R14 736 M (US$982 M), at an IRR of 20.7%, an undiscounted payback period of 8.4 years, and a peak funding requirement of R9 255 M (US$617 M). At three-year trailing average metal prices, the project yields a post-tax $NPV_{8.0\%}$ of R5 616 M (US$333 M), at an IRR of 13.3%, an undiscounted payback period of 11.2 years, and a peak funding requirement of R10 261 M (US$667 M).
- At the two pricing scenarios (spot and three-year trailing average) the project generates LOM average cash costs of US$640 / 4E oz and US$554 / 4E oz, respectively, which places Waterberg firmly within the lowest quartile of regional PGE producers.

Appendix A contains a comparison of the outcomes of this DFS to the 2016 PFS.

1.19 Adjacent Properties

Numerous mineral deposits have been outlined along the Northern Limb of the Bushveld Complex. The main projects in the area include Mogalakwena Mine, Aurora Project, Akanani Project, Boikgantsho Project, Hacra Project, and Platreef Project.

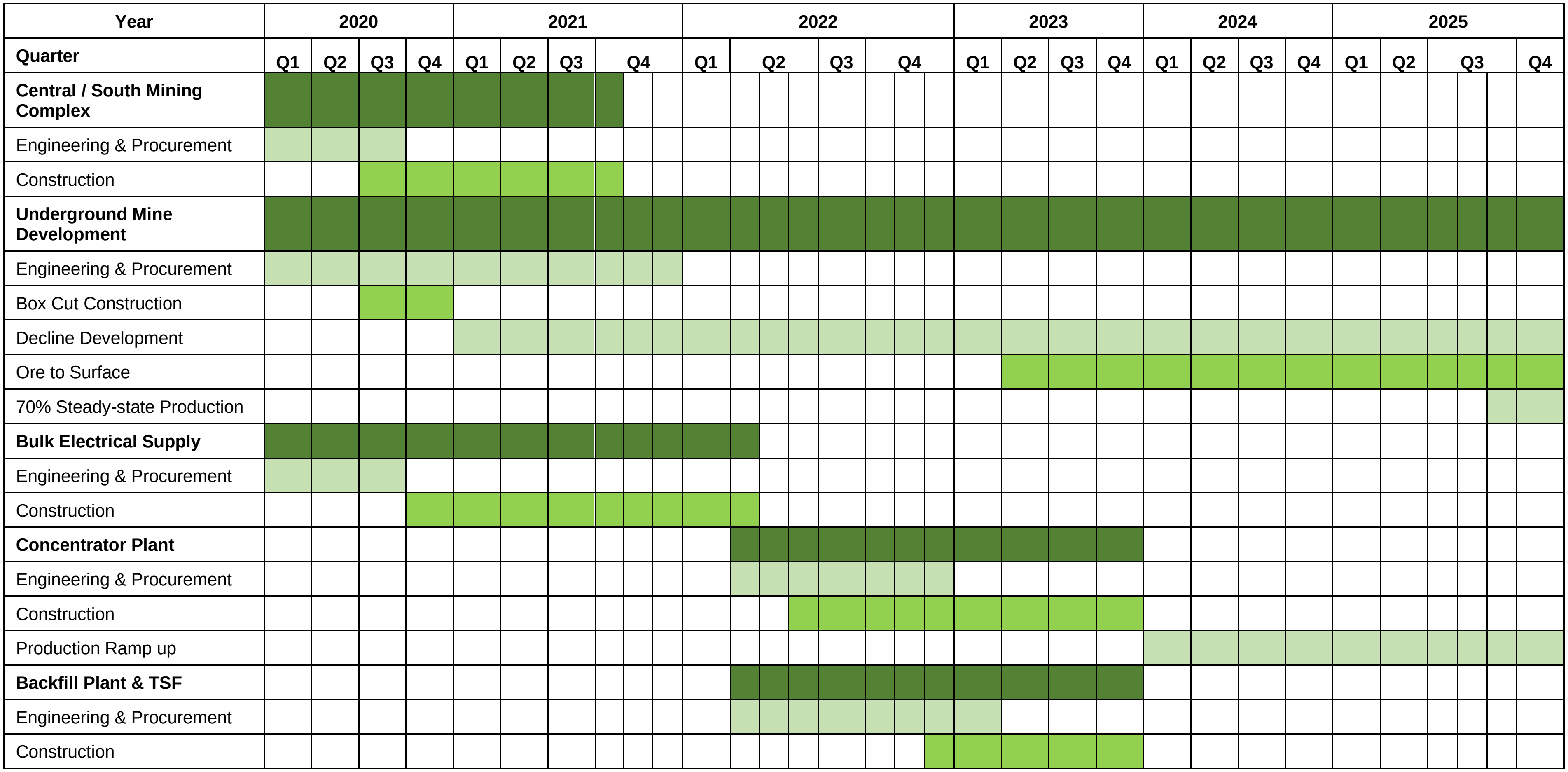
1.20 Project Implementation

The project schedule assumes a start date of January 2020 with the commencement of the detailed engineering and aims to achieve the following key milestones:

- Start of Project – January 2020
- Start of Construction of Central / South Mining Complex – June 2020
- Start of Decline Development – January 2021
- Completion of the 132 kV Bulk Electrical Supply – April 2022
- Start of Ore Processing in Concentrator– January 2024
- Achievement of 70% of Steady-state Capacity – September 2025
- Completion of Capital Period – December 2025

The project schedule is summarised graphically in Figure 1-10.

Figure 1-10: High-level Implementation Schedule



1.21 Interpretations and Conclusions

The database used for the Mineral Resource estimate consisted of 441 drill holes and 583 deflections. The Mineral Resource estimate was completed using geostatistics best practices and the M&I Mineral Resources are at an appropriate level of confidence to be considered in the DFS for mine planning.

The geometry and continuity of the mineral resource and the rock mass quality of the mineralized zones and surrounding rock mass make the Waterberg zones amenable to extraction using the Sublevel Longhole Stoping mining method using paste backfill. The mine design includes all development and infrastructure required to access the Central, South, and North Complexes and mine the estimated Mineral Reserves. A full 3D mine model was created for each complex and a LOM development and production schedule was prepared to determine the estimated tonnes, average grade, and metals profile mined and delivered to surface. Individual stope and development mining shapes were created and include planned dilution and modifying factors to account for geological losses, external overbreak dilution, and mining losses. The estimated Mineral Reserves are supported by a mine plan and economic analysis and demonstrate positive economics.

The development methods and mining methods are safe and highly mechanized and use common equipment and processes that are proven and used successfully in the global mining industry. The successful execution of these methods to achieve planned underground mine development and production at the Waterberg Project will require the operation to establish a culture focused on worker health and safety, investment and emphasis on worker skills training geared toward the equipment and technology used, and structured mine planning.

The metallurgical process selected is proven technology and is appropriate for the ore to be treated and will produce a concentrate containing about 80 g/t 4E at a recovery approaching 80%.

The economics show that the Waterberg Project is financially robust with peak funding at R9 255 M and a payback of 8.4 years for spot prices and R10 261 M with a payback of 11.2 years for three-year trailing prices. The cash cost estimate shows that the Waterberg Project will be in the lower quartile of PGM mining operations in the southern African region.

1.22 Recommendations

The key recommendations related to the Mineral Resource are summarized below.

- It is recommended that dedicated Mineral Resource definition drilling from both surface and underground be completed during the access period to upgrade some of the indicated Mineral Resources to measured Mineral Resources.
- Currently, only the larger geological structures have been modelled. It is recommended that a detailed structural analysis is conducted and modelled.

The key recommendations related to the mine design and Mineral Reserves are summarised below.

- There is Mineral Resource below the stope cutoff that is not included in the mine plan but is adjacent to planned development and stoping areas. A lower cutoff grade could potentially bring this material into the mine plan with incremental additional development and add to the Mineral Reserves. It is recommended to evaluate the potential for reducing the stope cutoff grade.
- There is Mineral Resource that is above cutoff that could not be included in a longhole stope shape due to local geometry. This material could be amenable to mining using Cut and Fill or Board and Pillar methods. It is recommended to determine the stoping cutoff for this material and evaluate the potential to include some of this material in the mine plan and add to the Mineral Reserves.
- It is recommended to monitor the progress and application of battery-powered mobile equipment technology and evaluate the opportunities this technology could present to the Waterberg Project.
- It is recommended that further geotechnical and geomechanical work be completed as part of project execution to validate mine design assumptions and support the detailed design for underground and surface infrastructure.

The following metallurgical test work is recommended during project execution.

- Further flotation testwork to confirm the effect of the available groundwater on flotation performance and to determine what adjustments to the raw water circuit would be required (if any)
- Concentrate thickening and filtration testwork.
- Further tailings thickening and filtration testwork for confirmation of backfill plant design criteria.

It is recommended Waterberg JV Resources continue their current permitting strategy to develop positive community support and streamline final project approval as outlined below.

- Maintain regular consultation activities with all appropriate national, provincial, and local regulatory agencies and officials.
- Maintain engagement with local communities.

Waterberg JV Resources has a programme of work in place to comply with the necessary environmental, social, and community requirements. Following is key work that should continue.

- Environmental, Social, and Health Impact Assessment (ESHIA) in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA).
- Public Participation Process in accordance with the NEMA.
- Specialist investigations in support of the ESHIA.

- Integrated Water Use License (WUL) Application in compliance with the National Water Act.
- Integrated Water Management License (WML) in compliance with the National Environmental Management Waste Act.

If the permits are received for construction and operation the project is recommended to move into the detailed design and planning for project implementation.

It is recommended that the concentrate off-take discussions be initiated with the JV partner (and others) to confirm the net smelter return payabilities for the economic metals in the concentrate to be sold by Waterberg, as this will have a material impact on the overall finances.

Based on the positive economics from the technical inputs and the financial analysis, it is recommended that the Waterberg Project be considered by the members of the Waterberg JV for an investment decision.

2 INTRODUCTION

2.1 Platinum Group Metals Ltd.

This report was compiled for Waterberg JV Resources, as directed by a Technical Committee of all of the Owners. Platinum Group Metals Ltd. acted as Manager.

The Waterberg Project is owned by Waterberg JV Resources. PTM RSA initially held a 74% share in the JV with Mnombo, a BEE partner, holding the remaining 26% share.

The Waterberg JV Project has since transferred to Waterberg JV Resources (Pty) Ltd and has ownership of the Waterberg Project. Currently, PTM has a 37.05% holding in Waterberg JV Resources, Mnombo has a 26.0% holding, JOGMEC has a 12.195% holding, Hanwa has a 9.755% holding, and IMPLATS has a 15.0% holding. Also note that in November 2011, PTM RSA acquired a 49.90% holding of Mnombo.

2.2 Terms of Reference and the Purpose of this Report

Waterberg JV Resources requested that Stantec – Mining (Stantec) compile an independent technical report on the Waterberg Project. The work for the Waterberg Project DFS was completed by Stantec, DRA Projects SA (Pty) Ltd (DRA), CJM, Turnberry Projects (Turnberry), Bateleur Environmental & Monitoring Services (BE), and Sustainable Slurry and Backfill Solutions (SSBS). The individuals performing the work were independent of Waterberg JV Resources.

The purpose of this report is to make public the updated Mineral Resource estimate and Mineral Reserve estimate along with the results of the DFS.

The following companies have undertaken work in preparation of the DFS.

- Stantec: overall report preparation, mineral reserve, and mining.
- DRA: metallurgical testwork, concentrator design, surface infrastructure, and financial analysis.
- CJM: geology, drilling, and mineral resource.
- Turnberry: mineral processing review.
- BE: hydrology and environmental.
- PTM RSA: property description, location, ownership, mineral tenure and marketing.

This report uses metric measurements. The currency used is ZAR and US$.

2.3 Sources of Information

Reports and documents listed in Section 3 and Section 27 of the Waterberg Project PFS were used to support preparation of the DFS. Additional information was provided by PTM RSA as supporting information for the QPs.

The QPs for this report used the data provided by the representative and internal experts of PTM RSA. This data was derived from historical records for the area as well as information currently compiled by PTM RSA.

2.4 Involvement of the Qualified Person and Personal Inspections

The QPs each visited the site and were involved in writing this NI-43-101 Technical Report.

- Michael Murphy visited the site on 01 October 2018.
- Gordon Cunningham visited the site on the following dates.
  - 27 February 2013 – two-day site visit to view core and site for evaluation of scoping study potential.
  - 13 October 2016 – one-day site visit to view PFS core and site infrastructure.
  - 12 February 2017 – one-day site visit for update on drilling and for infrastructure review for DFS preparation.
- Charles Muller visited the site on several occasions from 2015 to 2019.

2.5 Specific Areas of Responsibility

Following are the QPs specific areas of responsibility for this report.

- Michael Murphy, P. Eng., Stantec - Mining, Manager, Mining Engineering was responsible for: Sections 1.1, 1.2, 1.11, 1.12, 1.17, 1.19, 1.20, 1.21, 1.22; Parts of Section 2; Parts of Section 3; Sections 4.1 to 4.4; Parts of Section 6; Section 15; Section 16; Parts of Section 21; Section 23; Section 24; Sections 25.2, 25.3, 25.8; Sections 26.2, 26.3; Parts of Section 27.
- Charles Muller, CJM (Pty) Ltd, Independent Geological Competent Person was responsible for: Sections 1.3 to 1.8, 1.10, 1.21, 1.22; Parts of Section 2; Parts of Section 3; Parts of Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Section 25.1; Section 26.1; Parts of Section 27.
- Gordon Cunningham, Pr. Eng., Turnberry, Director, was responsible for: Sections 1.9, 1.13, 1.14, 1.15, 1.16, 1.17, 1.18, 1.20, 1.21, 1.22; Parts of Section 2; Parts of Section 3; Sections 4.5 to 4.8; Section 5; Section 13; Section 17; Section 18; Section 19; Section 20; Parts of Section 21; Section 22; Sections 25.4, 25.5; 25.6, 25.7, 25.8, 25.9; Sections 26.4, 26.5, 26.6, 26.7, 26.8; Parts of Section 27.

2.6 Effective Dates

Following are the effective dates for the information included in this report.

- NI 43-101 Technical Report Issuance 04 October 2019
- Mineral Resource Estimate Update on Waterberg Project 04 September 2019
- Mineral Reserve Estimate Update on Waterberg Project 04 September 2019

3 RELIANCE ON OTHER EXPERTS

The QPs who have authored this report take overall responsibility for the report. The QPs are relying, in part, on information provided by other experts in their field, but who are not QPs for this Technical Report.

The Geological QP, Charles Muller, relied on the following experts for some portions of his responsible sections.

- Geological drilling and assay information supplied by Waterberg JV Resources.
- Ownership and Permitting status supplied by Waterberg JV Resources legal tenure specialists.

The Mining QP, Michael Murphy, relied on the following experts for some portions of his responsible sections.

- Bluhm Burton Engineering (BBE) for mine air refrigeration design compiled from BBE Report No. 16020-TR-001-(R0).
- Open House Management Solutions (OHMS) for Geomechanical core logging.
- RockLab Division of SoilLab (PTY) Ltd for rock mechanics laboratory testing for rock properties.

The Process, Infrastructure, Environmental and Financial QP, and Competent Valuator, Gordon Cunningham, relied on the following experts for some portions of his responsible sections.

- Process plant design and mineralogical testwork was compiled by DRA.
- Mintek for all metallurgical testing and associated analyses, under the direction of DRA, and Turnberry for Waterberg JV Resources.
- Testwork analytical and survey data compiled by Waterberg JV Resources.
- Backfill surface preparation plant design compiled by SSBS for Waterberg Project.
- TSF and associated infrastructure for the Waterberg Project compiled by Epoch Resources (Pty) Ltd.; for information derived through the following documents: "*Feasibility Study of the Tailings Storage Facility*," and "*Associated Infrastructure for the Waterberg Project*."
- Surface geotechnical evaluation by Inroads Consulting under the direction of DRA and Epoch.
- Independent environmental studies filed with the DMR for the Waterberg Project were compiled by BE for information derived through the following documents.
  - Two annual Environmental Monitoring and Reporting documents in terms of the MPRDA.
  - Annual Financial Provision Determination reports of the financial guarantees in terms of the MPRDA.
- Community and Social Assessment supplied by PTM RSA.
- High-voltage power system design for transmission from the Eskom grid to Waterberg Project compiled by Tdx Power.
- Water sourcing, pumping, collection, and reticulation to the Waterberg Project compiled by WSM Leshika.

- Capital costing for the Waterberg Project was provided by the different technical experts and collated by DRA, Stantec, and Practara for inclusion in the financial model and the Technical Report.
- Operating costs were provided by the different technical experts and collated by Practara for inclusion in the Technical Report and the financial model.
- Marketing and contracts for the project was compiled by Turnberry.
- Metal prices as provided by BMO and Johnson-Matthey and collated by Practara and Turnberry.
- Waterberg JV Resources provided legal tenure specialists royalty and taxes assumptions for royalties and taxes for use in the financial model.
- The financial model was compiled by Practara for evaluation by the Waterberg JV partners and inclusion in the Technical Report.
- 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 were 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 and Competent Valuator.

- Charles J. Muller
- Gordon I. Cunningham
- Michael Murphy

These individuals are considered QPs under NI 43-101 definitions. The QPs reported and made conclusions within this report with the sole purpose of providing information for the Waterberg JV partners and the use is subject to the terms and conditions of the contract between the QPs and the Waterberg JV Resources.

The contract permits Waterberg JV Resources (and particularly PTM) to file this report, or excerpts thereof, as a Technical Report with the Canadian Securities Regulatory Authorities or other regulators pursuant to provincial securities legislation, or other legislation, with the prior approval of the 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.

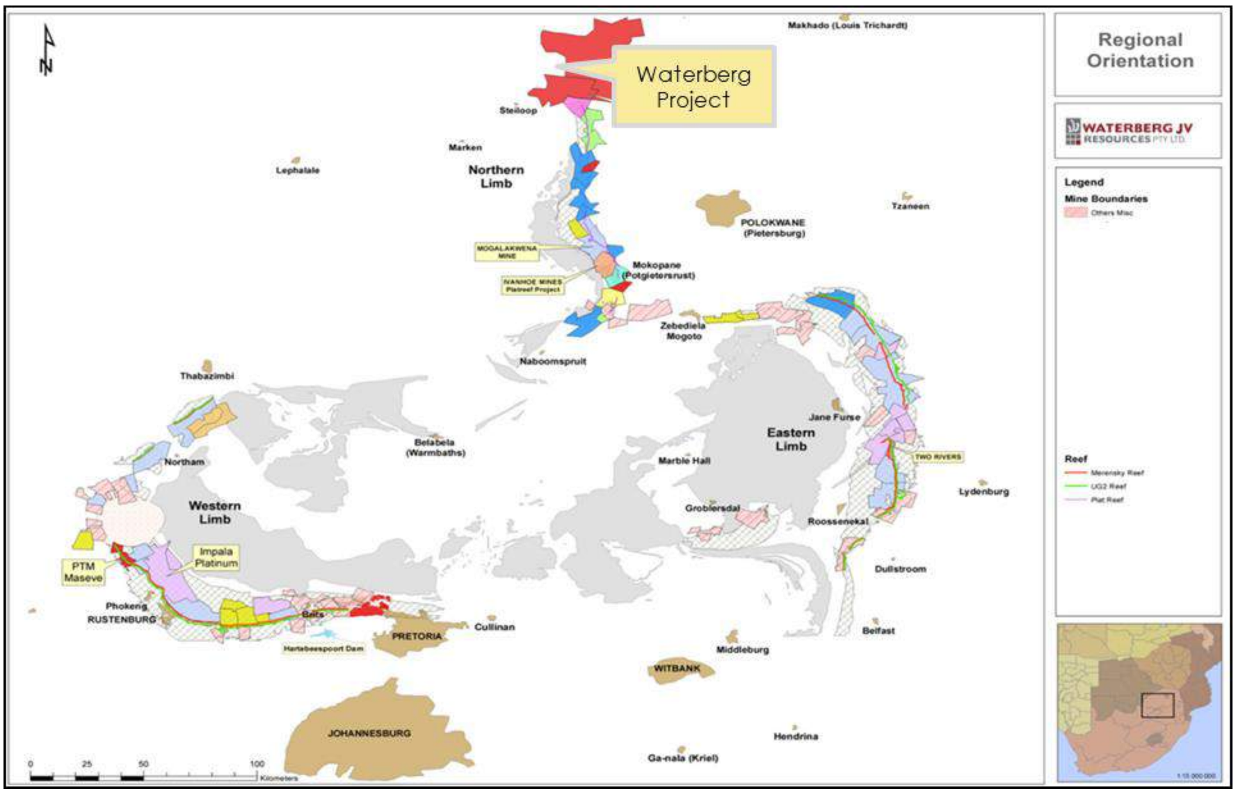
The QPs are not qualified to offer legal opinion on title and offer no opinion as to the validity of the titles claimed. The description of the properties and ownership is provided for general purposes only and was supplied by Waterberg JV Resources. The QPs were satisfied with the title to the extent required for the statement of Mineral Resources and Mineral Reserves and this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property and Title

The Waterberg Project is located 85 km north of the town of Mokopane (formerly Potgietersrus) in the province of Limpopo, South Africa approximately 330 km NNE from Johannesburg as shown in Figure 4-1. The Waterberg Project is approximately centered on Universal Transverse Mercator (UTM) coordinate (Latitude 23°23′15” S, Longitude 28°54′ 10” E). Elevation ranges from approximately 880 to 1 365 m above sea level.

Figure 4-1: Location of the Waterberg Project



The Waterberg 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 estimated Mineral Resources and Mineral Reserves

Waterberg JV Resources currently holds PRs covering an area of 92 672 ha. An application for a mining right covering an area of 20 482 ha was filed with the DMR Polokwane Regional Office and accepted on 14 September 2018. The mining right application area consist of farms of active PRs and farms of expired PR11013. The total project area, active PRs, and mining right application area covers a total area of 99 244 ha.

4.2 Type of Mineral Tenure

A summary of the mineral exploration and mining rights regime for South Africa is provided in Table 4-1. It should be noted that Waterberg JV Resources has a PR that 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 PR to a mining right.

Table 4-1: Summary of Mineral Exploration and Mining Rights (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
PR	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 PR
Area Reduction	No
Procedure	Apply to Regional DMR
Granted by	Minister
Prospecting Right	
Name	PR
Purpose	All exploration activities including bulk sampling
Maximum Area	Not limited
Duration	Up to 5 years
Renewals	Once for 3 years
Area Reduction	No
Procedure	Apply to Regional DMR
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 DMR
Granted by	Minister

4.3 Mineral Right Status

A summary of the PRs and their status is summarised in Table 4-2 and their location is presented in Figure 4-2. A mining right application was filed and accepted for consideration prior to the expiry dates recorded below on 14 September 2018. The farms included in the mining right application are shown in Figure 4-3.

Table 4-2: Summary of Mineral Exploration and Mining Rights (Waterberg JV Resources)

DMR PR Reference	ha	Period of PR	Minerals	Status	Status Details
11013 PR	15 256.90	30 Sep 15 to 29 Sep 18	PGM, Au, Cr, Ni, Cobalt (Co), Cu, Molybdenum (Mo), Rare Earths, Silver (Ag), Zinc (Zn), and Lead (Pb)	Expired	Expired 29 Sep 18 in terms of MPRDA
10667 PR		02 Oct 13 to 01 Oct 18	PGM, Au, Cr, Ni, Co, Cu, Mo, Rare Earths, Ag, Zn, and Pb	Expired	Registered in Mineral & Petroleum Titles Office (MPTO) 153/2013 21 Nov 13
10667 PR	6 254.80	Renewal Application filed with DMR 05 Jul 18 for a further period of 3 years from 01 Oct 18 to 02 Oct 21 In terms of Section 18 (5) MPRDA a PR for which an application for renewal was lodged, despite its expiry date shall remain in force until the renewal application was granted or refused	PGM, Au, Cr, Ni, Co, Cu, Mo, Rare Earths, Ag, Zn, and Pb	Pending	DMR acknowledged receipt on 06 Jul 18. New SAMRAD reference number given LP30/5/1/1/2/ 13201 PR. Applicable when renewal granted.
10809 PR		30 Aug 17 to 29 Aug 22	V and Fe	Granted	Notarially Executed 29 Aug 17
10668 PR	3 953.05	02 Oct 13 to 01 Oct 18	PGM, Au, Cr, Ni, Co, Cu, Mo, Rare Earths, Ag, Zn, and Pb	Expired	This PR shall not be renewed. A closure application shall be filed when the Waterberg Mining Right is granted
10804 PR		02 Oct 13 to 01 Oct 18	PGM, Cr, Cu, Au, Ni, V, and Fe	Expired	Registered in MPTO 106/2015 10 Sep 15
10804 PR	26 961.59	Renewal Application filed with DMR 05 Jul 18 for a further period of 3 years from 01 Oct 18 to 02 Oct 21 In terms of Section 18 (5) MPRDA a PR for which an application for renewal was lodged, despite its expiry date shall remain in force until the renewal application was granted or refused.	PGM, Cr, Cu, Au, Ni, V, and Fe	Pending	DMR acknowledged receipt on 06 Jul 18. New SAMRAD reference number given LP30/5/1/1/2/ 13203 PR. Applicable when renewal granted.
10805 PR		02 Oct 13 to 01 Oct 18	PGM, Cr, Cu, Au, Ni, V, and Fe	Expired	Registered in MPTO 49/2015 24 Apr 15
10805 PR	17 734.80	Renewal Application filed with DMR 05 Jul 18 for a further period of 3 years from 01 Oct 18 to 02 Oct 21 In terms of Section 18 (5) MPRDA a PR for which an application for renewal was lodged, despite its expiry date shall remain in force until the renewal application was granted or refused.	PGM, Cr, Cu, Au, Ni, V, and Fe	Pending	DMR acknowledged receipt on 06 Jul 18. New SAMRAD reference number given LP30/5/1/1/2/ 13202 PR. Applicable when renewal granted.
10805 PR – Section 102	4 475.13	Section 102 application when granted will have the same benefits as 10804 (PR will be granted from 1 Oct 13 to 2 Oct 18)	PGM, Cr, Cu, Au, Ni, V, and Fe	Accepted	Written acceptance by DMR on 09 Dec 13
10806 PR	13 143.53	30 Sep 15 to 29 Sep 20	PGM	Granted	Registered in MPTO 76/2017 19 Sep 17
10810 PR		23 Oct 15 to 22 Oct 18	PGM, Cr, Cu, Au, Ni, V, and Fe	Expired	Registered in MPTO 163/2013 03 Dec 13
10810 PR	4 189.86	Renewal Application filed with DMR 05 Jul 18 for a further period of 3 years w e f 01 Oct 18 to 02 Oct 21 In terms of Section 18 (5) MPRDA a PR for which an application for renewal was lodged, despite its expiry date shall remain in force until the renewal application is granted or refused	PGM, Cr, Cu, Au, Ni, V, and Fe	Pending	DMR acknowledged receipt on 06 Jul 18. New SAMRAD reference number given LP30/5/1/1/2/ 13200 PR. Applicable when renewal granted.
11286 PR	19 912.44	23 Nov 16 to 22 Nov 21	PGM, Au, Cr, Ni, Co, Cu, Mo, Rare Earths, Ag, Z, and Pb, V, and Fe	Granted	Registered in MPTO 54/2017 12 Jul 17

Notes:

- PR 11013 PR expired on the 29 September 2018. Renewed period of three years expired. No further provision for renewal under MPRDA.
- The farms Ketting 368 LR -Goedetrouw 366 LR-Disseldorp 369 LR form part of the Waterberg mining right application, which was accepted on the 14 September 2018 by the DMR and is currently undergoing the required adjudication process by DMR.
- PR 10667 LR, 10804 PR and 10805 PR all expired on the 01 October 2018 and 10810 PR expired on the 22 October 2018 and included in these PRs are certain farms which were included in the mining right application and are recoded below.
- PR 10667 PR – the farms Millstream 358 LR, Rosamond 357 LR are included in the mining right application.
- PR 10804 PR – the farms Lomondside 323 LR, Langbryde 324 LR, Old Langsine and Early Dawn 361 LR are included in the mining right application.
- The above PRs and PRs 10667 LR, 10804 PR and 10805 PR all expired on the 01 October 2018 and 10810 PR expired on the 22 October 2018 of which renewal applications were filed with the DMR for a further period of three years.
- The DMR recorded in its acknowledgment letters in respect of the renewal applications that in terms of Section 18 (5) MPRDA a PR for which an application for renewal has been lodged, despite its expiry date shall remain in force until the renewal application has been granted or refused.

Figure 4-2: Location of the Waterberg Project Prospecting Rights

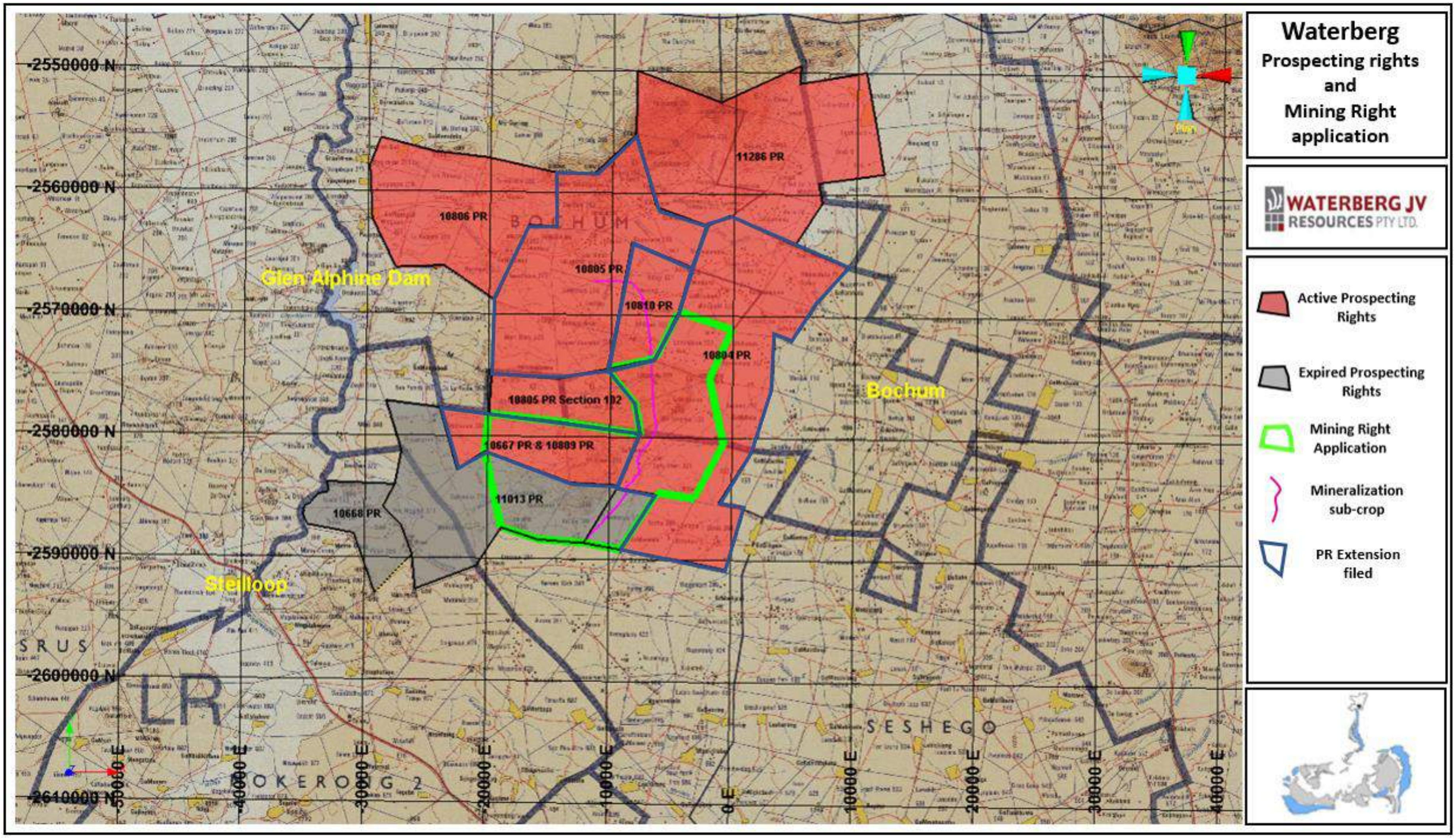
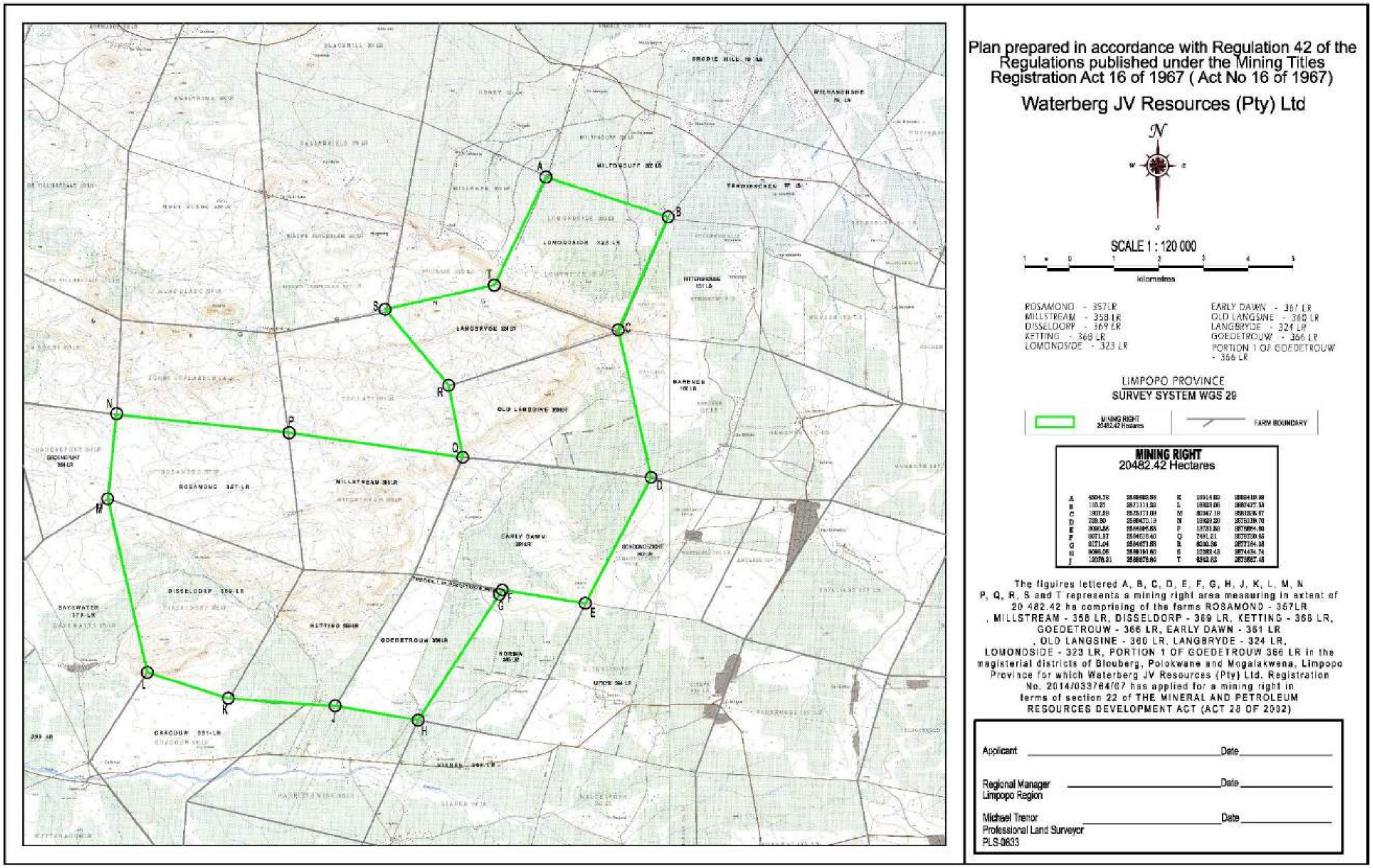


Figure 4-3: The Farms Included in the Mining Right Application



4.4 Holdings Structure

Historically, to cater to the needs, requirements, and objectives of the various ownership groups, the Waterberg Project was managed and explored under the direction of two separate technical committees – the JV and Extension Projects.

A second agreement described in Section 4.4.3 resulted in the consolidation of all holdings and the combined exploration and management of both areas.

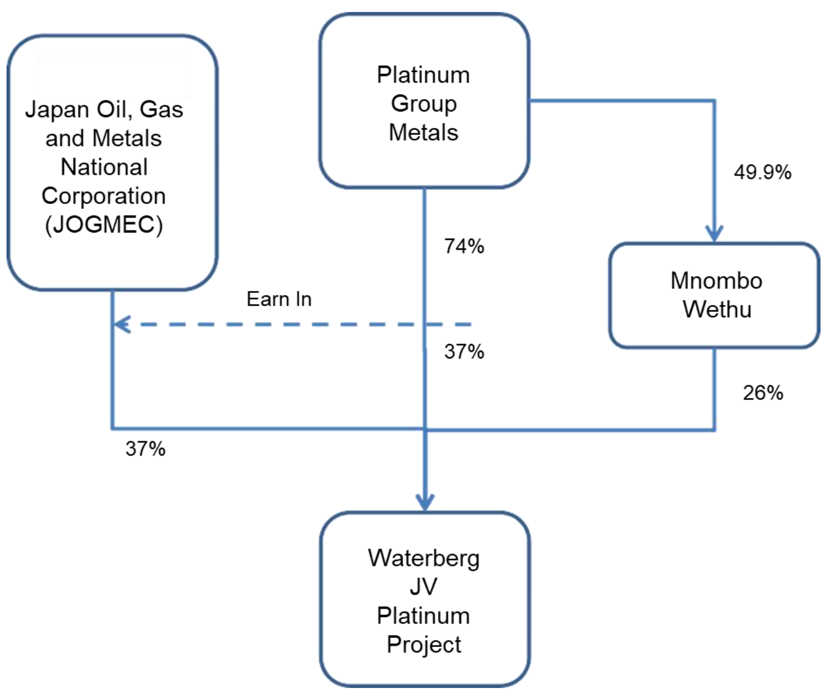
4.4.1 History of the Waterberg JV Project

PTM RSA applied for the original 137 km^2 PR for the Waterberg JV Project area in 2009, which was granted by the DMR in September 2009 and valid until September 2012. An application was completed for the renewal of this PR for a further three years. Under the MPRDA No. 28 of 2002, the PR remains valid pending the grant of the renewal.

PTM RSA initially held a 74% share in the Waterberg JV Project with Mnombo Wethu Consultants (Pty) Ltd. (Mnombo), a BEE partner, holding the remaining 26% share.

In October 2009, PTM RSA and Mnombo entered into a JV agreement with JOGMEC, whereby JOGMEC would earn a participating interest of up to 37% in the Waterberg JV Project for an optional work commitment of US$3.2 million over four years (Figure 4-4). 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).

Figure 4-4: Initial Holdings of Waterberg JV Project



In November 2011, PTM RSA entered into an agreement with Mnombo whereby PTM RSA 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 RSA would pay for Mnombo's 26% share of costs on the initial Waterberg JV area until the completion of the DFS. Mnombo would retain over 50% held for the benefit of historically disadvantaged persons or historically disadvantaged South Africans.

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 RSA funded Mnombo's 26% share of costs for US$1.12 million and the earn-in phase of the JV ended in May 2012. Pursuant to the JOGMEC Agreement, and prior to the closing of the 2nd Amendment (Section 4.4.3) interests in the Waterberg JV Project were held 37% by the Company, 37% by JOGMEC, and 26% by Mnombo. Due to the Company's 49.9% ownership interest in Mnombo, the Company had an effective interest in the Waterberg JV Project of approximately 50%. This ownership percentage will change if the 2nd Amendment, as described in Section 4.4.3, receives Section 11 approval.

During 2012, PTM RSA made application to the DMR to acquire three additional PRs 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 PRs covering a total of 118 km^2 became part of the existing JV with JOGMEC and Mnombo, bringing the total area in the JV to 255 km^2.

4.4.2 History of the Waterberg Extension Project

The former Waterberg Project includes contiguous PRs with a combined area of approximately 864 km^2 adjacent and to the north of the Waterberg JV Project.

The three PRs were executed in October 2013 and each was valid for a period of five years, expiring in October 2018. The company made an application under Section 102 of the MPRDA to the DMR to increase the size of one of the granted PRs by 44 km^2. The company has the exclusive right to apply for renewals of the PRs for periods not exceeding three years each and the exclusive right to apply for a mining right over these PR areas. Applications for a fourth and a fifth PR covering 331 km^2 were accepted for filing with the DMR in February 2012 for a period of five years. This PR (10806 PR) was registered on 19 September 2017.

PTM RSA held the PRs filed with the DMR for the Waterberg Extension Project, and Mnombo was identified as the Company's BEE partner. The Company held a direct 74% interest and Mnombo held a 26% interest in the Waterberg Extension Project, leaving the Company with an approximately 86.974% effective interest by way of the Company's approximately 49.9% shareholding in Mnombo.

4.4.3 Waterberg Project Consolidation

In May 2015, a Second Amendment Agreement (2nd Amendment) was concluded between PTM, PTM RSA, JOGMEC, and Mnombo. Under the 2nd Amendment, the Waterberg JV and Waterberg Extension projects (the Waterberg Project) were to be consolidated into a newly created operating company named Waterberg JV Resources (Pty) Ltd. PTM RSA held 45.65% of Waterberg JV Resources while JOGMEC owns 28.35% and Mnombo holds 26%.

Through its 49.9% share of Mnombo, PTM RSA holds an effective 58.62% of Waterberg JV Resources post-closing. Based on the June 2014 Waterberg Mineral Resource Estimate, the number of ounces owned by each entity did not change with the revised ownership percentages. The 2nd Amendment Agreement allowed all the Waterberg Project area to be considered from a Mineral Resource and engineering perspective, allowing for optimization of the 13 km target strike length and exploration and engineering to be aggressively advanced notwithstanding challenging mining markets.

Under the 2nd Amendment, JOGMEC committed to fund US$20 million in expenditures over a three-year period ending 31 March 2018. Of this, US$8 million was funded by JOGMEC to 31 March 2016 and the first US$6 million to be spent in each of the following 2 12-month periods would also be funded by JOGMEC. Project expenditures exceeding US$6 million in either of the following years were to be funded by the JV partners, pro-rata to their interests in Waterberg JV Resources.

PTM RSA subsequently entered into an agreement with Waterberg JV Resources, PTM, Mnombo, and JOGMEC in terms of which all the above PRs held by PTM RSA were ceded to Waterberg JV Resources.

In terms of the agreement, the consent of the Minister of Mineral Resources or his authorised delegate needed to be required for the said cession of the PRs from PTM RSA to Waterberg JV Resources in terms of Section 11 of the MPRDA. Such consent was granted on 22 December 2015.

On 21 September 2017 PTM RSA completed the transfer of all Waterberg Project prospecting permits into Waterberg JV Resources. Effective 21 September 2017, Waterberg JV Resources owned 100% of the PRs comprising the entire Waterberg Project area.

It is also recorded that the now ceded PRs as set out in Table 4-1 were included in the Shareholders Agreement which was executed by the Shareholders of Waterberg JV Resources on the 16 October 2017.

On completion of the transfer of all the PRs to Waterberg JV Resources, it was owned 45.65% by PTM RSA, 28.35% by JOGMEC and 26% by Mnombo.

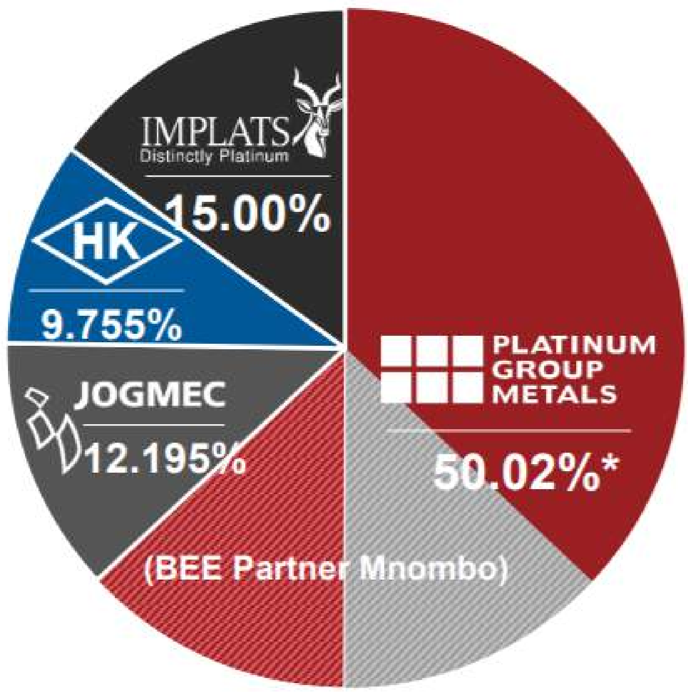
On 16 October 2017, definitive agreements were signed with IMPLATS where IMPLATS purchased 15% of Waterberg JV Resources shares acquiring from PTM RSA (8.6%), and JOGMEC (6.4%). Additionally, IMPLATS acquired a purchase and development option to increase its stake in

Waterberg JV Resources to 50.01% through additional share purchases and earn-in arrangements and acquired a right of first refusal to smelt and refine Waterberg Project concentrate. This transaction closed on 06 November 2017.

Certain proceeds of the IMPLATS transaction are ring-fenced by PTM RSA and disbursed to cover its share of the costs of this DFS. IMPLATS will have an option within 90 business days of the approval by Waterberg JV Resources Board of the completed DFS, to elect to exercise the purchase and development option to increase its interest in Waterberg JV Resources up to 50.01% by purchasing an additional 12.195% equity interest from JOGMEC and earning into the remaining interest by making a firm commitment to an expenditure of US$130.0 million in development work.

PTM RSA is the operator of the Waterberg Project, with JV partners being JOGMEC, Hanwa, IMPLATS, and Mnombo. Figure 4-5 is schematic diagram of the holdings of the Waterberg Project.

Figure 4-5: Waterberg Project Holdings



- Waterberg JV Resources, registration number 2014/033764/07, is a limited liability private company duly incorporated in South Africa.
- PTM is a limited liability public company duly incorporated under the laws of British Columbia, Canada. It is listed on both the Toronto Stock Exchange (PTM) and the New York Stock Exchange (PLG).
- PTM RSA, registration number 2000/025984/07, is a limited liability private company duly incorporated in South Africa and a wholly-owned subsidiary of PTM.
- JOGMEC is an incorporated administrative agency established in accordance with a statute enacted by the National Diet of Japan to promote and participate in oil, gas, petroleum, and metals mining exploration projects of potential benefit to the economy of Japan.

- Hanwa is a Japanese trading company that supplies a broad spectrum of products, including steel, non-ferrous metals, metals and alloys, food, petroleum, chemicals, machinery, lumber, and many other items to an equally diverse range of customers.
- Mnombo, registration number 2012/032630/07, a limited liability private company duly incorporated in South Africa. It is 100% blacked owned (50% black women).
- IMPLATS, registration number 1957/001979/06, is a limited liability public company duly incorporated in South Africa. IMPLATS is listed on the Johannesburg Stock Exchange.

4.5 Royalties and Encumbrances

4.5.1 The Mineral and Petroleum Resources Royalty Act, 2008 "The Royalty Act"

The Royalty Act came into effect on 01 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 Royalty Act distinguishes between refined and unrefined Mineral Resources, where refined minerals have been refined beyond a condition specified by the Royalty Act, and unrefined minerals have undergone limited beneficiation as specified by the Royalty 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 OpEx and CapEx incurred is deductible for the determination of earnings before interest and taxes (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 OpEx and CapEx incurred is deductible for the determination of EBIT.

Following is a formula for refined Mineral Resources.

$$\text{Royalty Rate} = 0.5 + \frac{\text{EBIT}}{\text{Gross Sales (refined)} \times 12.5} \quad \text{X } 100$$

The maximum percentage for refined Mineral Resources is 5%.

Following is a formula 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%.

4.5.2 Encumbrances

No liens, pledges, mortgage bonds, or any encumbrances of any nature are registered against the Waterberg PR.

4.6 Environmental Liability during the Prospecting Phase

All environmental requirements on the properties are subject to the terms of a current Environmental Management Plan (EMP) approved by the DMR prior to commencement of work on the properties. All rehabilitation of drill hole sites and access roads required in terms of this EMP were completed. 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 are obtained. There are no known encumbrances of an environmental nature that may restrict the exploration of the properties.

4.7 Legal Access

South Africa is a country with a long-established rich mining history. South Africa has detailed regulatory framework for mining and environmental approvals. The Mining Charter as a companion to the Mining Act sets out goals for employment, procurement, and black ownership.

The country has a detailed regulatory framework of mineral title, mining right grant, and mining authorization. The MRPDA is the current minerals legislation. An update to the Mining Charter setting goals for empowerment, procurement and employment has recently be proclaimed. The National Environmental Management Act 107 1998 also has relevance to the Waterberg Project. The company will need to comply with certain empowerment, procurement and management targets to be granted a mining right. A WUL will also be required.

The Waterberg Project SLP is the document in the mining right application that discusses the relationship with the local communities. The SLP was submitted with the Mining Right Application in August 2018 and is currently being evaluated by the South African regulatory authorities. Surface rights for the mining and tailings areas must be purchased or leased from owners and communities in the area.

No reason exists at this time to cause the permissions, permits, surface, and water use rights to not be achieved; however, these factors are a significant project risk. The risk is mitigated by following the established process of consultation in the environmental assessment for a new mining right.

Waterberg JV Resources consulted with the community and received permissions to access the land where it holds PRs. Ongoing rights of access to specific portions of the property will be required as exploration and potential development progresses. Negotiations for access to land for potential infrastructure and where needed the establishment of servitudes, are ongoing.

Further details on legal access are discussed in Section 20.

4.8 Permits

Permits to support mine development activities are more fully set out in Section 20. Waterberg JV Resources is the holder of the PRs listed in Table 4-2.

All exploration activities were conducted in compliance with applicable laws in South Africa.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Access

The Waterberg Project is located some 85 km north of the town of Mokopane (formerly Potgietersrus) in Seshego and Mokerong, districts of the Limpopo Province. Mokopane provides a full spectrum of local and urban infrastructure.

The Waterberg Project is situated some 25 km from the N11 national road that links Mokopane with the Groblers Bridge border post to Botswana. Paved roads provide access to within 30 km of the Waterberg Project from the N11 National Road. Access to the area from the national road is by unpaved roads that are generally in reasonable all-weather condition.

5.2 Local Resources

Minimal service-related infrastructure exists, as the area is largely undeveloped rural farmland. Roads are “basic” and unpaved, electricity is three-phase 22 kV rural farmland supply and water is obtained from drill holes with minimal reticulation. The local population is mostly engaged in pastoral-based or weekly migrant worker-based economic activities. Local industries are limited to small-scale mechanical workshops and general dealers. A local governmental hospital falls within the reach of the Waterberg Project; however, the more serious medical cases are dealt with on a referral basis at medical facilities in the city of Polokwane.

Mining services and recruitment are readily available from Mokopane, which has a long history of mining with the Mogalakwena Mine, formerly Potgietersrus Platinum Mine (Anglo Platinum), situated north of the town. Furthermore, drilling contractors, mining services, and consultants are readily sourced within the greater Gauteng area.

5.3 Regional Infrastructure

No rail facilities service the area. Access to the site is from the national road network; however, the local roads within 34 km of the site are unpaved but provide a connection to paved provincial and national roads.

No reticulated water system is noted to exist within 25 km of site.

Surface rights, access and construction of regional infrastructure may delay the Waterberg Project. Negotiation of surface agreements is provided for in the MPRDA and regional infrastructure construction is provided for in the project plan.

5.3.1 Power

There is an existing electrical supply to the area, provided with power at 22kV by the power utility - ESKOM. This supply is sufficient for the current economic activities and can be used for construction power if upgraded.

5.3.2 Water

The current activity in the area is in the form of local people undertaking small-scale farming on a subsistence basis for cattle and crops.

Drill hole 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, which is relied upon for local community needs. Regionally there are significant wells used for agriculture at 4 ML per day or more.

The Glen Alpine Dam is located 23 km to the NW of the Waterberg Project area but does not hold enough water capacity for the Waterberg Project. The company established a cooperation agreement for access and distribution of groundwater in the area and water resources are confirmed to be present in levels required for the Waterberg Project.

5.3.3 Roads

Secondary and tertiary unpaved roads service the local villages, schools and communities. The paved N11 from Mokopane to Grobler's Bridge border post passes approximately 25 km straight line distance from site but the road access from the N11 is about 30 km 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) lead to secondary and tertiary roads which service to site and local schools and villages.

5.4 Physiography

Cliffs of Waterberg sandstones rise abruptly forming the polygonal-parallelogram shaped Makgabeng plateau from the flat to gently sloping surrounding foothills. These are surrounded by Waterberg sandstones and shales of the Makgabeng formation. Sheet-like 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.

5.4.1 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 Biodiversity Impact Assessment, 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. 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.

5.4.2 Birds

The three typical bird species with the highest frequency of occurrence on the study area include the white-bellied sunbird *(Cinnyris talatala)*, dark-capped bulbul *(Pycnonotus tricolor)*, and white-browed scrub robin *(Cercotrichas leucophrys)*.

5.4.3 Herpetofauna

A combined total of 43 species of reptiles were encountered during the two Waterberg Project herpetofaunal surveys. The occurrence of one other species (*Python natalensis*) was confirmed by means of interviews with people from the local community. The currently recorded known species richness for the study area is 44 reptile species, comprised of one chelonian, 28 lizard, and 15 snake species.

5.4.4 Mammals

Based on the total number of observations during the biodiversity study, the most frequently observed mammal is the Chacma baboon (*Papio ursinus*), with tracks and signs from the porcupine (*Hystrix africaeaustralis)*. Smith's bush squirrel (*Paraxerus cepapi*) and scrub hare (*Lepus saxatilis*) are considered sub-dominant. Other mammals frequently observed are the steenbok (*Raphicerus campestris*), bush duiker (*Sylvicapra grimmia*), black-backed jackal (*Canis mesomelas*), slender mongoose (*Herpestes sanguinea*), and yellow mongoose (*Cynictis penicillata*), with tracks and sign from brown hyena (*Parahyaena brunnea*), honey badger (*Mellivora capensis*), and aardvark (*Orycteropus afer*).

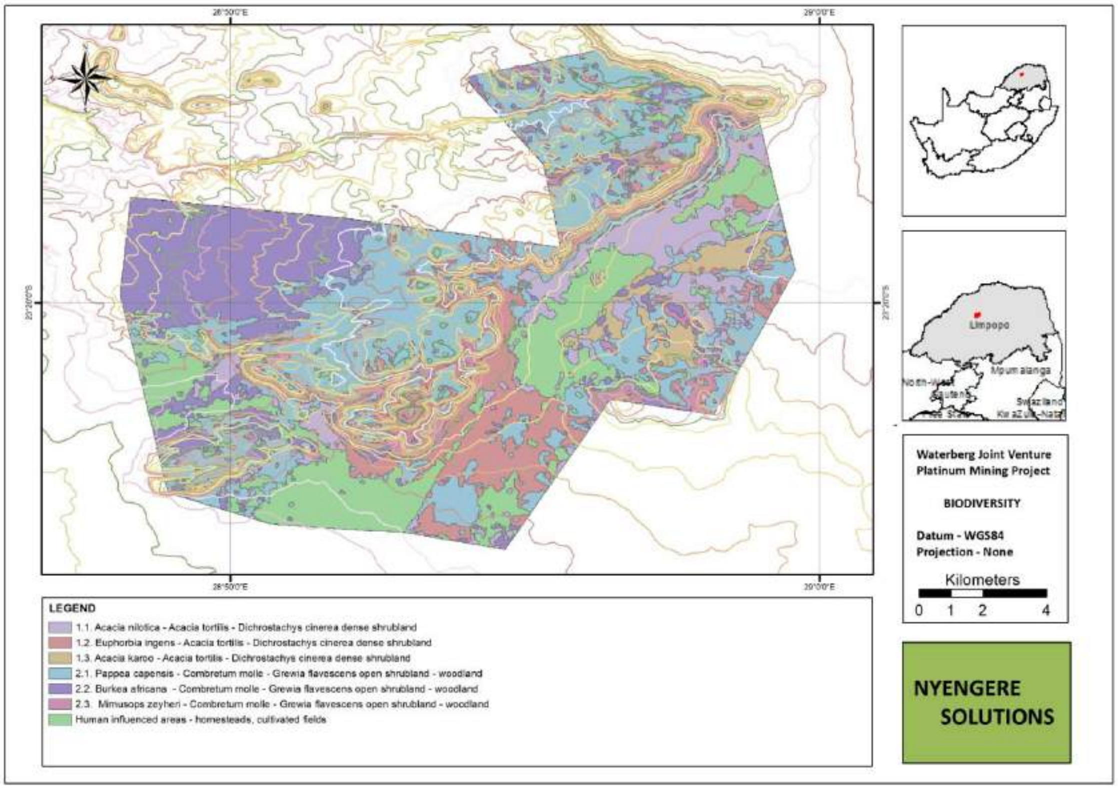
5.4.5 Vegetation

Based on the information available, it was concluded that the area is represented by the two main plant communities and six sub-communities listed below.

- Plant community 1 – *Acacia tortilis* – *Dichrostachys cinerea* – dense shrubland
  - Sub-community 1.1. – *Acacia nilotica* – *Acacia tortilis* – *Dichrostachys cinerea* – dense shrubland
  - Sub-community 1.2. – *Euphorbia ingens* – *Acacia tortilis* – *Dichrostachys cinerea* – dense shrubland
  - Sub-community 1.3. – *Acacia karoo* – *Acacia tortilis* – *Dichrostachys cinerea* – dense shrubland
- Plant community 2 – *Combretum molle* – *Grewia flavescens* – *open shrubland* – woodland
  - Sub-community 2.1. – *Pappea capensis* – *Combretum molle* – *Grewia flavescens* – open shrubland – woodland
  - Sub-community 2.2. – *Burkea africana* – *Combretum molle* – *Grewia flavescens* – open shrubland – woodland
  - Sub-community 2.3. – *Mimusops zeyheri* – *Combretum molle* – *Grewia flavescens* – open shrubland – woodland

Figure 5-1 shows the main plant communities and sub-communities for the Waterberg Project.

Figure 5-1: Waterberg Project Plant Communities and Subcommunities



5.4.6 Local Rock Art

While local legend records the presence of Bushmen rock art in the region in general, none is located within or adjacent to the Waterberg Project infrastructure area despite several scouting exercises. Local "experts" have also been unsuccessful in pointing out local rock art within or adjacent to the current infrastructure area. These sites will be protected if properly identified. No such sites have been located in the Waterberg Project development area.

5.4.7 Sites of Sensitivity in the Area

The pastoral village farming based community in the area has naturally allowed local gravesites to be developed in proximity to the homesteads and village groupings of dwellings. These were located, mapped, and demarcated for site preservation. Initial environmental assessments have located and mapped these sites in the area of the exploration work.

5.5 Climate and Length of the Operating Season

The climate is semi-arid with moderate winter temperatures and warm to hot in the summer. Temperate to Savannah, summer rainfall conditions prevail with highs reaching the low 40ºC values, but typically, mid 30ºC. Winter temperatures drop to low teens and may rarely reach single ºC temperatures.

Most of the 350-400 mm of average annual rainfall occurs in the period November to March. Climatic conditions have virtually no impact on potential mining operations in the Waterberg Project area. The dry season persists from April to mid to late September, typically. Mining and exploration activities can continue throughout the year.

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 ten years. PTM RSA 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 PRs for the properties were applied for based on the initial findings on the Waterberg Project combined with an analysis of publicly available regional government geophysical data that showed an arching north-northeast tend to the signature of the interpreted edge of the Bushveld Complex.

6.1 Exploration

The Council for Geoscience mapped the region, including the property, as presented on the 1:250 000 scale – Map No 2328 – Pietersburg. This sheet is the published geological map of the area and the basis for the metallurgical sheets, as well as regional aeromagnetic and gravity surveys that now form part of the public domain dataset.

There is no publicly available detailed exploration history available for the area. As a result of the cover rocks overlying the Bushveld Complex, it appears that no previous exploration for PGM was undertaken. The extensive exploration for PGM on the Platreef targets 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.

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 models were discarded with the current discovery and drilling data showing a strike to the north northeast.

Work completed to date includes data compilation, acquisition of satellite imagery, geological mapping, stream sediment and soil geochemical sampling, airborne geophysical survey, horizontal and longitudinal magnetic gradient, multi-channel radiometric, linear and barometric, altimetric and positional data, acquisition of whole-rock major and trace element data from selected intervals of mineralised zone, FALCON ® Airborne Gravity Gradiometer Survey and ground gravity survey, and diamond drill core drilling.

6.2 Historical Mineral Resource Estimate

6.2.1 September 2012

The initial Mineral Resource was declared in September 2012 for the T and F Zone mineralisation and is confined to only the property Ketting 368LR of the Waterberg Project. Data from the drilling completed by PTM RSA prior to September 2012 was used to undertake a Mineral Resource Estimate from more than 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. This estimate was presented in a Technical Report in 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, which was compliant with NI 43-101 standards.

Table 6-1: Waterberg Project, Mineral Resource Estimate, 01 September 2012, SAMREC Code, Inferred Mineral Resource at 2 g/t (4E) Cutoff 100% Project Basis

Cutoff = 2 g/t	Stratigraphic Thickness	Tonnage (Mt)	Pt (g/t)	Pd (g/t)	Au (g/t)	4E (g/t)	Pt:Pd: Au	4E (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	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	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 (k oz)			**2 049**	**3 733**	**806**					

Note:

- QP, Mr. K. Lomberg, Coffey Mining

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 were made up of areas of where the layers were absent due to faults, dykes, and mafic / ultramafic pegmatites.

6.2.2 February 2013

An updated Mineral Resource was declared for the T and F Zone mineralisation and confined to only the properties Ketting 368LR and Goedetrouw 366LR of the Waterberg Project. Data from the drilling completed by PTM RSA prior 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. Table 6-2 shows the Mineral Resource Statement for February 2013, which was compliant with NI 43-101 standards. This estimate was presented in a Technical Report 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)*".

Table 6-2: Waterberg Project Mineral Resource Estimate, 01 February 2013, SAMREC Code, Inferred Mineral Resource 2g/t (2PGE+Au) Cutoff 100% Project Basis

Cutoff = 2 g/t	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	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	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**					

Note:

- QP, Mr. K Lomberg, Coffey Mining

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 were made up of areas of where the layers were absent due to faults, dykes, potholes and mafic / ultramafic pegmatites.

6.2.3 September 2013

A Mineral Resource was declared for the T and F Zone mineralisation and confined to only the properties Ketting 368LR and Goedetrouw 366LR of the Waterberg Project. Data from the drilling completed by PTM RSA prior to 01 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, which was compliant with NI 43-101 standards. The data and the geological understanding and interpretation were considered of sufficient quality for the declaration of an inferred Mineral Resource. This estimate was presented in a Technical Report 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*."

Table 6-3: Waterberg Project-Mineral Resource Estimate, 02 September 2013, SAMREC Production Code, Inferred Mineral Resource 2g/t (4E) Cutoff 100% Project Basis

Cutoff = 2 g/t	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	2.30	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		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**					

Notes:

- Cutoff applied on 2PGE+Au grade
- QP, Mr. K. Lomberg, Coffey Mining

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 were made up of areas of where the layers were absent due to faults, dykes, potholes and mafic / ultramafic pegmatites.

Insufficient drilling was completed to support a Mineral Resource Estimate in September 2013 for the Waterberg Extension Project.

6.2.4 June 2014

The Waterberg Project was further advanced in exploration status and includes an Inferred Mineral Resource Estimate that was included in the Mineral Resource Statement in June 2014. The majority of the Waterberg Extension Project was still at an early exploration stage; however, drilling on the property Early Dawn 361LR just north of the Waterberg Project had enough surface drilling to confirm continuity of mineralisation, hence 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 flagged 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 FH1-FH6) within the F Zone were identified. It is the identification of these layers and the classification of historical exploration data to fit this 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 cutoff grade of 2 g/t PGM and a minimum thickness of 2 m.

Data from 138 drill holes was included in the database. 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 RSA, using the Strat 3D module of CAE Mining Studio™.

The coded drill hole database supplied by PTM RSA was composited for Pt, Pd, Au, Cu, Ni and density. For each unit a 3D 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. This estimate was presented in a Technical Report in June 2014 by Mr. KG Lomberg and Mr. AB Goldschmidt; entitled "*Technical Report for the Update on Exploration Drilling at the Waterberg Joint Venture and Waterberg Extension Projects, South Africa*." Table 6-4 shows the Mineral Resource Statement for June 2014, which was compliant with NI 43-101 standards.

Table 6-4: Waterberg Project-Mineral Resource Estimate (SAMREC Code) (12 June 2014) SAMREC Code, Inferred Mineral Resource 2 g/t (2PGE+Au) Cutoff 100% Project Basis

Cutoff=2 g/t	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	2PGE + Au (g/t)	Pt:Pd:Rh:Au	2PGE + Au (koz)	Cu (%)	Ni (%)	Cu (Mlbs)	Ni (Mlbs)
Waterberg Project Totals for both the JV and the Extension													
T1	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	2.75-60	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**		
Waterberg Project- (JV)													
	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	2PGE + Au (g/t)	Pt:Pd:Rh:Au	2PGE + Au (koz)	Cu (%)	Ni (%)	Cu (Mlbs)	Ni (Mlbs)
T1	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	2.75-60	164.58	0.88	1.91	0.05	0.13	2.97	30:64:2:5	15 713	0.07	0.18	247	649
Total	**2.44**	**218.64**	**0.94**	**1.91**	**0.03**	**0.29**	**3.17**	**30:60:1:9**	**22 268**	**0.09**	**0.16**	**455**	**763**
Content (koz)			**6 605**	**13 407**	**239**	**2 018**			**kt**	**206**	**346**		
Waterberg Project- (Extension)													
	Stratigraphic Thickness	Tonnage Mt	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	2PGE + Au (g/t)	Pt:Pd:Rh:Au	2PGE + Au (koz)	Cu (%)	Ni (%)	Cu (Mlbs)	Ni (Mlbs)
F(Cutoff=2g/t)	2.76-60	68.04	0.93	1.98	0.05	0.15	3.11	30:64:2:5	6 802	0.11	0.23	162	344
Total		**68.04**	**0.93**	**1.98**	**0.05**	**0.15**	**3.11**	**30:64:2:5**	**6 802**	**0.11**	**0.23**	**162**	**344**
Content (koz)			**2 043**	**4 325**	**102**	**331**			**kt**	**73**	**156**		

Notes:

- Cutoff applied on 4E grade
- QP, Mr. K Lomberg, Coffey Mining

6.2.5 July 2015

On 20 July 2015, the company declared a Mineral Resource Estimate for the Waterberg Project that include the JV and Extension areas combined. Infill drilling over portions of the Waterberg Project area and a revised estimation approach made it possible to update the Mineral Resource Estimate and to upgrade portions of the Mineral Resource to the Indicated category. Data used in this estimate comprised 220 original drill holes of the 231 with 270 deflections of the 374 drilled. Of these, 89 intersections occurred in the T Zone ranging from approximately 140 m to 1 380 m in depth below surface. A total of 365 intersections in the F Zone were used ranging from approximately 200 m to 1 250 m in depth. This estimate was presented in a Technical Report in July 2015 by Charles Muller; entitled "*An independent technical report on the Waterberg Project located in the Bushveld Igneous Complex, South Africa*." Table 6-5 shows the Mineral Resource Statement for July 2015, which was compliant with NI 43-101 standards.

Table 6-5: Summary of Mineral Resource Estimate Effective 20 July 2015 on 100% Project Basis

T Zone 2.5 g/t Cutoff										
Mineral Resource Category	Cutoff	Tonnage	Grade						Metal	
	2PGE + Au		Pt	Pd	Au	2PGE + Au	Cu	Ni	2PGE + Au	
	g/t	Mt	g/t	g/t	g/t	g/t	%	%	kg	Moz
Indicated	2.5	16.53	1.28	2.12	0.85	4.25	0.16	0.09	70 253	2.26
Inferred	2.5	33.56	1.25	2.09	0.83	4.17	0.13	0.08	139 945	4.50
F Zone 2.5 g/t Cutoff										
Mineral Resource Category	**Cutoff**	**Tonnage**	**Grade**						**Metal**	
	2PGE + Au		**Pt**	**Pd**	**Au**	**2PGE + Au**	**Cu**	**Ni**	**2PGE + Au**	
	g/t	**Mt**	**g/t**	**g/t**	**g/t**	**g/t**	**%**	**%**	**kg**	**Moz**
Indicated	2.5	104.47	0.93	2.00	0.15	3.08	0.06	0.16	321 768	10.35
Inferred	2.5	212.75	0.93	2.01	0.15	3.09	0.07	0.17	657 398	21.14

Note:

- QP, Charles Muller, CJM

6.2.6 April 2016

On 18 April 2016, the company declared an updated Mineral Resource Estimate for the Waterberg Project. This estimate was presented in a Technical Report in April 2016 by Mr. Charles Muller; entitled "*Mineral Resource Update on the Waterberg Project located in the Bushveld Igneous Complex, South Africa*." Table 6-6 shows the Mineral Resource Statement for April 2016, which was compliant with NI 43-101 standards.

Table 6-6: Mineral Resource Estimate Details as at 18 April 2016

F Zone								
Cutoff	Tonnage	Grade					Metal	
4E		Pt	Pd	Au	Rh	3PGE+Au	4E	
g/t	Mt	g/t	g/t	g/t	g/t	g/t	kg	Moz
Indicated								
2.00	281.184	0.91	1.94	0.15	0.03	3.03	851 988	27.392
2.50	179.325	1.05	2.23	0.18	0.03	3.49	625 844	20.121
3.00	110.863	1.19	2.52	0.20	0.04	3.95	437 909	14.079
Inferred								
2.00	177.961	0.83	1.77	0.13	0.03	2.76	491 183	15.792
2.50	84.722	1.01	2.14	0.17	0.03	3.35	283 819	9.125
3.00	43.153	1.19	2.53	0.20	0.04	3.96	170 886	5.494
T Zone								
Cutoff	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								
2.00	36.308	1.08	1.81	0.72	-	3.61	131 162	4.217
2.50	30.234	1.16	1.94	0.78	-	3.88	117 363	3.773
3.00	22.330	1.28	2.14	0.86	-	4.28	95 640	3.075
Inferred								
2.00	23.314	1.10	1.83	0.73	-	3.66	85 240	2.741
2.50	21.196	1.14	1.90	0.76	-	3.79	80 394	2.585
3.00	14.497	1.28	2.14	0.86	-	4.28	62 082	1.996
Waterberg Total								
Cutoff	Tonnage	Grade					Metal	
4E		Pt	Pd	Au	Rh	3PGE+Au	4E	
g/t	Mt	g/t	g/t	g/t	g/t	g/t	kg	Moz
Indicated								
2.00	317.492	0.93	1.92	0.22	0.03	3.10	983 150	31.609
2.50	209.559	1.07	2.19	0.26	0.03	3.55	743 207	23.894
3.00	133.193	1.21	2.46	0.31	0.03	4.01	533 549	17.154
Inferred								
2.00	201.275	0.85	1.77	0.21	0.03	2.86	576 423	18.533
2.50	105.918	1.04	2.09	0.28	0.03	3.44	364 213	11.710
3.00	57.650	1.21	2.43	0.37	0.03	4.04	232 968	7.490

Notes:

- 2PGE+Au = PGE (Pt+Pd) and Au
- 4E (Pt+Pd+Rh) and Au
- Conversion Factor used – kg to oz = 32.15076
- Numbers may not add due to rounding.
- QP, Charles Muller, CJM

6.2.7 October 2016

On 17 October 2016, the company declared a Mineral Resource Estimate for the Waterberg Platinum Project, the that includes the JV and Extension areas combined.

Infill drilling over portions of the Waterberg Project area and new estimation methodology has made it possible to estimate a new Mineral Resource Estimate and upgrade portions of the Mineral Resource Estimate to the Indicated category. This estimate was presented in a Technical Report in October 2016 by Robert L. Goosen, Charles J Muller, et al.; entitled "*Independent Technical Report on the Waterberg Project Including Mineral Resource Update and Prefeasibility Study.*" Table 6-7 shows the T Zone Mineral Resource Statement and Table 6-8 shows the F Zone Mineral Resource Statement for October 2016, both of which are compliant with NI 43-101 standards.

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.

Table 6-7: T Zone Mineral Resource Estimate at 2.5g/t 4E Cutoff (as of 17 October 2016)

T Zone 2.5g/t Cutoff											
Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Au	Rh	4E	Cu	Ni	4E	
	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Indicated	2.5	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 6-8: F Zone Mineral Resource Estimate at 2.5g/t 4E Cutoff (as of 17 October 2016)

F Zone 2.5g/t Cutoff											
Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Au	Rh	4E	Cu	Ni	4E	
	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Indicated	2.5	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

Notes:

- 4E = PGE (Pt+Pd+Rh) and Au – the cutoffs for Mineral Resources were established by a QP 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.
- The 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 were applied to the indicated and inferred categories, respectively.

Table 6-9 summarises the combined Mineral Resource Statement.

Table 6-9: Total Mineral Resource Estimate at 2.5g/t 4E Cutoff (as of 17 October 2016)

Waterberg Total 2.5g/t Cutoff											
Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Au	Rh	4E	Cu	Ni	4E	
	g/t	Mt	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Indicated	2.5	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

Notes:

- A cutoff grade of 2.5g/t 4E
- QP, is Charles Muller, CJM

6.2.8 September 2018

On 27 September 2018, the company declared a Mineral Resource Estimate for the Waterberg Project. Infill drilling over portions of the Waterberg 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 Measured category. All the JV partners have been involved in the development of the latest Mineral Resource Model, appropriate cutoff grades, economic parameters and Mineral Resource Model criteria. This estimate was presented in a Technical Report in September 2018 by Charles J Muller; entitled "*Technical Report on the Mineral Resource Update for the Waterberg Project Located in the Bushveld Igneous Complex, South Africa.*". It was determined in relation to basic working costs and in consideration of the overall resource envelope for the deposit, that at a 2.0 g/t cutoff grade the deposit has a reasonable prospect of economic extraction. Table 6-10 shows the Mineral Resource Statement at a 2.0 g/t (4E) cutoff for September 2018, which was compliant with NI 43-101 standards.

For purposes of the DFS, sensitivity analysis and comparison to the 2016 PFS, which utilised a 2.5 g/t 4E cutoff grade, a Mineral Resource Estimate at a 2.5 g/t cutoff grade is the preferred scenario. Table 6-11 shows the Mineral Resource Statement at a 2.0 g/t (4E) cutoff for September 2018, which was compliant with NI 43-101 standards.

Table 6-10: Summary of Mineral Resource Estimate effective 27 September 2018 on a 100% Project Basis at 2.0 g/t (4E) Cutoff

T Zone at 2.0 g/t (4E) Cutoff

Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	**2.0**	3 440 855	1.13	1.97	0.04	0.90	4.04	0.160	0.080	13 901	0.447
Indicated	**2.0**	22 997 505	1.22	2.06	0.03	0.79	4.10	0.186	0.090	94 290	3.031
M+I	**2.0**	**26 438 360**	**1.21**	**2.05**	**0.03**	**0.80**	**4.09**	**0.183**	**0.089**	**108 191**	**3.478**
Inferred	**2.0**	25 029 695	1.17	1.84	0.03	0.60	3.64	0.137	0.069	91108	2.929

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.0	48.8	1.0	22.2
Indicated	29.8	50.2	0.7	19.3
M+I	**29.6**	**50.0**	**0.7**	**19.7**
Inferred	32.1	50.5	0.8	16.6

F Zone at 2.0 g/t (4E) Cutoff

Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	**2.0**	75 332 513	0.82	2.00	0.05	0.14	3.01	0.079	0.191	226 833	7.293
Indicated	**2.0**	273 272 480	0.80	1.85	0.04	0.14	2.83	0.073	0.181	772 103	24.824
M+I	**2.0**	**348 604 993**	**0.80**	**1.88**	**0.04**	**0.14**	**2.87**	**0.075**	**0.183**	**998 936**	**32.117**
Inferred	**2.0**	121 535 227	0.70	1.62	0.04	0.13	2.50	0.067	0.162	303 722	9.765

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.2	66.4	1.7	4.7
Indicated	28.3	65.4	1.4	4.9
M+I	**28.0**	**65.6**	**1.5**	**4.9**
Inferred	28.4	64.8	1.6	5.2

Waterberg Aggregate Total 2.0 g/t Cutoff September 2018 100% Project Basis

Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	T	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	**2.0**	78 773 368	0.83	2.00	0.05	0.18	3.06	0.083	0.186	240 734	7.740
Indicated	**2.0**	296 269 985	0.83	1.86	0.04	0.19	2.92	0.082	0.174	866 393	27.855
M+I	**2.0**	**375 043 353**	**3.00**	**1.89**	**0.04**	**0.19**	**2.95**	**0.083**	**0.176**	**1 107 127**	**35.595**
Inferred	**2.0**	146 564 922	0.78	1.66	0.04	0.21	2.69	0.079	0.146	394 830	12.694

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.1	65.4	1.6	5.9
Indicated	28.4	63.7	1.4	6.5
M+I	**28.1**	**64.1**	**1.4**	**6.4**
Inferred	29.0	61.7	1.5	7.8

Table 6-11: Summary of Mineral Resource Estimate effective 27 September 2018 on a 100% Project Basis at 2.5 g/t (4E) Cutoff

T Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	3 098 074	1.19	2.09	0.05	0.90	4.23	0.160	0.090	13 105	0.421
Indicated	2.5	18 419 181	1.34	2.31	0.03	0.87	4.55	0.197	0.095	83 807	2.694
M+I	**2.5**	**21 517 255**	**1.32**	**2.28**	**0.03**	**0.88**	**4.51**	**0.192**	**0.094**	**96 912**	**3.116**
Inferred	2.5	21 829 698	1.15	1.92	0.03	0.76	3.86	0.198	0.098	84 263	2.709

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.1	49.4	1.2	21.3
Indicated	29.5	50.7	0.7	19.1
M+I	**29.3**	**50.6**	**0.7**	**19.4**
Inferred	29.8	49.7	0.8	19.7

F Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	54 072 600	0.95	2.20	0.05	0.16	3.36	0.087	0.202	181 704	5.842
Indicated	2.5	166 895 635	0.95	2.09	0.05	0.15	3.24	0.090	0.186	540 691	17.384
M+I	**2.5**	**220 968 235**	**0.95**	**2.12**	**0.05**	**0.15**	**3.27**	**0.089**	**0.190**	**722 395**	**23.226**
Inferred	2.5	44 836 851	0.87	1.92	0.05	0.14	2.98	0.064	0.169	133 705	4.299

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.3	65.4	1.5	4.8
Indicated	29.3	64.4	1.6	4.7
M+I	**29.1**	**64.8**	**1.5**	**4.6**
Inferred	29.2	64.4	1.7	4.7

Waterberg Aggregate Total 2.5 g/t Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	T	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	57 170 674	0.96	2.19	0.05	0.20	3.40	0.091	0.196	194 809	6.263
Indicated	2.5	185 314 816	0.99	2.11	0.05	0.22	3.37	0.100	0.177	624 498	20.078
M+I	2.5	242 485 490	0.98	2.13	0.05	0.22	3.38	0.098	0.181	819 307	26.342
Inferred	2.5	66 666 549	0.96	1.92	0.04	0.34	3.26	0.108	0.146	217 968	7.008

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	**%**	**%**	**%**
Measured	28.2	64.4	1.5	5.9
Indicated	29.4	62.6	1.5	6.5
M+I	**29.2**	**63.0**	**1.4**	**6.4**
Inferred	29.5	58.9	1.2	10.4

Notes:

- 4E = PGE (Pt+Pd+Rh) and Au.
- The cutoffs for Mineral Resources were established by a QP 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 entity.
- Conversion factor used – kg to oz = 32.15076.
- Numbers may not add due to rounding.
- A 5% and 7% geological loss were applied to the Measured / Indicated and Inferred Mineral Resource categories, respectively.

6.3 Historical Mineral Reserves Estimate – October 2016

On 17 October 2016, the first Mineral Reserve was declared for the Waterberg Project. The conversion to Mineral Reserves was undertaken initially at 3.0 g/t and the 2.5 g/t 4E stope cutoff 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. There are no inferred Mineral Resources included in the Mineral Reserves.

The project had a production rate of 600 ktpm, utilizing the following three mining methods which were selected for the Waterberg Project.

- Blind Longitudinal Retreat
- Transverse Sublevel Open Stoping
- Longitudinal Sublevel Open Stoping (LSLOS)

None of these methods utilised backfill and all stopes were left void after mining. This estimate was presented in a technical report in October 2016 by Robert L. Goosen, et al.; entitled "*Independent Technical Report on the Waterberg Project Including Mineral Resource Update and Prefeasibility Study*." All Mineral Reserves were classified as probable and no proved Mineral Reserves were declared. Table 6-12 shows the Mineral Reserves Statement at a 2.5 g/t (4E) cutoff for October 2017, which was compliant with NI 43-101 standards. Table 6-13 shows the Mineral Reserves Statement for contained metals as of 17 October 2016.

Table 6-12: Probable Mineral Reserve Estimate at 2.5 g/t – Tonnage and Grades (as of 17 October 2016)

Waterberg Probable Mineral Reserve – Tonnage and Grades									
Zone	Mt	Cutoff 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 6-13: Probable Mineral Reserve Estimate at 2.5 g/t – Contained Metal (as of 17 October 2016)

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**

Note:

- QP, is R.L. Goosen, WorleyParsons RSA (Pty) Ltd.

6.4 Production History

There is no historic production from the Waterberg Project.

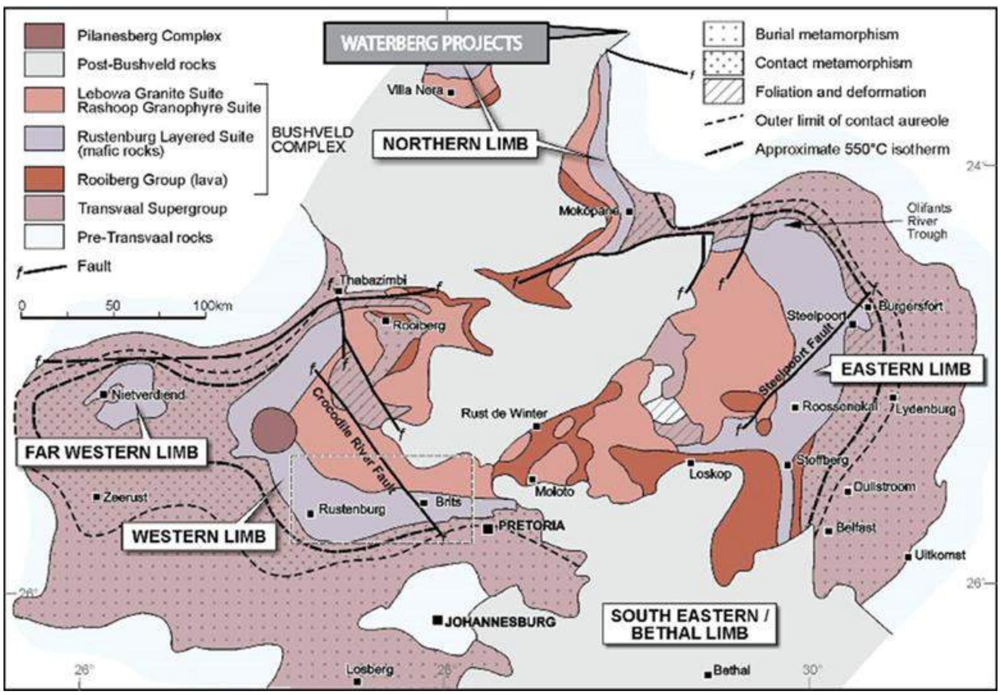
7 GEOLOGICAL SETTING, MINERALISATION, AND DEPOSIT TYPES

7.1 Geological 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. It is estimated to exceed 66 000 km^2 in extent, of which about 55% is covered by younger formations. The Bushveld Complex hosts several layers rich in PGM, Cr, and V, and constitutes the world's largest known Mineral 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 of the Bushveld Complex as shown in Figure 7-1.

Figure 7-1: Geological Map of the Bushveld Complex Showing the Location of the Waterberg Project



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 that marks the southern boundary of the South Marginal Zone (SMZ).

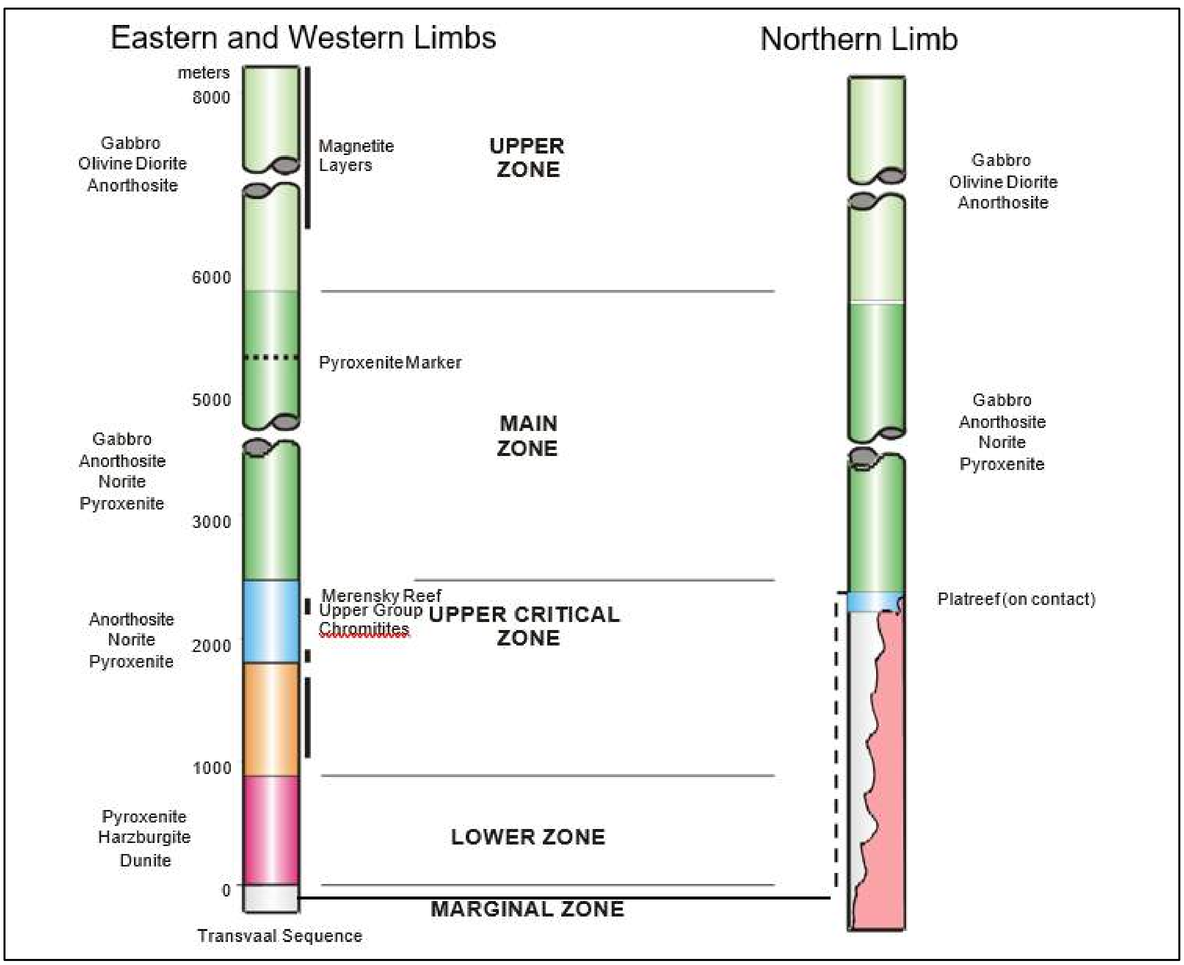
The SMZ is a 3 500 mega annum (Ma) aged compressional terrain formed within the Kaapvaal Craton during the collision with the Zimbabwe Craton. It is comprised of granulite facies granitic gneiss, amphibolitic gneiss, and minor quartzite. Within the SMZ, there are several major shears that trend parallel to the Hout River Shear (van Reenen, 1992) and trend through the Waterberg Project area. The footwall to the Bushveld on Waterberg Project is interpreted to be comprised of facies of the SMZ.

The Platreef characterises the geology of the Northern Limb of the Bushveld. It was first described by Van der Merwe (Van der Merwe, 1976). The Platreef is typically a wide, up to hundreds of metres, pyroxenite hosted zone of elevated Cu and Ni mineralisation with associated anomalous PGM concentrations. The sulphide mineralisation is typically pyrrhotite, chalcopyrite and pentlandite. It was postulated that the interaction with the basement rocks and the dolomites was instrumental in the formation of the mineralisation (Vermaak and Van der Merwe, 2000).

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 as shown in 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 The Northern Limb

The Northern Limb is a north-south striking sequence of igneous rocks of the Bushveld Complex with a length of 110 km and a maximum width of 15 km as shown in Figure 7-3 and Figure 7-4. It is generally divided up into three different sectors (Southern, Central and Northern), which have characteristic footwalls.

- The Southern Sector is characterised by a footwall of the Penge Formation of the Transvaal Supergroup.
- The Central Sector generally has a footwall of Malmani Subgroup.
- The Northern Sector has a footwall consisting of Archaean granite.

Figure 7-3: General Geology of the Northern Limb of the Bushveld Complex

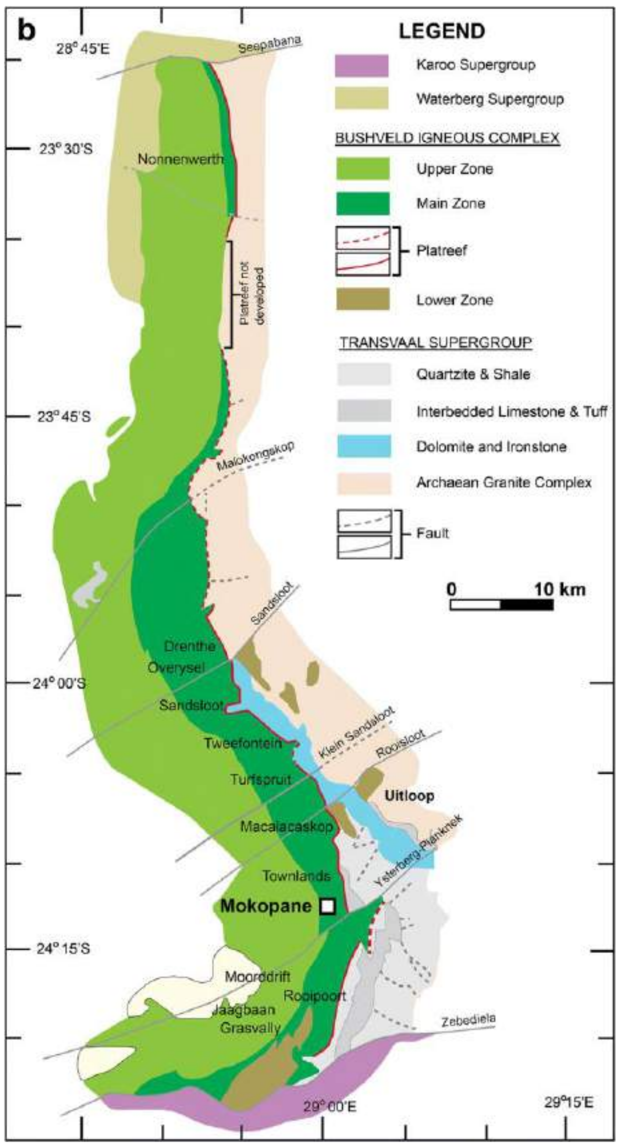
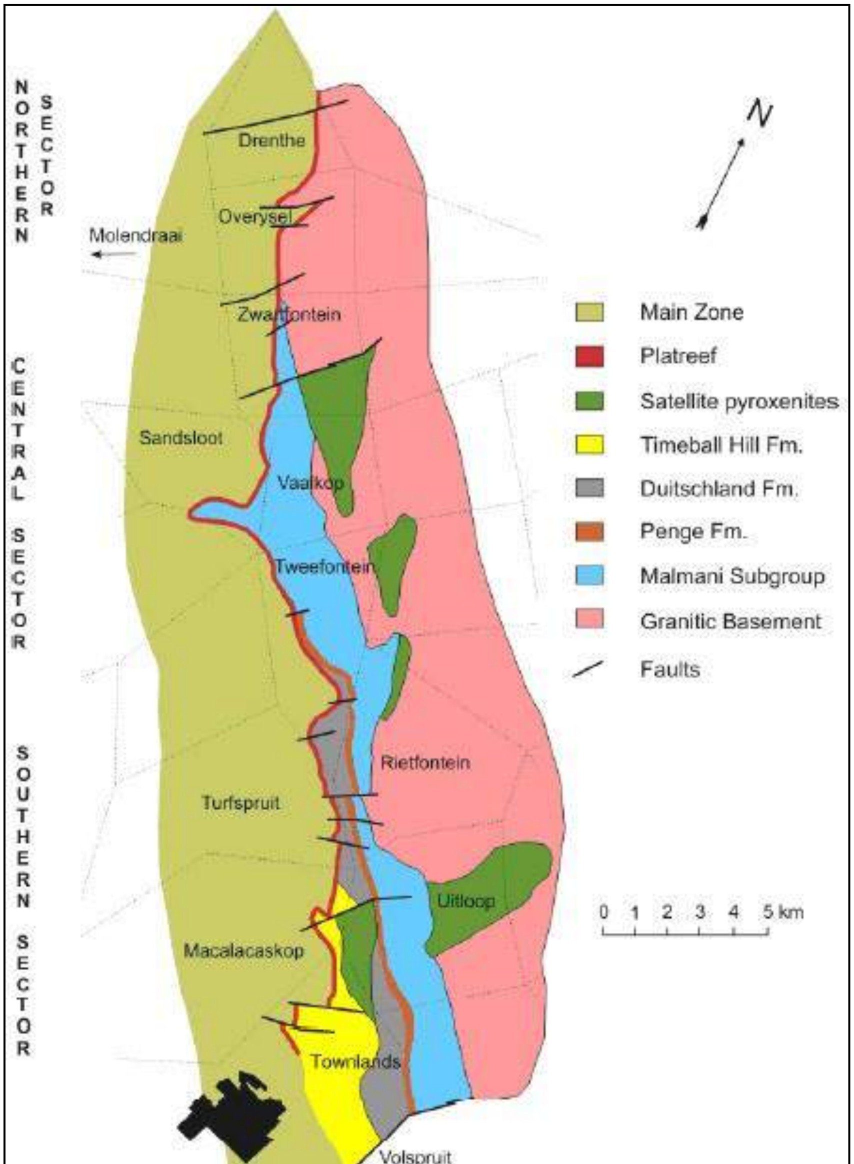


Figure 7-4: Geology of the Northern Limb of the Bushveld Complex Showing the Various Footwall Lithologies



Source: Sharman-Harris (2006)

7.1.3 Waterberg Group / Bushveld Complex Age Relationship

In general, the contact between the Waterberg Group and the weathered Bushveld Complex was observed in the drill hole core to be sharp. In several of the drill intersections, conglomerate and grit horizons are developed on the contact and appear to contain altered magnetite, suggesting the development of placer mineralisation. If present, such mineralisation is likely to be channelised, as the basal deposits appear to be fluvial. The atypical contact zone between the two rock units was examined by Professor McCarthy (McCarthy, 2012) and is interpreted as a palaeosol (fossilised soil) developed on the Bushveld gabbros. Features in the palaeosol reminiscent of modern weathering of Bushveld rocks were observed.

The weathering is considered typically spheroidal in character and finishes in a very fine-grained upper black turf layer (vertisol), corresponding to the ‘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 mineralisation in the gabbros (T or F Zones) (McCarthy, 2012).

Professor McCarthy observed that the northern extremity of the Northern Limb of the Bushveld Complex contains a well-developed Platreef horizon, but in addition, has mineralisation developed in the Upper Zone. The T Zone has a high Cu / Ni ratio and is Pd and Au dominated. Sulphides like 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). In addition, the layered sequence in the north is underlain by quartzite which appears to be a correlative of the upper Pretoria Group. This being the case, Professor McCarthy considers that there is the potential for the development of an extensive Bushveld sub-basin beneath the Waterberg which is also supported by a local gravity high in the area.

In the project area, the Waterberg Sedimentary package occurs mostly with the Makgabeng and Setlaole Formations. The whole package may have a thickness varying from 120 m to slightly over 760 m. Generally, the Waterberg Sedimentary package thickens in the southwest and thins towards the centre of the project area before thickening again to the north. The east-west trending feature through the southern part of the Waterberg Project is thought to be an erosional channel.

7.1.3.1 Setlaole Formation

This is the sedimentary formation underlying the Makgabeng Formation and occurs at the base of the Waterberg Group sedimentary succession. It is this formation that overlies the Bushveld Complex igneous rocks, and it was intersected in more than 90% of the drill holes within the Waterberg Project area.

Lithologically, the Setlaole Formation consists of medium to coarse grained sandstones and several mudstones and shales, that have a general purple colour 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 conglomerates. The rocks are frequently intruded by dolerite and granodiorite sills. A red shale band of variable thickness is generally present at the base of the Setlaole Formation, below the basal conglomerate.

7.1.3.2 Makgabeng Formation

This sedimentary formation overlies the Setlaole Formation and is mostly exposed in the mountain cliffs in the northern part of the Waterberg Project area. The formation is composed of light- red coloured banded sandstone rocks and is generally flat lying.

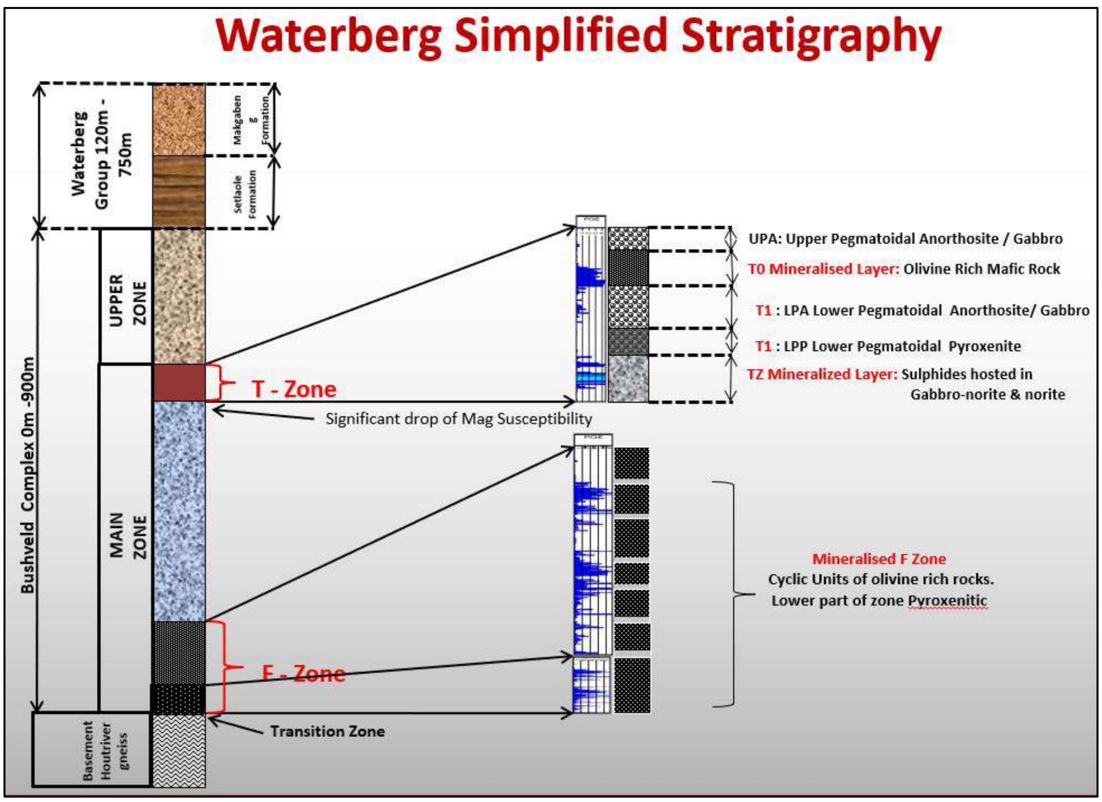
7.2 Nature of, and Controls on, Mineralisation

The Critical Zone of the Bushveld Complex hosts most of the PGE mineralisation in the Bushveld Complex and is characterised 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 mineralisation, north of Mokopane (formerly Potgietersrus), contains a wide zone of more disseminated style Pt mineralisation, along with higher grades of Ni and Cu than occur in the rest of the Bushveld Complex.

7.3 Geological Models

The initial phase of diamond exploration drilling (WB001 and WB002) during the Waterberg JV Project intersected Waterberg Group Sediments (sandstones) and Bushveld Upper Zone and Main Zone lithologies in the western portion of the Disseldorp 369 LR farm property. The follow-up drilling campaign revealed a generalised schematic stratigraphic section that was adopted for use in this property as presented in Figure 7-5.

Figure 7-5: Waterberg Simplified Stratigraphy



The initial phase of diamond exploration drilling on the farm Early Dawn 361LR intersected Waterberg Group Sediments (sandstones) and Bushveld Complex Main Zone lithologies. This indicates similar stratigraphy to the sequence in the south and, in general, the layers correlate well across farms.

The floor rocks underlying the Transitional Zone shown in Figure 7-5 are predominantly granite gneiss hosting remnants of magnetite quartzite, metaquartzite, metapelites, serpentinites and metasediments. Some drill holes within the Waterberg Project area have shown dolerite intrusions within the floor rocks, such as drill hole WB028.

Bushveld Complex lithologies underlie the Waterberg Group starting with the Upper Zone and underlain by the Main Zone.

7.3.1 The Main Zone

The 150 m to 900 m thick Main Zone hosts the PGM mineralised layers in its cyclic sequences of mafic and felsic rocks. It is largely composed of gabbronorite, norite, pyroxenite, harzburgite, and 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 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.

7.3.2 The Upper Zone

The southwestern part of the Waterberg Project area (west of the farm Ketting 368LR towards the farm Disseldorp 369 LR) has a thick package of Upper Zone lithologies. The package in the project consists of magnetite gabbro, mela-gabbronorite and magnetite seams and may be as thick as 350 m. Drill hole WB001 on the farm Disseldorp 369 LR collared in Upper Zone and drilled to the depth of 322 m and while still in the Upper Zone intersected a 2.5 m thick magnetite seam.

The appearance of the first non-magnetic mafic lithologies indicates the start of the underlying Main Zone.

7.3.3 Structure

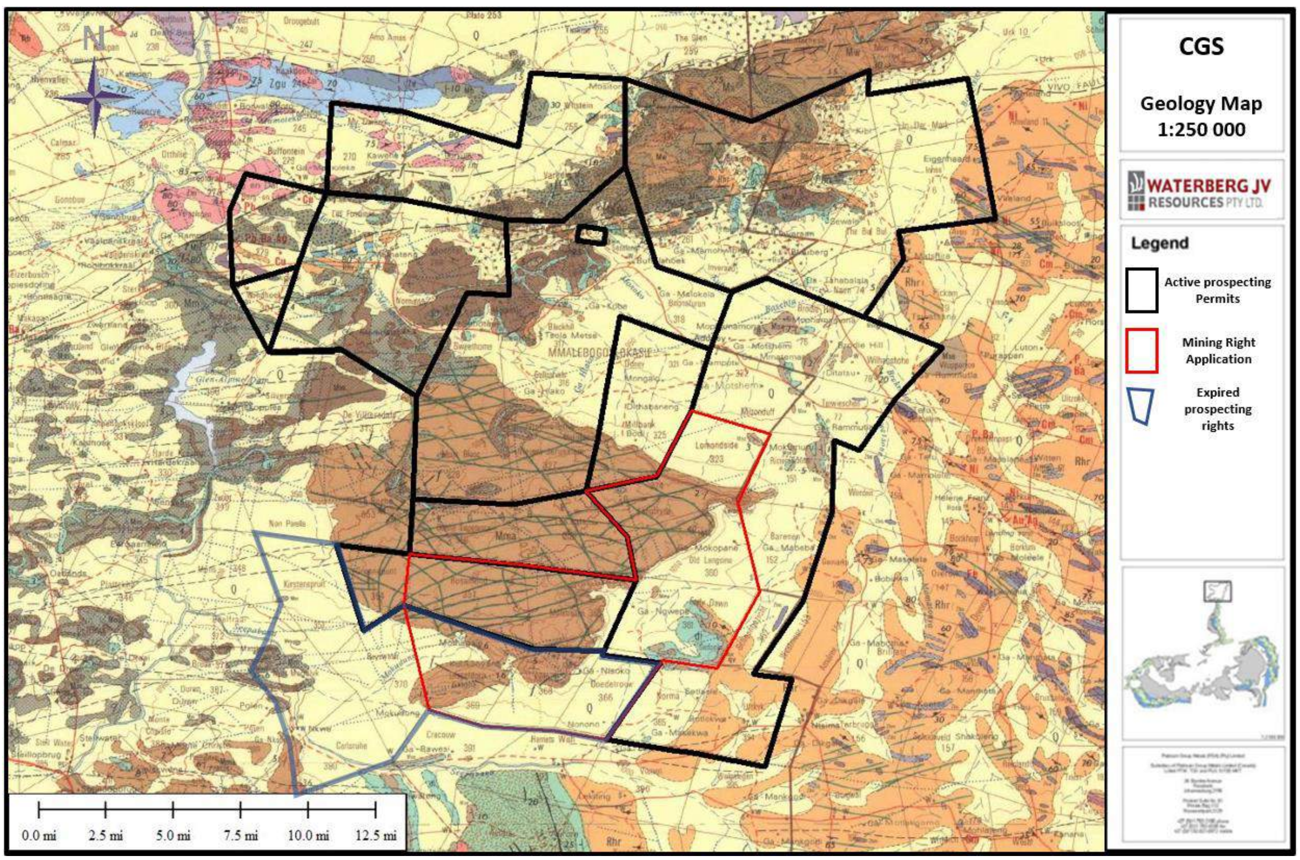
The Waterberg Sedimentary package is intersected by numerous crisscrossing dolerites or granodiorite sills or dykes. These usually range from as thin as 5 cm to as thick as 90 m.

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 300 m. A further fault splay has also been interpreted on the south-eastern part of Ketting 368LR.

7.4 Nature of Deposits on the Property

The Waterberg Project is located along the strike extension of the Northern Limb of the Bushveld Complex. The surface geology is depicted in Figure 7-6. The Bushveld Geology 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 hole WB001 and WB002. The Lower Magnetite Layer of the Upper Zone was intersected on the south of the Waterberg Project property (Disseldorp 369 LR)) where drill hole WB001 was drilled and intersected a 2.5 m thick magnetite band.

Figure 7-6: The Surface Geology of the Waterberg Project



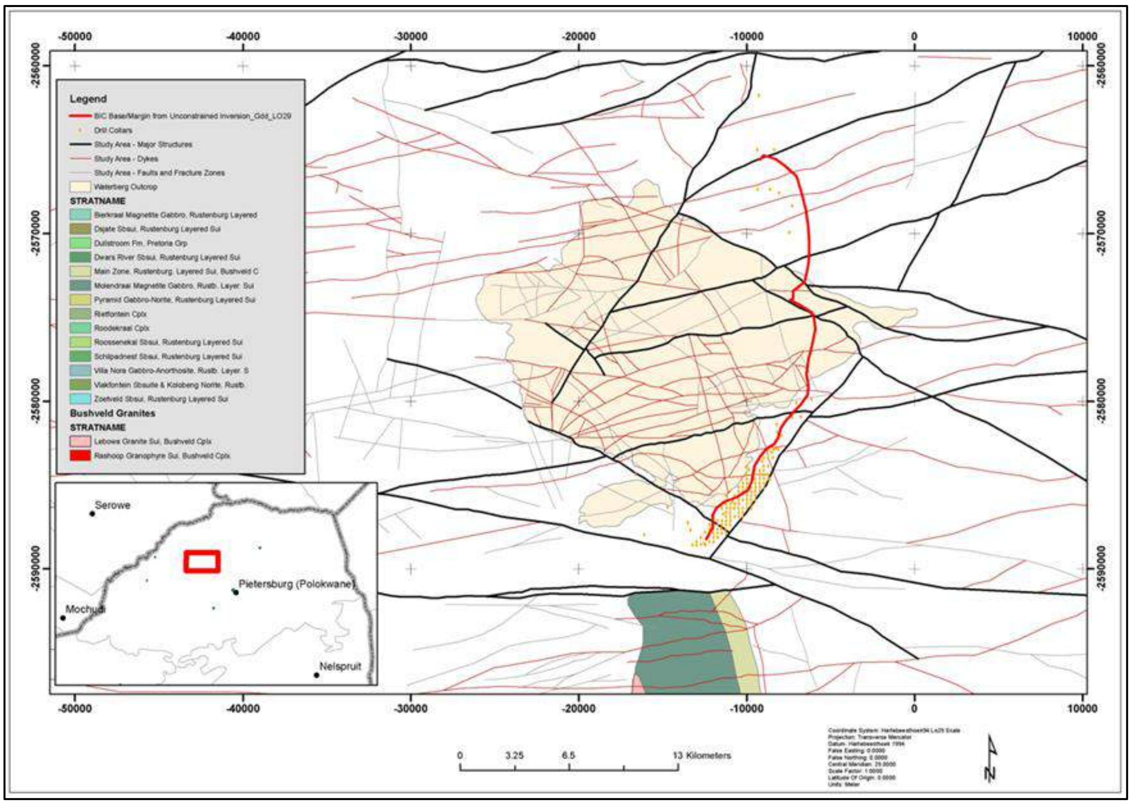
The Bushveld package strikes southwest to northeast with a general dip of 34º - 38º towards the west as observed from drill hole core for the layered units intersected on Waterberg property within the Bushveld package. Some blocks may be tilted at different angles depending on structural and /or tectonic controls.

The Bushveld Upper Zone is overlain by a 120 m to 760 m 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 Main Zone package is marked by the presence of a transitional zone that constitutes a mixed zone of Bushveld and altered sediments / quartzites before intersecting the Transvaal Basement Quartzite and Metasediments.

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 northwest and southwest trending as can be seen in Figure 7-7.

Figure 7-7: Project Geology of the Waterberg Project



The project geology in the north-eastern portion of the Waterberg JV Project appears to be similar to the geology in the southeast; however, due to the widely spaced drilling further north, the project geology is not as 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 300 m. 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 granodiorite sill with average thicknesses of 80 m appears to be exploiting the contact between the Bushveld Complex igneous rocks and the overlying Waterberg sedimentary rocks. This sill, as seen in drill hole intercepts, 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 Complex edge that explains the vertical difference in the position of the sill between surface and the projection from drill hole intersections.

8 DEPOSIT TYPES

The Platreef-type deposits can include the following features.

- Sulphide-hosted Ni, Cu, and PGM mineralisation considered to be of magmatic origin.
- A deposit hosted by a composite of norite, pyroxenite, and harzburgite rocks.
- Contact style mineralisation along the base of the intrusion, which may be several hundreds of metres thick.
- The mineralised rocks contain locally abundant xenoliths of floor rocks (typically dolomite and shale) suggesting interaction of the magma with relatively reactive floor rocks.
- Thick mineralised intervals greater than 5 m and locally tens to hundreds of metres thick.

The mineralised layers of the Waterberg Project meet some these criteria:

- The mineralisation is hosted by sulphides that are apparently magmatic in origin.
- The mineralised layers can be relatively thick, often greater than 10 m.

The other criteria relating to the Platreef have yet to be demonstrated. Consequently, this mineralisation is deemed to be similar, i.e. Platreef-like, but its stratigraphic position, geochemical and lithological profiles suggest a type of mineralisation not previously recognised in the Bushveld Complex.

8.1 Mineralisation Zones

PGM mineralisation within the Bushveld package underlying the Waterberg Project is hosted in two main layers: T Zone and 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 mineralised layers, three potential economical layers were identified: TZ, T1, and T0. 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 is 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.

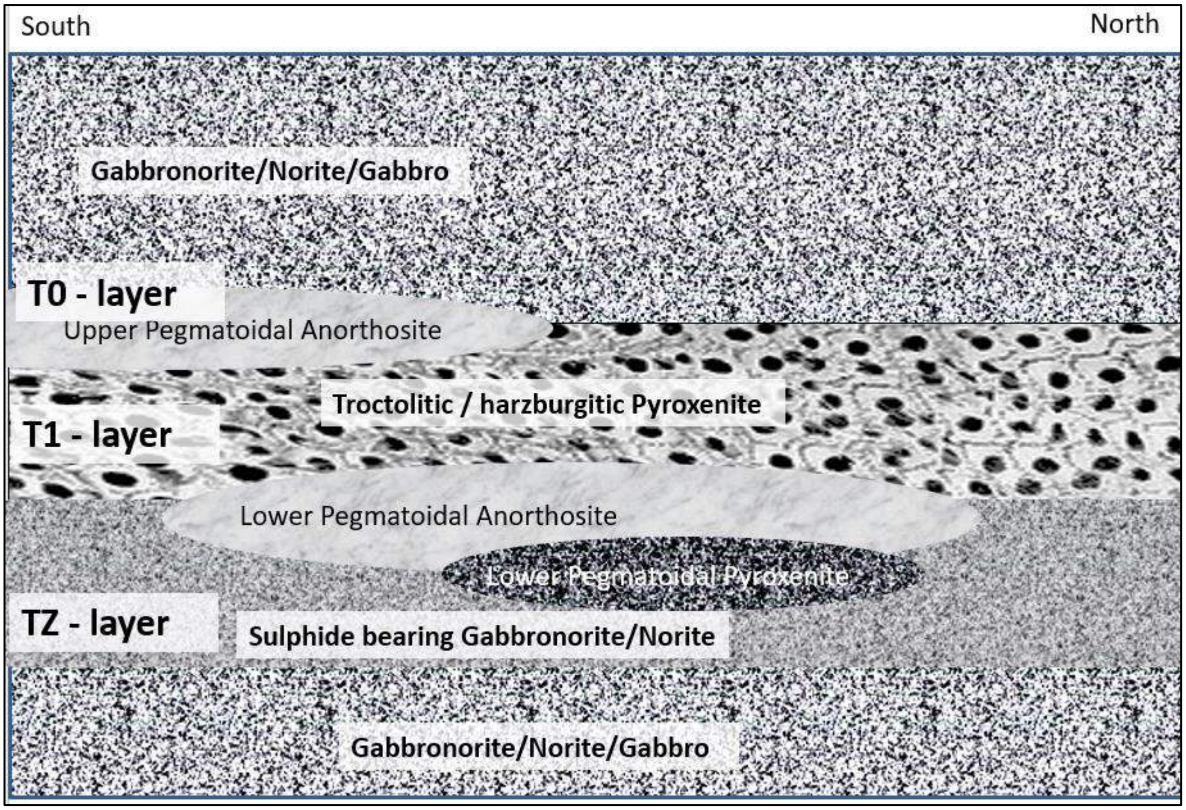
The mineralisation generally comprises sulphide blebs, net-textured to interstitial sulphides and disseminated sulphides within gabbronorite and norite, pyroxenite, and harzburgite.

Within the F Zone, basement topography may have played a role in the formation of higher grade and thicknesses where embayments or large-scale changes in magma flow direction may have facilitated the accumulation of magmatic sulphides. These areas are referred to as the “Super F”

Zones where the sulphide mineralisation is over 40 m in thickness and within the defined areas average 3 g/t to 4 g/t 4E. Layered magmatic sulphide mineralisation 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 several lithologically different and separate layers (Figure 8-1), which were initially recognised in the drilling. With subsequent drilling, it has become clear that the most easily identifiable and consistent are the TZ, T1, and T0 Layers.

Figure 8-1: Geological Interpretation of the T Zone



8.2 Description of T Zone Layering and Mineralisation

The T Zone is a unit that can be correlated and includes five identifiable layers. The three mineralised and economical potential layers are the TZ Layer, the T1 Layer, and the T0 Layer. Figure 8-1 is a geological interpretation of the T Zone layers.

8.2.1 Upper Pegmatoidal Anorthosite

The Upper Pegmatoidal Anorthosite (UPA) has a pegmatoidal texture and is mostly anorthositic with some gabbros. This unit is generally not mineralised; however, it was found to have some sulphide mineralisation towards the top of this zone that represents the T0 mineralised unit. The mineralisation is hosted within the mafic crystals of pegmatoidal texture.

The UPA has a thickness range from 2 m to as thick as 100 m and can be correlated in more than 80% of the drill holes. It must be noted that the unit is absent in some drill holes and it also appears more mafic in some instances due to alteration of the anorthositic and gabbroic phases.

8.2.2 T1 Layer Mineralisation

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 1-7 g/t with a Pt:Pd ratio of about 1:1.7. The Cu and Ni grades are on average 0.08% and 0.05%, respectively.

The unit is mineralised with blebby to net-textured Cu-Ni sulphides (chalcopyrite / pyrite and pentlandite) with very minimal Fe sulphides (pyrrhotite). The thickness of the layer varies from 2 m to 6 m.

8.2.3 Lower Pegmatoidal Anorthosite and Lower Pegmatoidal Pyroxenite

The direct footwall unit of the T1 Layer can be divided into two identifiable units: Lower Pegmatoidal Anorthosite (LPA) and Lower Pegmatoidal Pyroxenite (LPP). These units have an unconformable relationship with one another as both are not always present.

LPA is the first middling unit underlying the T1 Layer. It has the same composition as the UPA but is usually thinner. The LPA thickness ranges from 0-3 m and in some drill holes it is not developed. The LPA is mineralised in some drill holes.

LPP is the second middling unit that underlies the LPA and it is predominantly composed of pegmatoidal pyroxenite. It also ranges from 0-3 m as it is not developed in other drill holes. The LPP is a TZ Layer hanging wall. Mineralisation was not identified in this unit.

8.2.4 TZ Layer Mineralisation

Mineralisation within the TZ Layer is hosted in Main Zone norite and gabbronorite that shows a distinctive elongated texture of milky feldspars. In some instances, the TZ gabbronorite / norite tends to grade into pyroxenite and in places into a pegmatoidal feldspathic pyroxenitic phases, with the same style of mineralisation as in the gabbronorite / norite. The high-grade zones range from 2 m to approximately 10 m in true thickness within these lithologies. Sulphide mineralisation in TZ Layer is net textured to disseminated with higher concentration of sulphides compared to the overlying T1 Layer. The 4E grade (g/t) is typically 1-6 g/t with a Pt:Pd ratio of about 1:1.7. The Cu and Ni grades are typically 0.17% and 0.09%, respectively.

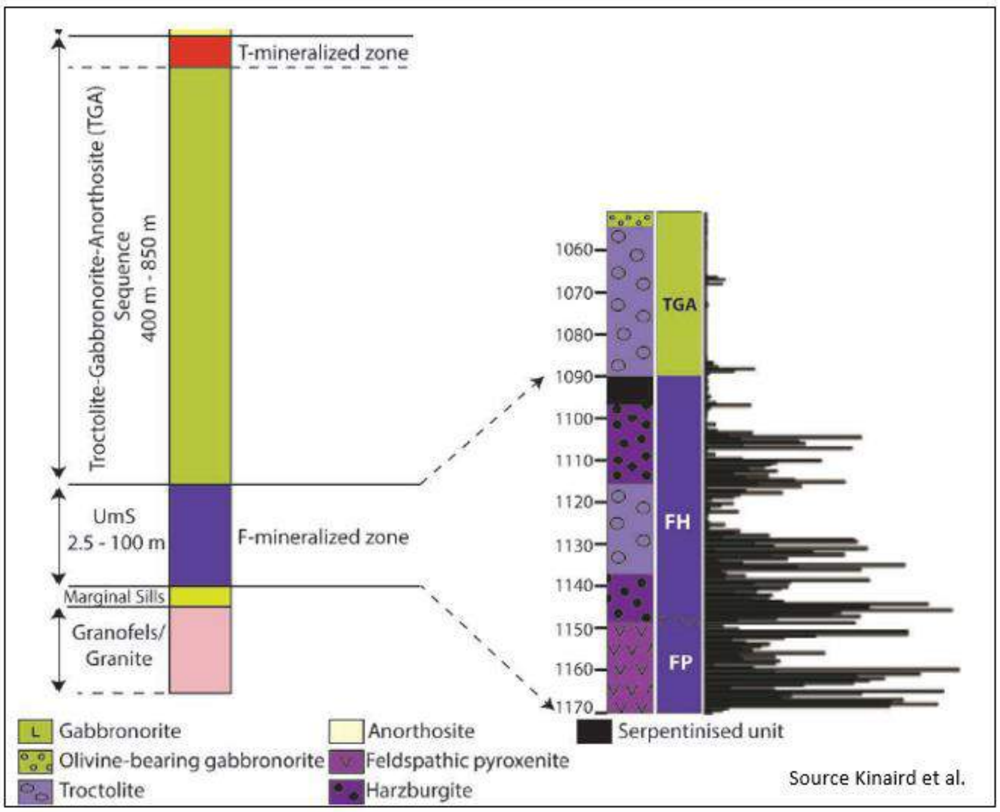
The Mineral Resource Estimate used the data to define the characteristics of the various layers based on their geological characteristics and geochemical signatures as shown in Figure 8-1.

8.3 Description of F Zone Layering and Mineralisation

A thick package of norite and gabbronorite ranging from 100 m to about 450 m underlies the T Zone and overlies the F Zone.

F Zone mineralisation is hosted in a thick package of troctolite, which usually occurs as thin layers of pyroxenite and/or pegmatoidal pyroxenite and harzburgite as shown in Figure 8-2. 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).

Figure 8-2: F Zone Mineralisation



F mineralised zone occurs in the ultramafic sequence pyroxenite and harzburgite. In the southern portion, the F zone is typically <10 m thick but in the central portion, the "Super F Zone" thickens to 60 m in true thickness, with grades of 2 to 4 g/t 4E over this interval. The mineralisation generally comprises blebs, net-textured to disseminated pyrrhotite, chalcopyrite and pentlandite with accessory chromite, 70 chalcocite, and pyrite. Chromite crystals are often enclosed in silicates, while chromite itself may host sulphide inclusions and rare chromitite stringers were identified in two drill holes. Magnetite has often replaced sulphides and chromite. PGM are variable with dominant sperrylite and subordinate Pt-Pd bismuthotellurides, Au-Ag alloys, Pd arsenides, and Pt-Rh sulpharsenides. More textural details will be described in a subsequent paper.

9 EXPLORATION DATA / INFORMATION

The Waterberg Project is at an advanced exploration status and includes an inferred, indicated, and measured Mineral Resource Estimates. Exploration further north has investigated the interpreted strike extension of the Bushveld Complex. As a result, 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 magnetics. Subsequently, exploration was driven by drilling and was undertaken by Waterberg JV Resources.

Engineering, including metallurgy, rock mechanics, mine and infrastructure design work is ongoing as part of the current DFS study.

A total expenditure of US$61 400 622 was spent on the Waterberg Project by the end of 2018. It is estimated that an additional US$5 000 000 will be spent to finalise the DFS in 2019.

Suitable exploration was undertaken with appropriate conclusions and follow-up work completed.

9.1 Remote Sensing Data and Interpretations

There is no remote sensing data relevant to this report. Extensive geophysics, including airborne data is discussed in Section 9.2.

9.2 Geophysics

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 area, with some overlap over the defined Bushveld edge geology on the advanced stage Waterberg JV Project.

9.2.1 Initial Survey

Approximately 60 lines of gravity and magnetic geophysical survey covering 488 km were traversed in March 2010. These were east-west trending lines traversed on the farms Disseldorp 369LR, Kirstenspruit 351LR, Bayswater 370LR, Niet Mogelyk 371LR and Carlsruhe 390LR. In March 2010, the PR for the farm Ketting 368LR was still pending. When this was granted, a second phase of geophysical survey was conducted on this farm from mid-August 2011 to September 2011.

Two supplementary north-south ground magnetics lines were surveyed over the farm Ketting 368LR in November 2012. This information was used to interpret and locate east-west striking structures.

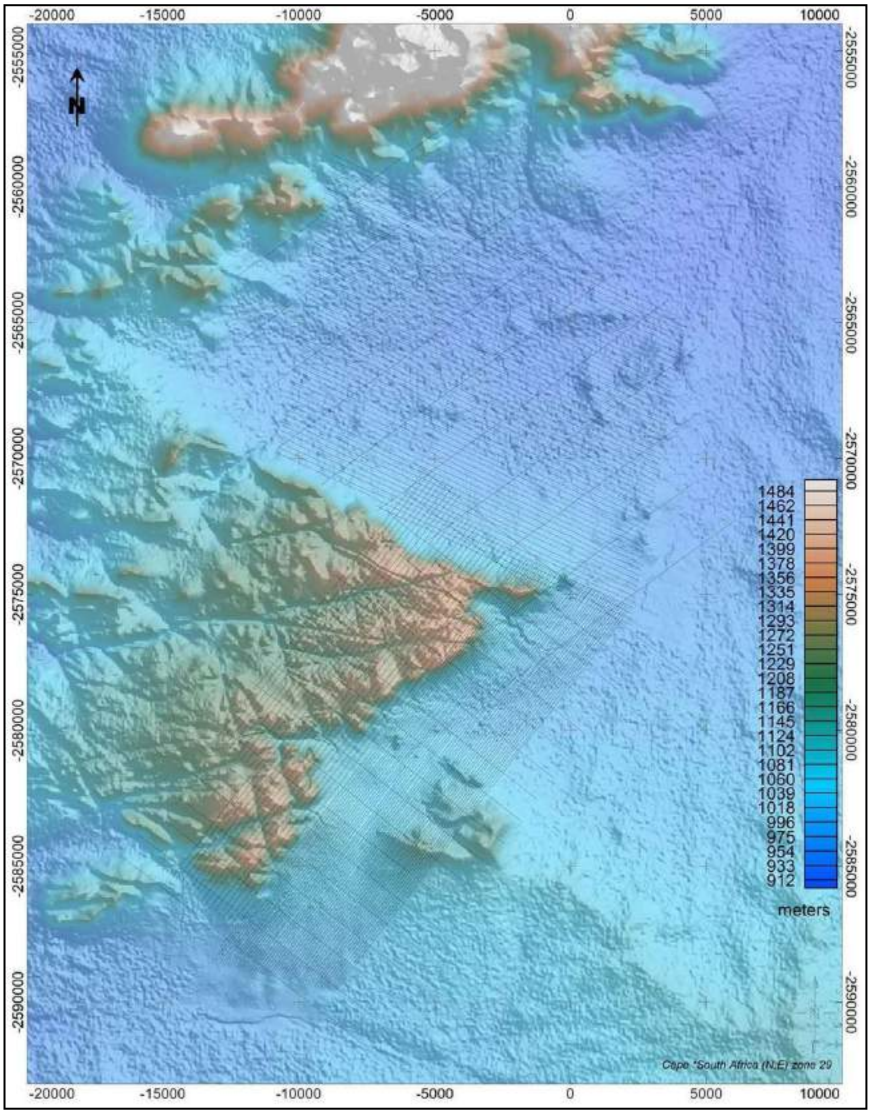
On the Waterberg Extension area, due to the presence of Waterberg Group cover rocks, there was no exposure of Bushveld Complex rocks. Geophysical techniques were used to assist in the modelling of the projected Bushveld Complex. A comparison of the regional geophysics modelling, the Falcon® airborne survey interpretation and the ground gravity profiles demonstrated general correlation, with local variations, of a north-northeast arch where the edge of the denser Bushveld Complex mafic intrusive rock may project beneath the Waterberg Group sediment cover.

9.2.2 Extended Airborne Gravity Gradient and Magnetics

An airborne gravity survey was completed on 100 and 200 m line spacing. An interpretation of the results of the survey suggests that there may be continuity to the Bushveld Complex rocks to the northwest and north, which has the potential to host PGM mineralisation to the northeast within the Waterberg Project area.

PTM RSA 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 subcropping of the Bushveld Complex to which the Waterberg sediments form the regional hanging wall. Conducted in April 2013, the survey was comprised of 2 306.16-line km of airborne gravity gradiometry data and 2 469.35-line km of magnetic and radiometric data. The total extent of the survey covered approximately 25 km of interpreted Bushveld Complex edge in the north-eastern part of the project area as shown in Figure 9-1.

Figure 9-1: Airborne Gradient Gravity and Magnetic Survey Flight Lines



Interpretation was based on creating a starting model using the known geology from drilling and linking it to the airborne response as shown in Figure 9-2 and Figure 9-3. The geological units were modelled in 3D to facilitate a 3D stochastic inversion of the geometry and density of the units making use of the gravity gradient data.

Figure 9-2: Waterberg Project Airborne Gradient Gravity Plot with Interpreted Bushveld Complex Edge

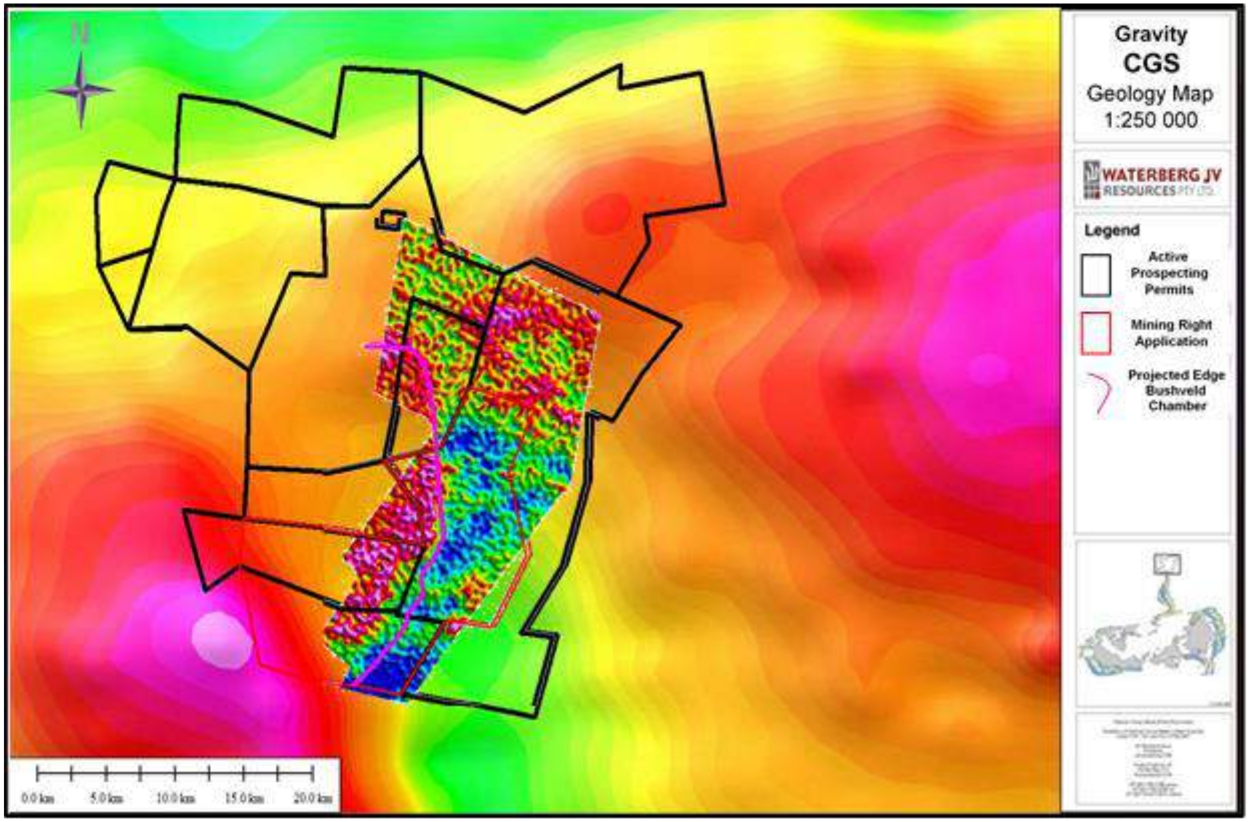
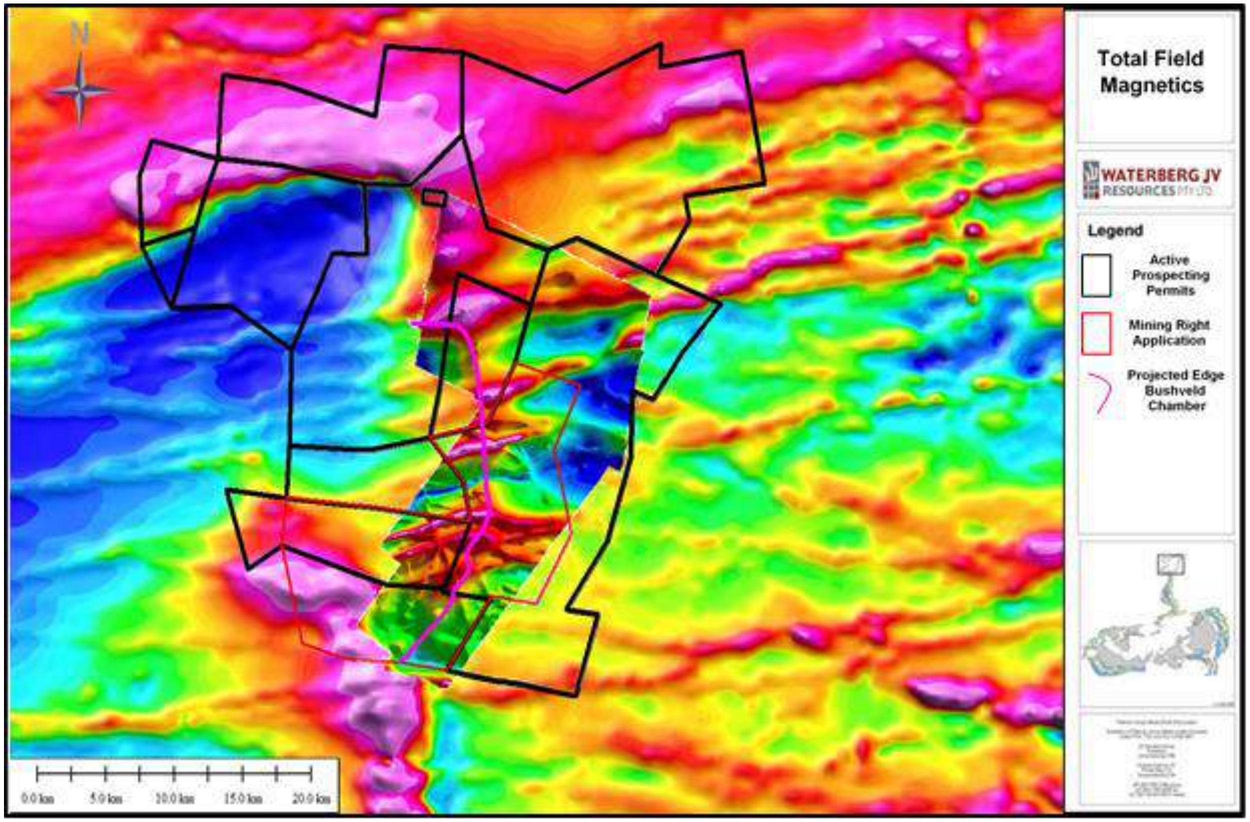


Figure 9-3: Airborne Total Field Magnetics Plot with Interpreted Bushveld Complex Edge



9.2.3 Ground Gravity

A total of nine ground gravity traverses were completed by Geospec Instruments (Pty) Ltd along roads and tracks. The survey lines were designed to traverses 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 good correlation between gravity data sets 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 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 mineralisation.

9.2.4 High-resolution Aeromagnetic and Radiometric Survey

A high-resolution, aeromagnetic and radiometric survey was conducted by Xcalibur Airborne Geophysics in November 2017.

9.2.4.1 General Survey Information

The project blocks consisted of approximately 1 595 line-km. The survey commenced on 28 November 2017 and was completed on 30 November 2017. Data collected was magnetic, radiometric and digital terrain model.

Figure 9-4 through Figure 9-6 show the location and design of the survey blocks.

Figure 9-4: Survey Area Location

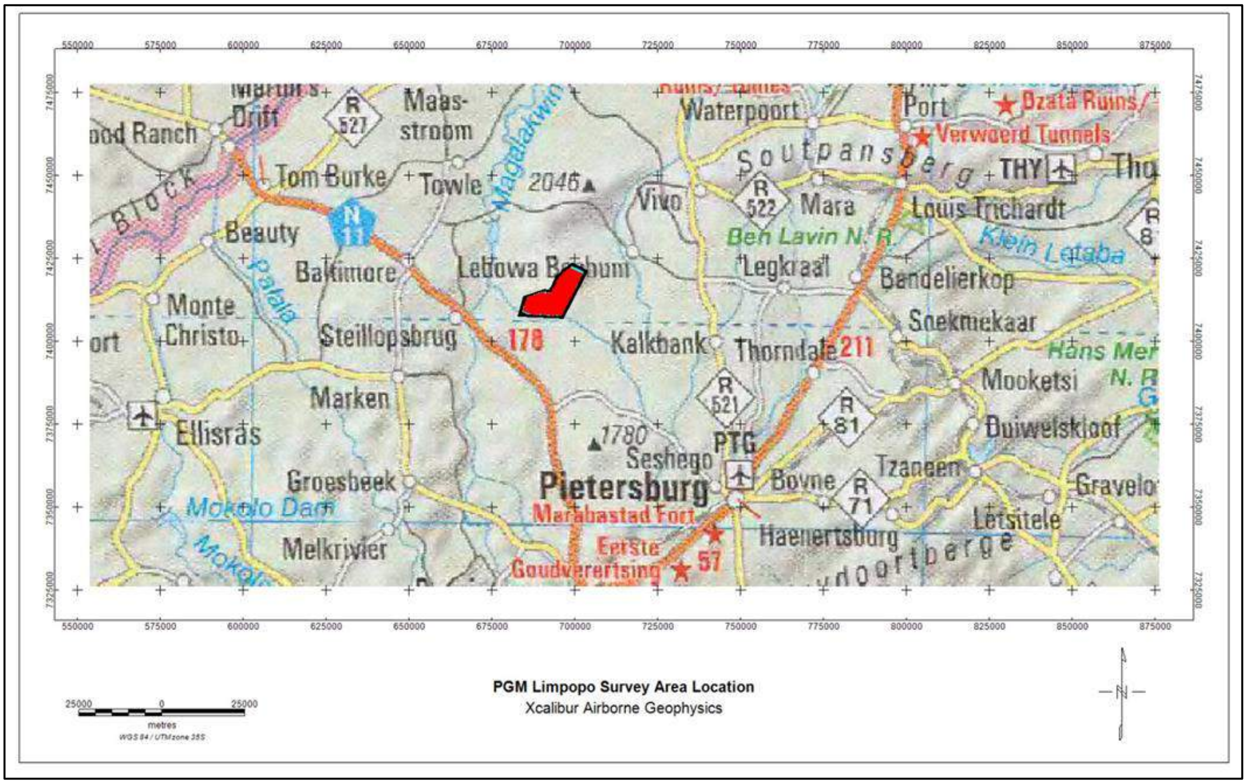


Figure 9-5: Survey Area SRTM Image

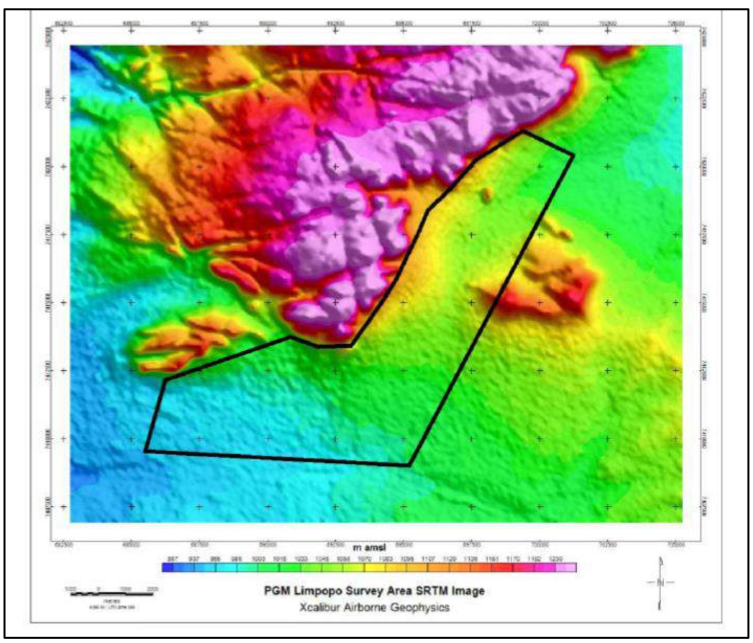
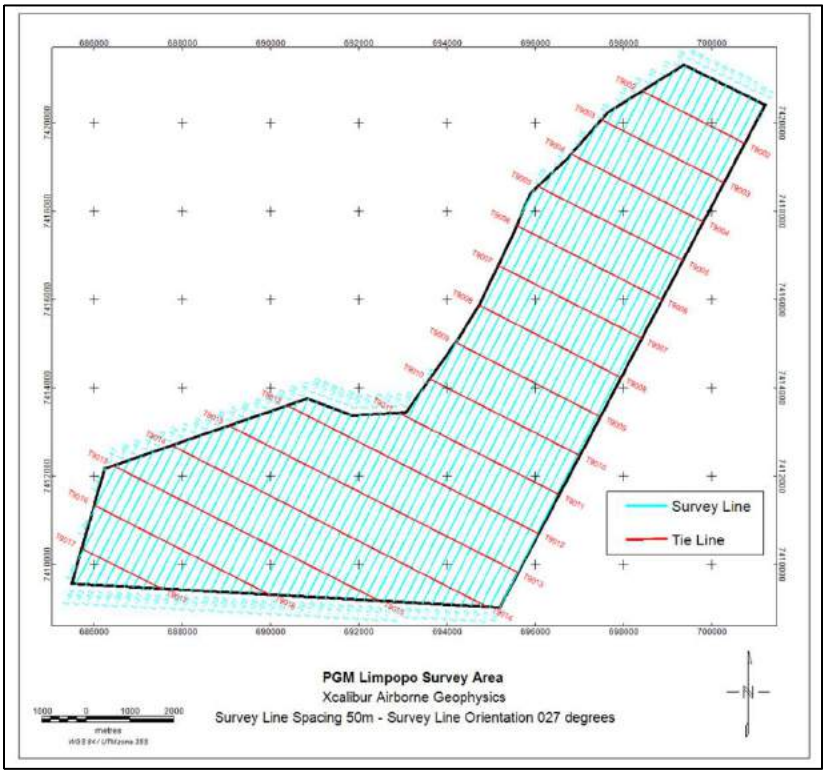


Figure 9-6: Survey Area Line Spacing 50 m and Line Orientation 027 Degrees



9.2.4.2 Basic Survey Parameters

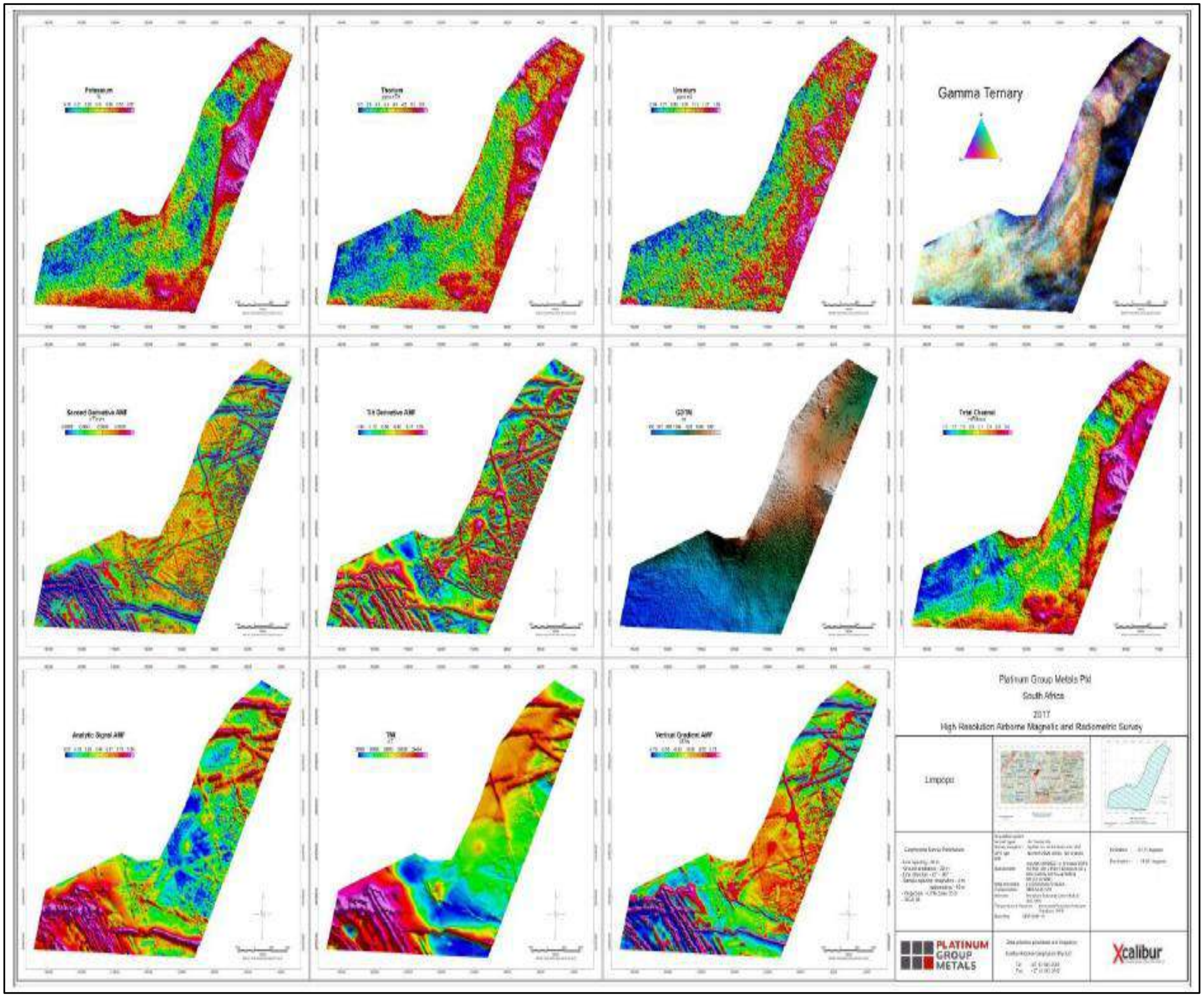
All data were recorded, processed and delivered in the UTM35 south projection system using the UTM WGS 84 datum.

- Line Direction: 27-207° with Respect to UTM 35S Zone Coordinate System
- Tie Line Direction 117-297° with Respect to UTM 35S Zone Coordinate System
- Ground Clearance: 35 m (Hazard Dependent)
- Line Spacing: 50 m
- Tie Line Spacing: 500 m
- Sample Spacing: Magnetic: 4 m, Radiometric 40 m

9.2.4.3 Basic Data

The high-resolution data is shown in Figure 9-7.

Figure 9-7: High-resolution Airborne Magnetic and Radiometric Survey Data



9.3 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 included 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 rock and quaternary sand deposits.

Data for any outcrop observed (or control point) was recorded in the field book: point's name, description of the outcrop's rock, identified rock name, XYZ 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 Waterberg 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.

9.4 Structural Studies

Pertinent structural geology is discussed in detail in Section 7.

10 DRILLING

Drilling was done by a specialised contractor, Discovery Drilling (Pty) Ltd mobilised out of Marken, South Africa. All drilling was undertaken by diamond drill coring and are near-vertical at their collars. Generally, drill holes were drilled using NQ core (47.6 mm), occasionally necking down to BQ if poor ground conditions were encountered or deep drilling was required. Metallurgical holes were drilled using NQ sized core. Table 10-1 summarises the drilling by year.

Table 10-1: Drilling by Year

Year	Number of Holes	Deflections	Total Metres	Cumulative Metres
2010	2	2	1 935	1 935
2011	1	3	1 774	3 709
2012	38	98	49 067	52 776
2013	86	132	86 403	139 179
2014	103	139	108 021	247 200
2015	47	64	35 322	282 522
2016	45	65	25 189	307 711
2017	53	43	22 375	330 086
2018	66	37	32 207	362 293
Total	**441**	**583**	**362 293**	**362 293**

The average drill hole length is 617 m, the minimum drill hole length is 200 m (WB218), and the maximum drill hole length is 1 643 m (WB004).

10.1 2010 Drilling

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 1 935 m was drilled for the first two drill holes in 2010. These holes intersected the “T” layers of mineralisation.

10.2 2011 Drilling

Drilling resumed in 2011 with a third drill hole WB003 drilled on the farm Ketting 368LR. This hole intersected both T and F Zone mineralisation.

10.3 2012 Drilling

The 2010 and 2011 drill holes led to the 2012 drill campaign which delineated a portion of the Waterberg mineralisation. In 2012 49 067 m from 38 holes with 98 deflections were completed. This work delineated the southern portion of the Waterberg Deposit.

10.4 2013 Drilling

A total of 86 403 m of core was drilled during 2013 from 86 holes and 132 deflections. A basic 250 m x 250 m grid drilled grid was used to position the drill holes where possible.

Drilling in some areas proved to be difficult due to bad ground formations, particularly in the Waterberg sediments. Consequently, some drill holes had to be re-drilled a few metres away, totally abandoned, or moved.

Diamond drilling commenced towards the north-east in October 2013 upon the official granting of the PR for the Waterberg Extension Area. 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 2 km. Six diamond drill machines were mobilised. Eight of the nine initial drill holes intersected Bushveld Complex stratigraphy.

10.5 2014 Drilling

A total of 103 drill holes and 139 deflections were completed during 2014, resulting in 108 021 m of core. The majority of drill holes were infill drilling of the 250 x 250 m grid aimed at upgrading portion of the inferred Mineral Resource to an indicated Mineral Resource.

10.6 2015 Drilling

The initial database for the July 2015 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.

A total of 35 322 m was drilled from 47 drill holes and 64 deflections during 2015.

10.7 2016 Drilling

Another 45 drill holes and 65 deflections were drilled during 2016 with a total of 25 189 m of core, mainly to increase the indicated Mineral Resource.

10.8 2017 / 2018 Drilling

Infill drilling continued during the 2017 / 2018 period to improve geological understanding and confidence in the Mineral Resource Estimates. A total of 119 drill holes and 80 deflections were completed during this period with a total of 54 582m of drill core. The raw database consisted of 441 drill holes with 583 deflections totaling 362 293 m.

10.9 Collar Surveys

A contracted certified land surveyor used a differential Trimble global positioning system to conduct collar surveys on all completed holes. Stations were tied in with survey stations established by the National Survey General Directorate. Drill hole coordinates were given in the Hartebeesthoek 1994 LO29 national coordinate system.

10.10 Downhole Surveys

Downhole surveys are done on 1 m intervals using a gyroscopic tool with some older holes using an electronic multi-shot survey tool. Deflections were done using a gyroscopic survey tool. There are five mineralised, vertically drilled original holes that were not surveyed due to bad ground conditions (WB108 – 427.60 m, WB110 – 1 276.47 m, WE006 – 498.23 m, WE016 – 883.80 m and WE025 – 736.28 m).

10.11 Drilling Quality

CJM examined core from randomly selected drill holes. The core recovery and core quality met or exceeded industry standards. The quality of the work in the drilling programme s was excellent.

Following is the drilling process. Drilled core is cleaned, de-greased, and packed into metal core boxes by the drilling company. The core is collected from the drilling site daily by Waterberg JV Resources 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 signed off by Waterberg JV Resources. 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
- Markings on Core for Sample Referencing at 1 m-interval
- Re-checking of Depth Markings for Accuracy

Each core box was photographed using a digital camera from fixed vertical distance. The photographs were stored on a network server.

10.12 Geological Logging

Standardised geological core logging conventions were used to capture information from the drill core. Detailed geological logging was completed daily by qualified geologists onto a proforma capture sheet under supervision of the Waterberg 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, colour, and optional comments); stratigraphic units; type and degree of alteration (infill, partial, or pervasive); and mineralisation (type, style, and visible percentage of sulphides).

Three magnetic susceptibility readings were taken and averaged together from the beginning of the Bushveld Complex lithologies to the end of hole at 1 m intervals.

Once the geological logging was captured into the Sable database on site, the logs were printed, and a qualified geologist checked the core against the captured logs to verify that the data was recorded and captured correctly. The printed logs were then signed off and stored in the drill hole file.

All data was captured in the field directly in the Sable database located at Waterberg JV Resource offices, Johannesburg via the company network.

All documentation relating to each drill hole, including geological logs, survey certificates, collar certificates, sampling sheets, assay certificates, etc. were collated and filed in a file for each drill hole at the field camp. All documentation was scanned and sent electronically to the Waterberg JV Resources office in Johannesburg and saved on the server along with all available digital photographs.

10.13 Diamond Core Sampling

Sample selection was undertaken by qualified geologists based on a minimum sample length of approximately 25 cm with an average length of 50 cm. Not all drill hole core was sampled, but all core with visually identifiable sulphide mineralisation was analysed and low-grade to waste portions straddling these layers were also sampled. A maximum sample length of 1.5 m was applied where appropriate. The true width of the shallow dipping (30° to 35°) mineralised zones that were sampled are approximately 82% to 87% of the reported interval from the vertical drill hole.

The sampled core was split using an electric powered circular diamond blade saw. Samples were cut according to the sampling sheet created by the geologist logging the hole.

10.14 Core Recovery

Core recoveries, rock quality designation (RQD), and a note of core quality were recorded continuously for each drill hole and for each drill run. The core recovery within the first few metres of drill holes (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, core recovery was commonly 100% once drilling reached the Main Zone hanging wall, reef horizons, and footwall rocks. The recoveries only show a substantial decrease within faulted / sheared zones.

10.15 Sample Quality

CJM examined selected drill holes and assessed that the quality of sampling met or exceeded industry standards.

10.16 Interpretation of Results

The results of the drilling and the general geological interpretation were digitally captured in Sable and a geographic information system (GIS) software package named ARCVIEW. The drill hole locations together with the geology and assay results were plotted on plan. Regularly spaced sections were 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.

In CJM's opinion, the quantity and quality of the lithological, geotechnical, collar, and downhole survey data collected in the exploration and infill drill programme s are enough to support Mineral Resource estimation as shown below.

- Core logging meets industry standards for PGE–Au–Ni–Cu exploration.
- Collar surveys and downhole surveys were performed using industry-standard instrumentation.
- Recovery from core drill programme s is acceptable to allow reliable sampling to support Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Sampling

11.1.1 Sampling Method and Approach

Waterberg Project staff members were responsible for the following activities.

- Sample Collection
- Core Splitting
- Sample Dispatch to the Analytical Laboratory
- Sample Storage
- Sample Security

Once geological logging is complete and validated, the qualified geologist identifies the units to be sampled based on stratigraphic, lithological and visible sulphide mineralisation criteria. Continuous sampling from the top of the mineralised zone to well below footwall contacts is undertaken. The geologist varies the thickness of sampling intervals according to changes in stratigraphy, lithology, and mineralisation to ensure that samples do not crosscut 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 are inserted. A blank is inserted for one in every ten samples. A standard is also inserted for one in every ten samples. The result is a quality control sample after every five primary samples.

Before any sampling takes place, the core is orientated and secured with tape where it is broken. 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. It is the sampler’s responsibility to ensure that representative samples are taken (i.e. one side of the core is sampled for all samples). It is also the sampler’s responsibility to ensure the correct ticket is allocated to the correct sample on the sample sheet and that the sample plastic bags are properly labelled.

Each sampler is assigned an assistant whose responsibility it is to remove the tape from the samples, squeeze the air out of the sample bags, wrap the sample bags properly, weigh the samples (with weight of the sample bags normalised on the scale), and staple the sample bags.

The section of core to be sampled is placed in a plastic bag with a sample ticket from the ticket book.

For inserted certified reference material (CRM) standards, 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. All the CRM labels are filed in the field camp 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 sealed and the sample number written on the bag. The sample in the bag is weighed and the weight in grams is 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.

11.1.2 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 SG is calculated and matched to the assay results for that sample for modelling purposes.

Following is the formula for SG.

SG = Mass in Air (Ma) / [Ma-Mass in Water (Mw)]

33 754 samples were measured for bulk density. These densities are representative of the stratigraphic and lithological units used within the geological model.

11.1.3 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 Waterberg JV Resources contractors.

When samples are prepared for shipment to the analytical facility, the steps listed below 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.
- The sample shipping bags are sealed and 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.

11.1.4 Security

Samples are not removed from secured storage location without completion of a chain-of-custody document, which 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 Waterberg Project site and analytical facility, the samples are inspected and signed for by each person 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 destination.

The supervising geologist ensures that the analytical facility is aware of the Waterberg JV Resources 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 Waterberg JV Resources. A photocopy of the chain-of-custody document, signed and dated by an official of the analytical facility, is faxed to Waterberg JV Resources offices in Johannesburg upon receipt of the samples by the analytical facility and the original signed letter is returned to Waterberg JV Resources 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 Waterberg JV Resources 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 that the sample shipment may have been tampered with or otherwise compromised, the responsible geologist will immediately notify the Waterberg JV Resources 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, Waterberg JV Resources 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 Waterberg JV Resources, 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.

11.1.5 Sample Preparation and Analysis

The laboratories that were used to date are Set Point Laboratories (South Africa), Bureau Veritas Testing and Inspections South Africa (Pty) Ltd (Bureau Vertias) as the primary laboratories and Genalysis Laboratory Services Pty Ltd (Genalysis) (Perth, Western Australia) for the referee samples.

Bureau Veritas (Rustenburg, 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 programme, 2 045 primary samples from previous drilling programme s 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, following demonstration of its technical competence, to operate in accordance with International Standards Organization (ISO) / International Electrotechnical Commission (IEC) 17025, which includes the 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.

The laboratory inserts their own certified reference materials to measure accuracy (sample type code LABSTD in the Sable database) where accuracy refers to the closeness of a measured value to a standard or known value. The laboratory also inserts blanks to check for contamination (sample type code LABBLK).

Random primary samples are split to create preparation duplicates (coarse rejects with a sample type code of LABCRD) and to create pulp duplicates (with a sample type code of LABDUP) with a ratio of one to every 20 primary samples of each. These are then inserted into the sample stream. Results are compared to the corresponding primary samples to test the precision of the laboratory measurements where precision refers to the closeness of two or more measurements to each other.

Samples are analysed for Pt (g/t), Pd (g/t), and Au (g/t) by standard 25 g Pb fire-assay using Ag as requested by a co-collector to facilitate easier handling of prills as well as to minimise losses during the cupellation process. The resulting prills are dissolved with aqua-regia for inductively coupled plasma (ICP) analysis.

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

The base metals (Cu, Ni, Co, Cr, and S) are analysed using ICP/OES after a multi-acid digestion. This technique results in "almost" total digestion.

Samples submitted for Rh analysis are assayed by fire assay using Pd collection followed by ICP/OES.

All pulp rejects and coarse rejects are returned to the field camp for storage.

The assay results are reported to the Waterberg JV Resources database manager as Excel spreadsheets via email. The Excel spreadsheets are imported directly into the Sable database using customised import routines. There is no editing or manipulation of the Excel spreadsheet before import. Once imported, QA/QC checks are done using Sable software and in Excel.

11.1.6 Sampling Audit Process

The first stage of the audit process starts at the drill rigs. At this stage, the quality of the core recovered (recoveries & RQDs) is checked. Other key attributes perused include packing the drill core into core trays, labeling the respective core trays, and core handling during shipment from drill sites to exploration camp.

The second stage of the auditing process is performed at the exploration camp where the drill core is logged, sampled, and shipped to the laboratory. The process starts with observing how core trays are laid out in preparation for logging and sampling. The entire sampling workflow listed below is observed.

- Generation of Sample Logs
- Orientation of Drill Core in Preparation for Splitting
- The Splitting Process
- Bagging the Samples into Plastic Bags

- Labeling the Respective Plastic Bags and Insertion of Standards and Blanks
- Ticketing of Individual Samples
- Recording of Individual Sample Weights
- Bagging Samples into Batches
- Order Number Requisitions
- Preparing Relevant Paperwork to Accompany the Samples
- The Sample Dispatch

The third stage of the audit is at the laboratory. The laboratory tour begins at the sample receiving area and continues in a logical sequence to the end of the analytical process. Questions regarding quality control procedures, internal pass / fail frequencies, and database related questions are posed to the laboratory manager.

The fourth and last stage of the audit process involves auditing the company database and scrutinising how assay results are reported and imported into the database. The process of how batch failures are communicated with the laboratory is intensely scrutinised at this stage.

Once an audit is complete, an audit report with recommendations is compiled and forwarded to the Technical Manager, Project Manager, and Database Manager for remedial actions.

Since the inception of the Waterberg Project, two audits were conducted by Barry Smee (Smee Associates) and one audit by the senior exploration team (Maja Herod, Aleck Mkhabela, and Edwin Matiwane). Ad hoc laboratory inspections were also conducted by the Project Manager (Aleck Mkhabela).

The first audit was conducted by Barry Smee from the 12 to 19 July 2013. Most of the issues accompanied by remedial actions were identified during this audit and outlined in a report titled, “Results of an Audit of the Setpoint Preparation Laboratory and Full Reviews of Quality Control Data and Field Methods, Waterberg Project, Republic of South Africa.”

The following risks were identified.

- The Laboratory Information Management System (LIMS) caused concerns with the assay database as no fixed format was imported. The recommendation was that the laboratory fix a work order number for all their laboratories. The work order number was to consist of an alpha laboratory location (i.e., MOK for Mokopane), a number with the year, and a five-digit number for the actual job number (i.e., MOK1300345). This system has made it easier to work with the database.
- Plastic bags were used to package milled samples. The recommendation was to replace the plastic bags with Kraft paper wire-top sample bags.

- Rejects were not obtained. The entire sample was pulverised. As the primary samples were big enough to yield rejects, the recommendation was to obtain rejects for each sample and store them in a plastic bag (labeled accordingly).
- Only B2000 pulveriser bowls were used by the laboratory. These bowls were not highly effective on smaller samples. The recommendation was to obtain B1000 and B500 bowls and use the appropriate bowl to fit the sample weight.

Recommendations from the audit reports were communicated with the laboratory and the exploration team with a mandate to execute.

In November 2014, the Waterberg senior exploration team conducted an audit at both the laboratory and exploration site. The objectives of the audit were to check if both the laboratory and exploration site adhered to industry standards. It was also to confirm that recommendations from the initial audit by Barry Smee were implemented. Upon completion of the audit, an audit report with recommendations was compiled titled, "An Audit of Waterberg Field Sampling Collection Methods and the Setpoint Laboratory."

From 01-03 July 2015, Barry Smee visited the Waterberg Project for a follow-up audit. The general sentiment was that there were significant improvements compared to the previous audit.

11.1.7 Geochemical Soil Sampling

In March 2010, two north-south sampling lines were completed. Sampling stations were made at intervals of 25 m. Each sample hole extended to a minimum depth of 50 cm to 1 m, at most.

During December 2011 and January 2012, two additional north-south lines on the property Niet Mogelyk 371LR were also sampled. 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 50 m 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 discover by the previous sampling programme.

A total of 723 samples were collected during this process; 367 were soil samples, 277 stream sediment samples, and 79 rock chip samples. Geochemical sampling of the soils was also partially compromised due to very thin overburden because of subcropping rock formations. Geochemical sampling showed elevated PGMs and this increased exploration interest in the area in 2011.

11.2 Database Management

Databases in use at Waterberg JV Resources currently include Sable, which is an SQL-based relational database. This is a centrally managed database containing all aspects of drilling information including logging and assay results. In addition, Waterberg JV Resources uses ARCVIEW, a GIS database system that is also SQL based for all spatial information relating to exploration activities. Several 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.

11.3 Quality Assurance and Quality Control Analysis

11.3.1 Quality Assurance and Quality Control Procedure

Waterberg JV Resources has a well-established and functional QA/QC procedure.

Quality monitoring needs to be assessed on two basic factors – assessing the accuracy (how close results are to actual figures) and gauging the precision (the repeatability of the results). The various aspects involved in this process can be divided into quality assessment measures, and QA/QC.

The QA/QC of assays is defined as the combination of QA, the process or set of processes used to measure and assure the quality of results, and quality control, which is the procedure for determining the validity of analytical procedures and specific sampling.

QA includes a broad plan for maintaining quality, which encompasses monitoring activities, proper documentation, training, and data analysis and management.

Once the analysis is complete, various quality assessments are done to measure the accuracy and overall precision of the results.

The tools used for these assessments are a combination of Microsoft Excel and SatQc (Sable software for producing auditable, statistical and graphical reports demonstrating that the data in the database has passed the required checks).

As the project progressed, the assessments changed. Visual checks were done with some rudimentary analysis in Excel before results were imported into the Sable Data 1 database. Once all data was migrated to the Sable Data Warehouse, the original premise was that Sable's SatQc module would be used to do the assessment. For a period of approximately one year, this module was totally unusable. SatQc attempted to prepare reports for the entire database all at one time and the module ran out of memory and froze.

In the interim, until SatQc was "fixed," Microsoft Excel was used to do all assessments. Scripts were written to do the evaluation and comparisons of the results required. Imported results already loaded into the database were extracted into Excel and evaluated. For the assessment of the entire database of assay results, Excel is still the preferred tool. Excel has the flexibility of customised graphs and annotations. Excel also allows data to be evaluated by someone who does not have Sable software. It can also be emailed and serves as a snapshot of the data status at the time the assessment was performed and dated.

Reported results are extracted to Excel by drill hole for all batches belonging to that drill hole. There are separate tabs in the Excel spreadsheet for all field results (primary samples, inserted standards and blanks), the inserted standards (results, certified mean, and + 3 SDs), and the inserted blanks (results with the maximum acceptable value of 10 x detection). There are also tabs to laboratory coarse reject duplicates and pulp duplicates where the results are compared, and a percentage difference calculated. The scripts evaluate the reported result with respect to upper and lower acceptable limits and returns a pass or fail as the QA/QC status per element. It is very easy to identify exceptions that need to be investigated further.

Any exceptions are recorded in an exception control sheet. In some cases, the field staff are asked to check which standard or blank was inserted. On some occasions, the sampling sheet had a record of one standard, but another standard was put in the plastic bag.

If the duplicates, inserted standards or blanks have perceived erroneous values, the samples to be investigated are highlighted in the original spreadsheet received from the laboratory. The 5 primary samples before in the sequence and the 5 primary samples after in the sequence are also highlighted to indicate that if needed, repeats will be carried out on all highlighted samples. This file is returned to the laboratory for investigation. The exceptions spreadsheet is updated with the outcomes of all investigated and flagged as resolved, results accepted or other comments.

Guidelines were defined by an expert in QA/QC (Barry Smee) as to what statistics and graphs should be compiled for evaluation purposes. This means that results have a batch-specific Excel spreadsheet containing all QC samples. This is archived in the database confirming that wherever possible and feasible, exceptions were resolved. Laboratory inserted standards and blanks are also represented in tabs and results flagged as passing or failing acceptable limits.

When SatQc became operational, it was possible to create PDF reports directly from the database to demonstrate that the results in the database pass all checks. These PDF reports are also archived in the database for each sample type.

Finally, checks of the entire dataset of QC samples are also done in Excel. These checks are done annually but can be done at any time. Graphs plotted include Z-score graphs for standards (both field and laboratory certified reference materials), plots for blanks and x-y plots for duplicates. Z-score graphs are very efficient for displaying all standards on the same graph for comparative purposes.

Waterberg JV Resources are the custodians of the QA/QC results. Over the history of the Waterberg Project, CJM reviewed the findings of QA/QC results for the purposes of establishing validity of the data for inclusion into the Mineral Resource Estimate, with focus on the results since the last Mineral Resource Statement. To this end, data from Set Point, Bureau Veritas, and Genalysis were examined.

11.3.2 Analytical Quality Assurance and Quality Control Data

Table 11-1 shows the laboratories and methods used throughout the history of the Waterberg Project.

Table 11-1: The Laboratories and Methods used throughout the History of the Waterberg Project

Laboratory	Method for PGEs	Method for Base Metals	Detection Limits for Elements	Units for Reporting
Set Point	Fire assay with Pb collection fire assay and ICP/OES analysis NiS collection fire assay for Rh	4 acid digestion with ICP/OES analysis	Au 0.01 g/t, Pt 0.01 g/t, Pd 0.01 g/t, Rh 0.02 g/t, Cu 10 ppm, Ni 10 ppm	g/t for Au, Pt, Pd and Rh ppm for Cu and Ni
Bureau Veritas	Fire assay with Pb collection fire assay and inductively coupled plasma / mass spectrometry (ICP/MS) analysis	4 acid digestion and ICP/MS analysis	Au 0.001 g/t, Pt 0.005 g/t, Pd 0.005 g/t, Cu 2 ppm, Ni 2 ppm	ppm for Au, Pt and Pd ppm or Cu and Ni
ALS	Fire assay with Pb collection fire assay and ICP/MS analysis	4 acid digestion and ICP/OES analysis	0.01 ppm for Pt, Pd and Au, 10 ppm for Cu and Ni	ppm for Au, Pt and Pd ppm or Cu and Ni
Genalysis	Pb collection fire assay and ICP/MS analysis NiS collection fire assay for Rh	4 acid digestion and ICP/OES analysis	Au 1 part per billion (ppb), Pt 1 ppb, Pd 1 ppb, Rh 1 ppb, Cu 20 ppm and Ni 20 ppm	Au=ppb, Pt=ppb, Pb=ppb, Rh=ppb Cu=ppm, Ni=ppm

The laboratories used have the following certifications.

- Set Point Laboratories, Part of Torre Industries, is an ISO 17025 accredited analytical chemistry lab.
- Bureau Veritas Testing and Inspections South Africa (Pty) Ltd (Rustenburg, South Africa) was certified when used for the Waterberg Project. The laboratory is now closed and no longer has a certificate on the SANAS web site.
- ALS is an ISO 17025 accredited analytical chemistry laboratory. SANAS Accreditation Number T0387.

- Set Point Laboratories (SANAS Accreditation Number T0223) is 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." Accreditation Number 3244.

The QA/QC results are within acceptable limits; therefore, the results for the primary samples are deemed to be reliable and can be used for Mineral Resource Estimates.

A selection of commercial certified reference materials was used by both the laboratories as well as inserted in the field by the samplers to assess the QA/QC process. These CRMs are documented in Table 11-2.

Table 11-2: List of Certified Reference Materials used by Laboratories and for Field Standards

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
AMIS0001	PGE Ore Reference material	0.765	0.07	1.04	0.08	0.12	0.024				
AMIS0002	PGE Ore Reference material	0.82	0.112	0.89	0.098	0.155	0.016	1 310	120	1 970	150
AMIS0005	UG2 Reef (Ore Grade) PGE Reference Material	3.38	0.33	2.23	0.18	0.02		59	8	1 081	333
AMIS0006	UG2 Reef (Feed Grade) PGE Reference Material	1.43	0.15	0.91	0.08	0.02		823	82	787	79
AMIS0007	Merensky Reef (Feed Grade) PGE Reference Material	2.48	0.28	1.5	0.2	0.13	0.02	1 312	150	2 072	208
AMIS0008	Merensky Reef (Ore Grade) PGE Reference Material	8.66	0.78	4.36	0.39	0.36	0.05	2 262	231	3 782	335
AMIS0010	UG2 Reef (High Feed Grade) PGE Reference Material	2.13	0.2	1.32	0.15	0.025		750	66	1 084	166
AMIS0013	Merensky Reef Low Feed Grade PGE Reference Material	10.85	0.86	4.9	0.41	0.52	0.06	2 187	284	4 040	460
AMIS0014	UG2 Reef (Feed Grade) PGE Reference Material	1.95	0.22	1.2	0.13	0.038		102	19.2	886	172
AMIS0027	UG2 Reef (Ore Grade) PGE Reference Material	2.39	0.36	1.59	0.24	0.05		125	14	1 078	222
AMIS0034	Merensky Feed Grade Pt Ore Reference Material	3.69	0.36	1.63	0.18	0.43	0.08	1 544	100	2 079	148
AMIS0044	African Minerals Standards for Au					2.9	0.19				
AMIS0053	Merensky Reef PGE Reference Material	2.41	0.3	1.18	0.14	0.22	0.03	812	52	1 652	156
AMIS0056	Platreef Low Grade Pt Ore Reference Material	0.81	0.1	0.88	0.08	0.16	0.02	1 401	183	2 009	176
AMIS0064	PGE Ore Reference Material	1.24	0.12	0.58	0.06	0.11	0.02	636	66	1 452	134

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
AMIS0067	Pt (PGM) Merensky Reef Ore Reference material	1.95	0.16	0.98	0.08	0.15	0.02	895	44	1 728	182
AMIS0074	Pt (PGM) ore UG2 Reef Western Limb Bushveld Complex South Africa	1.07	0.1	0.72	0.06	0.05	0.012	65	6.4	668	94
AMIS0075	UG2 Reef, Eastern Limb PGE Reference Material	1.14	0.14	1.49	0.12	0.07	0.016	234	26	1 051	124
AMIS0089	Pt (PGM) Reference Material - UG2 Reef - Western Limb - Bushveld Complex - South Africa	1.09	0.12	0.7	0.06	0.04	0.012	59	6	452	52
AMIS0099	Pt (PGM) Merensky Reef Ore Bushveld Complex South Africa	0.59	0.07	0.225	0.034	0.089	0.016	256	18	443	48
AMIS0110	Au and Uranium (U) Ore Witwatersrand - South Africa					2.3	0.18				
AMIS0118	Cu Oxide Ore Reference Material from Lonshi DRC							4 615	270		
AMIS0122	Pt - PGM UG2 Reef Eastern Limb Bushveld Complex	2.61	0.21	3.17	0.24	0.115	0.016	506	47.3	1 351	196
AMIS0124	Platreef Low Grade PGE Reference Material	0.84	0.07	0.87	0.06	0.16	0.02	1 324	106	1 917	136
AMIS0132	Pt PGM UG2 Tailings Eastern Limb Bushveld Complex SA	0.46	0.04	0.21	0.02	0.028		47.2	7.6	684	121
AMIS0140	Tantalum Standard used by Genalyis -										
AMIS0146	Internal Set Point Standard not certified	1.29	0.05	1.76	0.06	0.164	0.018	1150	83	1 841	139
AMIS0148	Pt (PGM) Platreef Ore Bushveld Complex	1.64	0.1	1.13	0.08	0.84	0.04	541	55	900	77
AMIS0149	Not certified?										

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
AMIS0151	Pt (PGM) Merensky Reef Ore Bushveld Complex South Africa	4.64	0.36	3.15	0.28	0.072	0.014	150	14	1 281	195
AMIS0160	Cu Co oxide ore Mukondo DRC							31 000	1 800		
AMIS0164	Pt (PGM) Platreef Concentrate Bushveld Complex - South Africa	23.86	1.72	26.75	1.5	2.97	0.16	25 500	1 700	35 550	1 670
AMIS0165	Pt (PGM) Platreef Concentrate Bushveld Complex - South Africa	16.9	1.36	19.1	1.36	1.66	0.14	17 710	1 030	28 160	1 780
AMIS0167	Au and U Ore Grade Witwatersrand reference material					7.29	0.38				
AMIS0171	Pt (PGM) Merensky Concentrate Bushveld Complex SA	58.28	3.62	36.86	2.7	4.7	0.28	16 220	1 030	24 680	1 530
AMIS0192	Pt (PGM), Merensky Ore Bushveld Complex, South Africa	7.93	0.4	4.04	0.18	1.68	0.12	1 562	112	2 776	258
AMIS0207	Pt (PGM) Reference Material UG2 Reef, Western Limb, Bushveld Complex, South Africa	2.28	0.22	1.26	0.08	0.085	0.012	85	9	1 059	125
AMIS0208	Au and U Ore - Witwatersrand - South Africa					1.38	0.1				
AMIS0209	Pt (PGM) - Merensky Bushveld Complex - South Africa	1.21	0.1	0.63	0.06	0.09	0.01	447	20	909	35
AMIS0210	Au and U Ore - Witwatersrand - South Africa					1.26	0.16				
AMIS0252	Pt (PGM) -UG2 Bushveld Complex - South Africa	2.89	0.28	1.53	0.14	0.042	0.012	104	17	1 212	232
AMIS0253	Pt (PGM) -UG2 Bushveld Complex - South Africa	4.03	0.32	2.34	0.18	0.07	0.01	134	23	1 220	168

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
AMIS0254	Pt (PGM), Merensky Bushveld Complex South Africa	2.19	0.16	1.12	0.08	0.2	0.02	762	49	1 735	177
AMIS0256	Pt (PGM), Merensky Ore Bushveld Complex South Africa	4.86	0.22	2.5	0.12	0.34	0.04	1 252	69	2 913	181
AMIS0257	Pt (PGM) UG2 Ore Bushveld Complex, South Africa	1.66	0.16	0.95	0.08	0.11	0.02	65	10	961	157
AMIS0278	Pt (PGM) Platreef Ore Bushveld Complex - South Africa	1.7	0.1	2.12	0.14	0.26	0.02	1 294	80	2 026	236
AMIS0282	Ni-Cu-PGM ore Sudbury basin Canada	0.97	0.1	1.41	0.12	0.19	0.01	1.68	0.12	4 971	560
AMIS0283	Ni-Cu-PGM ore Sudbury basin Canada	0.82	0.08	0.49	0.06	0.092	0.01	27 410	1 810	22 570	1 980
AMIS0302	Au and U Ore Witwatersrand - South Africa					4.47	0.34				
AMIS0325	Pt (PGM) Platreef Ore Bushveld Complex - South Africa	2.06	0.18	2.25	0.18	0.3	0.04	2426	178	4 091	283
AMIS0326	Pt (PGM) Platreef Ore Bushveld Complex - South Africa	1.05	0.08	1.25	0.08	0.17	0.02	1403	89	2 446	99
AMIS0328	Pt (PGM) - Merensky Bushveld Complex - South Africa	2.14	0.18	1.38	0.12	0.14	0.01	669	38	1 945	226
AMIS0337	Au Ore siliceous matrix Navaho Mine Namibia					0.66	0.06				
AMIS0354	Pt (PGM), Merensky Bushveld Complex, South Africa	2.25	0.25	1.34	0.08	0.71	0.05	582	31	1 839	226
AMIS0367	Pt (PGM) - Merensky Bushveld Complex - South Africa	1.8	0.24	0.84	0.08	0.17	0.02	826	41	1 766	66
AMIS0395	Pt (PGM) Platreef Ore - Bushveld Complex - South Africa	0.51	0.04	0.62	0.06	0.095	0.014	847	44	1 606	161

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
AMIS0396	Pt (PGM) Platreef Ore Bushveld Complex	0.75	0.06	0.93	0.06	0.105	0.016	969	54	1 840	157
AMIS0411	Pt (PGM) Platreef Ore Bushveld Complex	0.54	0.06	0.67	0.06	0.078	0.012	742	60	1 368	101
AMIS0413	Pt (PGM) Platreef tails Bushveld Complex, South Africa	0.265	0.032	0.349	0.036	0.044	0.006	579	36	1 030	47
AMIS0416	Pt (PGM) UG2 Ore Bushveld Complex, South Africa	1.46	0.18	0.75	0.12	0.14	0.04	93	11	1 094	148
AMIS0426	Internal Set Point Standard not certified	2.13	0.16	1.07	0.1	0.04	0.018				
AMIS0427	Internal Set Point Standard not certified	0.48	0.02	0.64	0.02	0.081	0.022				
AMIS0442	Pt (PGM) Platreef Ore Bushveld Complex South Africa	2.11	0.13	2.66	0.16	0.33	0.03	1 029	45	1 996	78
AMIS0443	Pt (PGM) Platreef Ore Bushveld Complex, South Africa	0.78	0.07	0.97	0.07	0.14	0.02	951	47	1 918	104
AMIS0448	Pt (PGM) Platreef Ore Bushveld Complex, South Africa	1.899	0.203	1.98	1.98	1.31	0.15	1 286	114	2 375	270
AMIS0450	Pt (PGM), Merensky Ore Bushveld Complex South Africa	3.17	0.2	1.56	0.09	0.22	0.02	990.2	94.3	2 004	145
AMIS0459	Pt (PGM) Pulps Bushveld Complex, South Africa	0.431	0.047	0.241	0.021	0.119	0.014	200.6	24.3	686	58
AMIS0484	Blank Silica Powder	0.005		0.005		0.001		2.5		8.5	
CDN-PGMS-19	CDN-PGMS-19 Pt Group Ore Reference Material	0.108	0.012	0.476	0.042	0.23	0.03				
CDN-PGMS-23	CDN-PGMS-23 Platinum Group Ore Reference Material	0.456	0.04	2.032	0.166	0.496	0.058				
CDN1	CDN-PGMS-1 Platinum Group Ore Reference Material	2.3	0.18	10.35	0.74	0.23	0.06				

CRM	Description	Pt Mean (g/t)	Pt 2SD (g/t)	Pd Mean (g/t)	Pd 2SD (g/t)	Au Mean (g/t)	Au 2SD (g/t)	Cu Mean (ppm)	Cu 2SD (ppm)	Ni Mean (ppm)	Ni 2SD (ppm)
CDN11	CDN-PGMS-11 Platinum Group Ore Reference Material	0.107	0.016	0.405	0.038	0.219	0.03				
CDN2	CDN-PGMS-2 Platinum Group Ore Reference Material	0.21	0.04	3.9	0.47						
CDN3	CDN-PGMS-3 Platinum Group Ore Reference Material	0.13	0.03	0.59	0.07	0.33	0.06				
CDN5	CDN-PGMS-5 Platinum Group Ore Reference Material	1.24	0.11	5.76	0.3						
CDN6	CDN-PGMS-6 Platinum Group Ore Reference Material	0.12	0.02	0.64	0.06	1.37	0.2				
CDN7	CDN-PGMS-7 Platinum Group Ore Reference Material	1.01	0.16	3.71	0.47	2.59	0.3				
CDN8	CDN-PGMS-8 Platinum Group Ore Reference Material	0.107	0.016	0.405	0.038	0.219	0.03				

Notes:

- 2SD = + Two Standard Deviations
- The Mean is the Expected Value
- Values are Certified

11.3.2.1 Quality Assurance / Quality Control Results for Set Point from 2010 to January 2018

Inserted field standards sent to Set Point have a low number of 117 exceptions (<1%) for the total 14 987 QC samples submitted. The results for only 8 samples (0.05%) were not resolved. The largest error of 51 samples (43.59% of the total exceptions or 0.34% of the total QC samples submitted) is due to human error as a different standard was bagged than the standard specified on the sample sheet. Exceptions caused during laboratory operations and the analysis of samples were resolved for 29.91% of the exceptions or 0.22% of the total number of QC samples submitted. The number of results where repeats confirmed the original results and were accepted is 16.23% of the exceptions or 0.12% of all QC samples. This low number of unresolved exceptions is deemed acceptable for the updated Mineral Resource Statement.

Inserted field blanks sent to Set Point have a low number of 17 exceptions (0.11%) for the total of 15 180 QC samples submitted that have not been resolved. There is very little evidence of sample swaps, incorrect samples being prepared or contamination.

Inserted laboratory preparation duplicates for Set Point show good precision where 99% of all duplicate pairs have a HARD of less than 20% for each element. 255 (5.89%) of the preparation duplicates were repeated although only 36 repeats were necessary for PGEs. Results are deemed to be acceptable for all elements. All exceptions were discussed in detail for each element.

Inserted laboratory pulp duplicates for Set Point show good precision where 99% of all duplicate pairs have a HARD of less than 10% for each element for Pd, Cu, and Ni. Au has 93% of all duplicate pairs with a HARD of <10%. Au shows variability at grades > 2 g/t due to a possible nugget effect. Pt has 96% of duplicate pairs that are with a HARD of < 10%. Results are deemed to be acceptable for all elements.

Inserted laboratory standards for Set Point have acceptable results with a range of exceptions between 0.23% for Cu and Ni, 1.36% for Pt, 1.12% for Pd and 0.49% for Au. Most of the exceptions are due to AMIS0146 and AMIS427 being used. These are in-house standards that are not certified. Eight are reported as one standard but another standard was inserted and analysed. There are 25 exceptions (0.17%). that are unexplained or unresolved of the 14 531 samples analysed. This low number of unresolved exceptions is deemed acceptable for the updated Mineral Resource Statement.

Inserted laboratory blanks have exceptionally good results for the 10 442 QC samples analysed. There are no exceptions (>10 x the detection limit) for Pt, Pd, or Au. There is one sample for Cu and Ni that has results >10 x the detection limit (100 ppm). This is a possible sample swap or contamination. The laboratory does not allow blanks to be reported that are greater than 100 ppm for Cu or Ni. It is assumed that they are repeated along with affected samples until an acceptable result is achieved.

The results of the analysis have shown that the data reported by Set Point is acceptable with variability outside acceptable limits explained wherever possible.

11.3.2.2 Quality Assurance / Quality Control Results for Set Point Reported During 2018

Inserted field standards sent to Set Point have a low number of exceptions (<1% for each element) for the total 2 256 QC samples submitted. This low number of exceptions is well within accepted norms according to industry best practices.

Inserted field blanks sent to Set Point have a low number of 2 exceptions for Cu and Ni only (0.09%) for the total of 2 167 QC samples submitted that have not been resolved. There is very little evidence of sample swaps, incorrect samples being prepared or contamination. In general, the failure rate is deemed not to have a material effect on the data, with more than 99% of the assays falling within acceptable limits.

Inserted laboratory preparation duplicates for Set Point show good precision where 99% of all duplicate pairs have a HARD of less than 20% for each element. Results are deemed to be acceptable for all elements.

Inserted laboratory pulp duplicates for Set Point show good precision where 99% of all duplicate pairs have a HARD of less than 10% for each element for Cu and Ni. Au has 93% of all duplicate pairs with a HARD of <10%. Au shows variability at grades > 2 g/t due to a possible nugget effect. Pt has 96% of duplicate pairs that are with a HARD of < 10%. Pd has 99% of duplicate pairs that are with a HARD of < 10%. Results are deemed to be acceptable for all elements.

Inserted laboratory standards for Set Point have acceptable results with very few exceptions. Most of the exceptions are due to AMIS0146, AMIS0426 and AMIS427 being used. These are inhouse standards that are not certified.

Inserted laboratory blanks have exceptionally good results for the 1 719 QC samples analysed. There are no exceptions (> 10 X the detection limit) for all elements reported. It is assumed that they are repeated along with affected samples until an acceptable result is achieved.

The results of the analysis have shown that the data reported by Set Point during 2018 is acceptable with exceptions outside acceptable limits explained wherever possible.

11.3.2.3 Quality Assurance / Quality Control Results for Bureau Veritas

Results for QC samples reported by Bureau Veritas along with primary samples show that the data is acceptable with exceptions outside acceptable limits explained wherever possible.

Inserted blind standards reported by Bureau Veritas show acceptable results on Z-score graphs for most samples although AMIS0395 plots outside acceptable limits for Au. AMIS0395 is not a suitable standard for Au as the expected value of 0.095 g/t is less than 10 times the detection limit.

Inserted blind blanks reported by Bureau Veritas show acceptable results with more than 90% of the assays falling within acceptable limits. Numerous results for Au plot above the acceptable limit of 0.01 g/t (10 times detection) and indicates that that Bureau Veritas's detection limit for Au is closer to 0.005 g/t. There are also numerous failures (> 10 x detection) for Ni. This indicates that the detection limit for Ni is closer to 10 ppm rather than 2 ppm. Operationally, there is very little evidence of contamination, sample swaps or the incorrect sample being prepared.

Inserted laboratory preparation duplicates reported by Bureau Veritas show good precision where 98-99% of all duplicate pairs have a HARD of less than 20% for each element. Results are deemed to be acceptable for all elements. The percentage of Au samples with HARD within 20% is 95% which is slightly lower than for the other elements. Au is prone to a possible nugget effect. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection. The original analysis versus the duplicate analysis showed minimal irregular values. This indicates minimal sample swapping.

Inserted laboratory pulp duplicates reported by Bureau Veritas show good precision where 98-99% for all duplicate pairs have a HARD of less than 10% for each element. Results are deemed to be acceptable for all elements. The percentage of Au samples with HARD within 20% is 95%, which is slightly lower than for the other elements. Au is prone to a possible nugget effect. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection.

Inserted laboratory standards for Bureau Veritas have acceptable results with very few exceptions (AMIS0354 – 2 exceptions for Cu and AMIS0367 – 3 exceptions for Ni).

Inserted laboratory blanks for Bureau Veritas have acceptable results with more than 99% of the assays falling within acceptable limits. Rock-RSB is the only blank that shows results that are greater than the background. It is not a certified blank.

11.3.3 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 mineralisation being investigated. It should be noted that validation is different from verification; the latter implies 100% repeatability. The assay validation programme should entail the following activities.

- A re-assay programme conducted on standards that failed the tolerance limits set at two and three SDs from the round robin mean value of the reference material.
- Ongoing blind pulp duplicate assays.
- Check assays conducted at an independent assaying facility.

Re-assays are routinely completed for failed standards, laboratory coarse duplicates, and pulp duplicates before the acceptance of each batch and final QC sign-off by the Waterberg JV Resources database manager.

11.3.3.1 Quality Assurance / Quality Control Results for Field Duplicates Submitted to Set Point

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 are not routinely used on this project due to the assemblage of the core and the different comparative results relative to the primary samples. The only explanation is that the core is heterogeneous, and mineralisation is not evenly distributed (i.e. there is a nugget effect).

The core is split lengthwise during sampling. Half the core is sent as the primary sample for analysis. The other half of the core is retained to preserve the core record in terms of lithology, stratigraphy, and mineralisation. Field duplicates are taken by bagging the other half (or quarter) of the core and assigning a new sample number, which is then dispatched to the same laboratory for analysis.

Field duplicates (670) were submitted for analysis. Graphs showing the relative distribution of the elements (scatter plots with primary results on the X-axis and the corresponding field duplicate result on the Y-axis) as well Thompson-Howarth plots to show the precision obtained by re-analysis of the field duplicates were plotted for each element. The precision graphs show that field duplicates cannot be used to measure precision.

The percentage of Au samples with HARD within 20% is 74%, which is lower than for the other elements. Au is prone to a possible nugget effect. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection. Pt and Pd have percentages of 78% and 82%, respectively, where HARD is within 20%. This indicates that Pt and Pd are also prone to a nugget effect but to a lesser degree than Au.

Scatter plots of original results versus paired duplicate results show a lot of scatter relative to the regression line. The high number of results that differ cannot be due to sample mix-ups alone. The only explanation is a nugget effect confirming that mineralisation in drill hole core is not evenly distributed. There is a poor correlation between original results and paired field duplicate results.

11.3.3.2 Quality Assurance / Quality Control Results for Field Pulp Duplicates Submitted to Set Point

The purpose of having field pulp duplicates is to measure the precision of the primary laboratory.

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.

Field pulp duplicates (1 893) were submitted for analysis.

The percentage of Au samples with HARD within 10% is 82% which is lower than for the other elements. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection. The other elements all have a percentage of samples with HARD within 10% that is greater than 90%, which is acceptable.

Graphs showing the relative distribution of the elements (scatter plots with primary results on the X-axis and the corresponding field pulp duplicate results on the Y-axis) as well Thompson-Howarth plots to show the precision obtained by re-analysis of the field pulp duplicates were plotted for each element.

There is some scatter relative to the regression line on the scatter plots, which may be due to sample mix-ups. There is a good correlation between original results and paired field pulp duplicate results.

The norm is that precision should be less than or equal to 10% for field pulp duplicates when compared to primary samples. The graph for Pt shows that the best precision possible for field pulp duplicates relative to primary samples is less than 20% but more than 10%, which is outside acceptable limits. The paired results are far from each other. This better precision when compared to duplicates split from the core itself shows that field pulp duplicates are homogenised. The sample selection is different; however, there is something that still results in variability between the results for the original sample and the pulp duplicate. Further research would assist in investigating the causes of the variability.

There is moderate (for Au, Cu, and Ni) to good (for Pt and Pd) correlation between original sample results and the field pulp duplicate results although there is some scatter relative to regression lines for each element. This may be due to sample mix ups. Precision ranges from 10% to 20% depending on the element. Field pulp duplicates show better precision than field core duplicates, but precision is not as good as for coarse reject duplicates and laboratory pulp duplicates.

There is no issue with the laboratory precision as proven results for laboratory coarse reject duplicates and laboratory pulp duplicates do fall within acceptable limits of precision and variability. There may be a possibility that the results for the ore body are not normally distributed. This would affect the precision estimates shown by the graphs.

In general, re-assayed coarse rejects and pulp duplicates analysed at the same time as the primary samples show good correlation with the original sample with greater than 90% of the data falling within acceptable limits. Further submissions of pulp duplicates would provide better clarity in terms of assay validation to ensure that assays are repeatable within statistical limits for the styles of mineralisation being investigated.

11.3.4 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. Most 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, Set Point reporting results for the primary samples.

11.3.4.1 Quality Assurance / Quality Control Results for Umpire Samples Sent to Genalysis Prior to 2018

A HARD statistic was calculated for each element and for each sample analysed at both laboratories. This is not to measure precision as the laboratories are different. This to identify whether there is agreement between the results between the laboratories. Samples with significantly different results may have been mixed up during the repackaging process before dispatch to the umpire laboratory or during processing at the umpire laboratory. At least 90% of the samples should have a HARD within 10%.

Cu and Ni have more than 90% of the samples having a HARD that is greater than 90% showing that the results of the two laboratories are comparable. The percentage of Au samples with HARD within 10% is 73%, which is slightly lower than for the other elements. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection. The percentage of samples with HARD within 10% for Au, Pt (81%), and Pd (81%) is lower than the acceptable limit of 90%. The cause of this is not clear. Sample mix-ups are one possible cause but not to such an extent. All results with a HARD greater than 10% are less than 5 g/t for Pt. Further analysis may confirm this phenomenon or may indicate that this poor performance is specific to this dataset.

Scatter plots and Q-Q plots were plotted for each element. Scatter around the regression lines on each of the plots are equally distributed with acceptable correlation and there is no bias indicated by either of the laboratories for Pt, Pd, Cu and Ni. Au does show some scatter above grades of 2 g/t with less correlation than Pt and Pd. Set Point results show a positive bias for grades greater than 4 g/t relative to Genalysis results. There is a slight positive bias for Genalysis Ni results when compared to Set Point results on the Q-Q graph.

11.3.4.2 Quality Assurance / Quality Control Results for Umpire Samples Sent to Genalysis in 2018

Umpire samples (602) were sent to Genalysis during 2018. The Genalysis results confirm the satisfactory performance of the primary laboratory, Set Point. Genalysis results show better recovery of Au and Ni during analysis at higher degrees of mineralisation. Results over common sample ranges in mineralisation for both laboratories are similar for all elements.

A HARD statistic was calculated for each element and for each sample analysed at both laboratories. This is not to measure precision as the laboratories are different. This to identify whether there is agreement between the results between the laboratories. Samples with significantly different results may have been mixed up during the repackaging process before dispatch to the umpire laboratory or during processing at the umpire laboratory. At least 90% of the samples should have a HARD within 10%.

For Pt, Pd, Cu, and Ni, the percentage of samples having a HARD within 10% are within acceptable limits of approximately 90-97%. There is an improvement relative to the previous 665 samples analysed. The percentage of samples with HARD within 10% for Au, Pt, and Pd is lower than the acceptable limit of 90% for the previous 665 samples. What caused the low percentages for the previous samples is not known. Sample mix-ups may have caused these discrepancies. The results for the 2018 indicate that there may also have been sample swaps or samples having a nugget effect, but such samples are within acceptable limits. The percentage of Au samples with HARD within 10% Is 67.2%, which is lower than for the other elements and lower than the 73% for the previous 665 samples. Au is prone to a possible nugget effect. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection.

The scatter around the regression line for Pt, Pd, Cu, and Ni are equally distributed and there is a good correlation of the duplicate pairs. Results are within acceptable limits. Genalysis shows a positive bias for Pt due to better recovery during analysis

The distribution graphs for each laboratory and each element are similar.

Compared to Pt and Pd, Au shows less correlation and more scatter around the regression line for Set Point versus Genalysis results. Genalysis results have a positive bias as indicated by the regression line. This may be due to better recovery of Au during the analytical process by Genalysis. The R2 of 0.9164 for Au is acceptable. This means that Set Point Au results are conservative. It is better to have an underestimate of grade by a primary laboratory than an overestimate. There is a positive bias for Genalysis Ni results > 5 000 ppm as there is better Ni recovery during analysis relative to Set Point. This means that Set Point Ni results are conservative.

11.3.4.3 Quality Assurance / Quality Control Results for Umpire Samples Sent to Bureau Veritas

Samples (772) were sent to both Set Point and Bureau Veritas.

A HARD statistic was calculated for each element and for each sample analysed at both laboratories. Samples with significantly different results may have been mixed up during the repackaging process before dispatch to the umpire laboratory or during processing at the umpire laboratory. At least 90% of the samples should have a HARD within 10%. Cu and Ni show good comparability between laboratories with 97% of samples having a HARD within 10%. Pt has 92% of the samples with HARD within 10%. This is acceptable. The percentage of Au samples with HARD within 10% is 45%, which is very low. Au is also subject to higher variability due to the analytical technique used (fire assay with Pb collection) at low grades (<0.1 g/t). Au also has more samples with results closer to the limit of detection. The percentage of samples with HARD within 10% for Au and Pt (87%) is lower than the acceptable limit of 90%. The cause of this is not clear. Sample mix-ups are one possible cause but not to such an extent. Results with a HARD greater than 10% for Pt may indicate a positive bias in results from Bureau Veritas.

The distribution graphs for each laboratory and each element are comparable.

The correlation between Set Point and Bureau Veritas results is acceptable for Pt, although there is an observed positive bias for a few Bureau Veritas results when compared to Set Point for grades greater than 2 g/t. There is some scatter at grades less than 4 g/t for Pd and Bureau Veritas results show a positive bias for some samples when compared to Set Point Pd results for grades greater than 2 g/t. The correlation between Bureau Veritas and Set Point for Au is poor with an R^2 of 0.889. Bureau Veritas has a negative bias when compared to Set Point results for Au. Au shows a correlation up to a grade of 1 g/t, which is within the range of most mineralised samples.

Cu results are comparable up to 3 000 ppm, which is within the range of most mineralised samples. There is a negative bias of Bureau Veritas results when compared to Set Point results above 3 000 ppm.

There is a good correlation between Set Point and Bureau Veritas results for Ni. The result distributions are comparable up to values of 4 000 ppm for Ni which is in the range of most mineralised samples.

11.3.5 Sample Security

The QA/QC practice of Waterberg JV Resources 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 CJM's opinion, the QA/QC procedures as well as the sample preparation and security procedures are adequate to allow the data to be used with confidence in the Mineral Resource Estimate.

12 DATA VERIFICATION, AUDITS, AND REVIEWS

12.1 Verification of Data by Qualified Person

CJM conducted data verification as part of the Mineral Resource Estimate for the Waterberg Project as explained below.

Printed logs for 90% of the holes were checked with the drilled core. The depths of mineralisation, 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.

Regarding missing 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 may be present. Given the verification process, should there be a bias or inconsistency in the data, the error will be of no material consequence in the interpretation of the model or evaluation.

The data was checked for errors and inconsistencies at each step of handling. The data was rechecked at the stage where it was captured into the deposit-modelling software. In addition to ongoing data checks by project staff, the senior management and directors of Waterberg JV Resources completed spot audits of the data and processing procedures. Audits were also completed on the recording of drill hole information, assay interpretation, and final compilation of the information.

The individuals in Waterberg JV Resources' senior management and certain directors of the company who completed the tests and designed the processes were non-independent mining or geological experts.

The QP's opinion is that the data is adequate for use in Mineral Resource estimation.

12.3 Possible Reasons for not Completing a Data Verification Process

All Waterberg JV Resources data was verified before being statistically processed. Copies of the QA/QC data analysis can be provided on request.

12.4 Independent Audits and Reviews

Each Mineral Resource Estimate and Report to date involved an independent audit and review of the data and procedures used by Waterberg JV Resources. This included site visits, drill hole position verification, logging verification, assay verification, visits and audits on laboratories used among other checks to ensure accuracy of the Mineral Resource Statement.

An independent high-level review of the Mineral Resource Estimate by the QP was completed by QPs at AMEC GRD SA (Netherlands) (AMEC). The AMEC review made comments on the methodologies applied by the QP. The AMEC review identified moderate to low risks and these were considered by the QP in formulation of the conclusions of this Technical Report compliant with NI 43-101 standards.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testing on the Waterberg material was initiated by Waterberg JV Resources in 2013 as part of the Preliminary Economic Assessment (PEA) and included metallurgical characterisation of a single T-South zone sample and a single F-Central zone sample at SGS, South Africa. Further investigative testwork was performed on a F-Central zone composite sample, under the management of JOGMEC during 2013 to 2014. More testwork was conducted by MINTEK between August 2014 and September 2016 as part of the PFS. The aim of this campaign was to further assess the metallurgical response and to generate enough data to support the PFS study design.

The DFS metallurgical testing focused on evaluating the degree of variability in metallurgical response of the various mining zones within the Waterberg deposit. The DFS testwork was conducted at MINTEK during 2018 to 2019.

13.2 Historical Metallurgical Testwork

13.2.1 Comminution Testwork

Comminution testwork on the following Waterberg lithology units were conducted at MINTEK between 2013 and 2016: T-South (T2a sample), F-Central (F4 sample), F-Boundary drill cores, and F-North drill cores.

The comminution characterisation testwork 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, each sample was subjected to BBWi and MINTEK grindmill testing. This allowed for comparison and benchmarking of the different samples against each other by means of various simulation methods. Refer to Table 13-1 for a summary of the results on the tests conducted per lithology unit.

Table 13-1: Summary of Waterberg Samples Comminution Test Results

Waterberg Lithology Unit (Sample Reference)	SG	SMC	UCS			CWi	Ai	BRWi	BBWi	
	t/m³	A*	Min MPa	Max MPa	Avg MPa	Avg kWh/t	Avg g	1 180 µm kWh/t	106 µm kWh/t	75 µm kWh/t
T-South (T2a sample)	2.92	51.6	63.4	120.1	83.0	10.8	0.194	16.28	19.54	21.63
F-Central FH Upper (F1)	2.98	30.8	87.1	244.9	196.0	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 historical comminution testwork results can be summarised listed below.

- The SMC test classified the T-South material as being of medium hard competency, while both the F-Central samples were classified as being of hard competency.
- The UCS test classified the T-South material as soft while the F-Central samples were classified as hard.
- The CWi test results classified the T-South and F-Central material as soft.
- The bond Ai test results indicated that each of the Waterberg samples tested were moderately abrasive.
- BRWi and BBWi test results classified all the samples all as hard to very hard.

13.2.2 Flotation Testwork

Three separate testwork campaigns were conducted between 2013 and 2016.

- PEA / scoping study testwork in 2013 as part of the PEA and included metallurgical characterisation of a single T-South sample and a single F-Central sample at SGS, South Africa.
- Investigative testwork was performed on a F-Central composite sample under the management of JOGMEC from 2013 to 2014.
- Four phases of PFS testwork was conducted by MINTEK between August 2014 and September 2016 to assess the metallurgical response and to generate enough data to support the PFS study design.

Refer to Table 13-2 for a summary of the historical flotation testwork.

Table 13-2: Summary of Historical Flotation Testwork

Campaign Description (Laboratory Used)	Scope of Work	Summary of Key Findings
PEA, 2013 (SGS, South Africa)	Preliminary mineralogical characterisation, and single stage (MF1) cleaner, bench-scale flotation test was conducted on two area composite samples. • T-South @ 6.7 g/t 3E[1] • F-Central @ 3.6 g/t 3E	• Quantitative mineralogy highlighted that the T-South sample had better beneficiation properties, compared to the F-Central sample, due to better liberation. This was confirmed by flotation testwork, with T-South sample showing a higher flotation rate and maximum recovery. • T-South sample contained more clayish minerals and floatable gangue, compared to F-Central. • The single MF1 cleaner flotation test on the F-Central sample reported a 76% 3E recovery at 18 g/t; while the T-South sample achieved an 85.8% recovery at 60 g/t.
JOGMEC scoping 2013 – 2014. (SGS)	Evaluating the response of a single F-Central composite sample (3.52 g/t 4E[2]) when applying different reagent schemes in a MF1 flowsheet.	• 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 product. • 74% of the Cu was recovered, while 45% of the Ni was recovered.
PFS Phase 1a 2014 - 2015 (MINTEK)	The Phase 1a campaign targeted the production of a typical concentrate for preliminary third-party smelting and PGM refining discussions, using two composite samples from F-Central area at 2.8 g/t 3E, and 3.2 g/t 3E. The scope of work included the following items. • MF1 (mill-float) & MF2 (mill-float-mill-float) bench-scale flotation testing. • Mineralogical characterisation of final concentrate. • Magnetic separation testing on final concentrate aimed at reducing the Fe content in the product.	• MF1 circuit utilising Oxalic acid and Thiourea achieved concentrate grades between 97 g/t 3E and 145 g/t 3E while achieving 70.6% to 81.0% recovery. Cu recovery varied between 73.8% to 86.9%, with Ni recovery ranging from 38% to 46.9%. • MF2 circuit utilising typical South African reagents achieved concentrate grades between 91.9 g/t 3E and 115 g/t 3E while achieving 78.7% to 81.8% recovery. Cu was recovered at 83.1%, with Ni recovery ranging from 35.5% to 38.5%. • The MF1 circuit tests with Oxalic acid and Thiourea achieved higher Fe and S in the final products. • The mineralogy search showed that the primary circuit product was mainly Pt/Pd-arsenides and Pd-bismuth tellurides, with minor Pt-sulphides. The secondary circuit product was primarily Pt/Pd-arsenides and Pd-bismuth tellurides. • 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% while 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. Ni and Cu 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. • A full chemical analysis, by XRF, did not reveal any deleterious elements in the F-Central product. • The magnetic separation testing was not successful in reducing the Fe content in the product, without negatively effecting the recovery. PGE losses to the Fe fraction of between 15% and 38% was reported.
PFS Phase 1b 2014 - 2015 (MINTEK)	The Phase 1b flotation campaign focused on determining the optimum flotation flowsheet to process the F-Central material. The scope of work included the following items. • MF1 and MF2 bench-scale and locked cycle flotation testing on a composite sample of the F-Central material at 2.95 g/t 3E. • Mineralogical characterisations of the F-Central composite sample.	• Head grade analysis by a variety of analytical methods, resulted in notable assay variability despite several re-assay checks. This was attributed to coarse nugget effects, mostly noted on the Au and Pd assays. • MF2 tests revealed that extensive scavenger and cleaner circuit capacity is essential, while low primary recleaner and secondary recleaner mass pulls are to be targeted in order to maximise the final product grade. Ni recovery averaged 35% and Cu recoveries averaged 80%. The inclusion of a regrind stage in the MF2 circuit did not show any benefits in terms of recovery or product grade. • The use of an alternative collector (sodium isopropyl xanthate) in the MF1 testing improved both the PGE and Ni recoveries at similar PGE grades, although it also resulted in significantly higher Fe content in the final product. The addition of Oxalic acid and Thiourea in the MF1 circuit resulted in an increase in PGE recovery and grade; however, reduced Ni recoveries were reported. Regrinding of the slow floating fraction prior to scavenger cleaning did not show any benefits in terms of recovery or product grade. • Comparing MF2 open circuit vs MF1 open circuit tests, it was noted that the F-Central material performance was similar between the two circuits. The MF1 circuit achieved the higher Ni recovery (42% vs 38%), while the MF2 circuit achieved the higher Cu recovery (~80% vs ~66%).

[1] Pt, Pd, and Au
[2] Pt, Pd, Rh, and Au

Campaign Description (Laboratory Used)	Scope of Work	Summary of Key Findings
PFS Phase 2, 2014 - 2015 (MINTEK)	Campaign focused on evaluating the effect of various collector schemes on flotation response using a MF1 flowsheet. The aim was to improve the recovery of both the PGEs and Ni. The testwork was conducted using the F-Central master composite sample (Phase 1b) and included bench-scale collector optimization tests.	• There was 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 Ni, when compared to the baseline test. The result was supported by the mineralogical characterisation work which indicated that the pentlandite was locked in fine gangue minerals. • The addition of $CuSO_4$ to the rougher circuit resulted in ~1% higher PGE recovery.
PFS Phase 3, 2014 - 2015 (MINTEK)	The Phase 3 flotation campaign evaluated the flotation response of a composite F-North sample (3.51 g/t 3E) from the Early Dawn farm area, when applying the flowsheet developed in Phase 1b. The scope of work included the following items. • MF1 and MF2 flotation testing. • Mineralogical study on the flotation feed sample.	• The MF2 testing indicated similar PGE rougher recoveries (approximately 86%) to the F-Central master composite sample. The test did, however, highlight that significantly lower upgrade ratios (UGR) could be expected for the F-North ore. It was noted that the F-North material PGE recovery was highly sensitive to product grade and mass pull. Testing achieved a high-grade final product of 133 g/t (3E) at 71% recovery, or a lower grade 53 g/t (3E) product at 81% recovery. The Cu and Ni 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. • The MF1 testing achieved a high-grade final product of 91 g/t (3E) at 76% recovery, or a lower grade 56 g/t (3E) product at 81% recovery. Cu an Ni recoveries were 87% and 56% respectively for the lower grade product. • Comparing the results for MF2 open circuit tests vs. the MF1 open circuit tests, it was noted that the F-North composite sample achieved a marginally higher PGE recovery for the MF2 circuit. The MF1 circuit achieved the higher Ni recovery (56% vs 54%), while both circuits achieved similar Cu recoveries of ~88%.
PFS Phase 4, 2014 - 2015 (MINTEK)	Phase 4 involved further MF1 and MF2 grind and reagent optimization testwork on the following items. • Various T-South material composite samples (4.0 – 4.6 g/t 3E). • F-Boundary master composite sample (3.6 g/t 3E). • Mine Blend sample comprising a 50% T-South:50% F-Central blend at 3.4 g/t 3E.	• MF2 grind optimisation tests on T-South samples indicated that the sample was amenable to a finer secondary grind (90%- 75µm) as it resulted in a higher PGE and Cu recovery. Similar Ni recoveries were noted at the finer grind. A finer grind on the MF1 flowsheet did not result in a recovery improvement. • T-South material achieved significantly higher PGE recoveries with the MF2 compared to the MF1 circuit. The MF1 circuit achieved the higher Cu recovery (88% vs 84%) whereas the MF2 circuits achieved slightly higher Ni recoveries (47% vs 45%). • Testing of the F-Boundary composite sample achieved an 85% 3E recovery to produce a 71 g/t product (UGR of 20) when targeting 80% - 75µm secondary grind. • Grind optimisation tests on the Mine Blend composite sample indicated that a secondary grind of 90% - 75µm was detrimental to the 2E + Au recovery, as a 4% lower recovery was reported at a UGR of 20 (~ 70 g/t 3E product). The finer grind resulted in increased Cu recovery (88% vs 86%); however, the finer grind had a negative impact on the Ni recovery reported (42% vs 46%). • Different individual metal recoveries were noted for the precious metals. Pt recovery was generally higher than Pd recovery (between 3% – 7% on the T-South samples). Au recovery was generally the lowest, being between 12% - 18% lower than the Pt recovery. • Reagent optimization testwork 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 PGM recovery. The use of a KU92 guard depressant showed potential to reduce S 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 testwork to improve the overall 3E recovery, when compared to the F-Central flowsheet.

13.2.3 Other Testwork

In addition to the comminution and flotation testwork, the following further testwork was conducted during 2013 to 2016.

- Heavy liquid separation (HLS) testing.
- Flotation tailings dewatering, filtration, and rheology testing.

Refer to Table 13-3 for a summary of the above.

Table 13-3: Summary of Other Historical Testwork

Campaign Description (Laboratory Used)	Scope of Work	Summary of Key Findings
HLS, 2014 (MINTEK)	HLS testwork was conducted on a single F-Central drill core sample to assess the amenability of the material to density pre-concentration.	• The results from the HLS testwork indicated limited scope for pre-concentration based on density. Albeit that a waste rejection of up to 40% could be achieved, high precious metal losses (in excess of 20%) rendered the application uneconomical.
Tailings Dewatering, 2015 (Vietti Slurrytec, South Africa)	Tailings dewatering testwork was conducted on a F-Central composite flotation tailings sample (at a grind of 80% passing 75µm). The scope of work included the following items. • Particle size determination. • High-level mineralogical characterisation. • Thickening testwork. • Filtration testwork. • Sample preparation of a thickener underflow sample, which was submitted to Paterson & Cooke Consulting Scientists in South Africa for rheological characterisation testwork.	• The material was found to be non-settling if unflocculated, due to the presence of smectite and talc clays, and the low conductivity of the process water used. • 200 g/t Magnafloc 1597 was selected as conditioning agent in conjunction with 20 g/t Magnafloc 919 as flocculant. • The optimum thickener feed solids concentration: 10% weight/weight (w/w). • The optimum solids flux rate for a high rate thickener: 0.4 t/h/m^2. Underflow slurry solids concentration of 60% w/w was achieved. • The optimum solids flux rate for a paste thickener: 0.5 t/h/m^2. Underflow slurry solids concentration of 67% w/w was achieved. • The un-sheared vane yield stress of the sample was 197 Pa under high rate conditions and 356 Pa under paste conditions at an underflow solids concentration of 63% w/w and 71% w/w, respectively. • The material did dewater under vacuum filtration, although 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/m^2.

13.3 Definitive Feasibility Study Metallurgical Testwork

The following section summarises the current metallurgical testwork outcomes, as conducted by MINTEK under the management of Waterberg JV Resources and DRA, between March 2018 and June 2019.

The DFS testwork campaign initially focused on evaluating the degree of variability in the comminution parameters and flotation response of each of the Waterberg lithology units (i.e. T-South, F-Central, F-North, F-Boundary, and F-South) using individual drill core samples selected from the anticipated early mining areas, and processing using the flowsheet as developed during the PFS. Following the variability testing on the individual lithology units, further flotation testwork was conducted on two different Mine Blend samples as directed by the mining plan, on composite samples.

These testwork results were used, in conjunction with the PFS testwork results, to derive the recovery estimates.

13.3.1 Testwork Scope

Refer to Table 13-4 for a summary of the testwork conducted as part of the FS.

Table 13-4: Summary of Definitive Feasibility Study Testwork Scope

Testwork Description	Laboratory	Sample Info	Scope of Work
Comminution Variability Testing	MINTEK	2 x T-South Area Composite Samples 3 x T-South Individual Cores 4 x F-South Individual Cores 10 x F-Central Individual Cores 4 x F-Boundary Individual Cores 5 x F-North Individual Cores	SMC Bond Ai Test BBWi
Flotation Variability Testing	MINTEK	9 x T-South Individual Cores 5 x F-South Individual Cores 19 x F-Central Individual Cores 9 x F-Boundary Individual Cores 9 x F-North Individual Cores	Open Circuit MF2 Test on each Individual core, applying optimised PFS flowsheet Parameters.
Mine Blend Open Circuit Flotation Testing	MINTEK	4 x Mine Blend Composites (Mine Blend 1, Mine Blend 4, Mine Blend 5, Mine Blend 6)	Open Circuit MF2 test on each Mine Blend composite sample.
Mine Blend Locked Cycle Flotation Testing	MINTEK	1 x Mine Blend Composite (Mine Blend 6)	MF2 locked cycle test on Mine Blend 6 composite sample.
Backfill Sample Preparation (MF1 Testing)	MINTEK	2 x Mine Blend Composites (Early Mine Blend, Late Mine Blend)	Open circuit MF1 test on each Mine Blend composite sample to generate enough tailings for backfill testing.
Ni & PGE Entitlement Study	XPS, Canada	4 x T-South Composite Samples 2 x F-Central Composite Samples 1 x F-Boundary Composite Sample 1 x F-North Composite Sample	PGM, Cu, and Ni Deportment Study on each of the composite samples.

13.3.2 Sample Selection and Characterisation

Drill core samples consisting of ¾ NQ core from each lithology unit, were selected based on grade, spatial location, and the sample mass available to represent a fair spread of the anticipated mining area and head grades. Due to the mass requirements to complete the scoped comminution testing, it was required to generate area composite samples for the T-South comminution testing, where the individual drill cores could not supply enough sample mass. Refer to Figure 13-1 through Figure 13-3 for illustration of the sample positions from the South, Central, and North Complex.

Figure 13-1: South Complex Sample Location Map

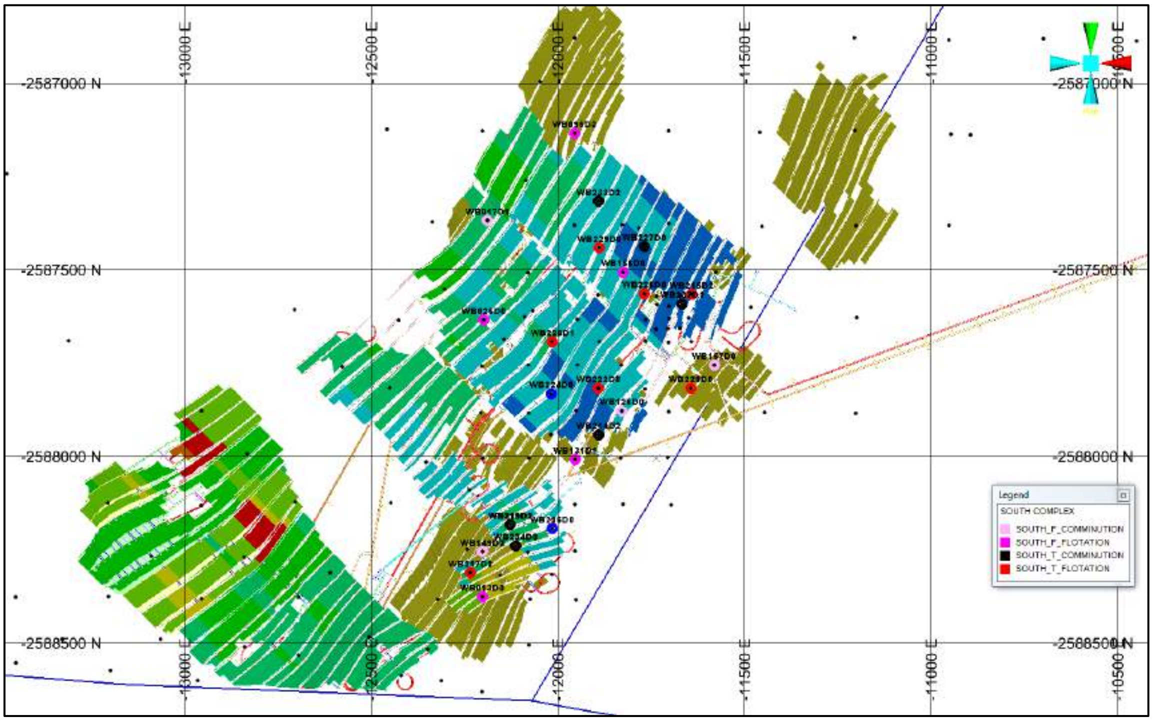


Figure 13-2: Central Complex Sample Location Map

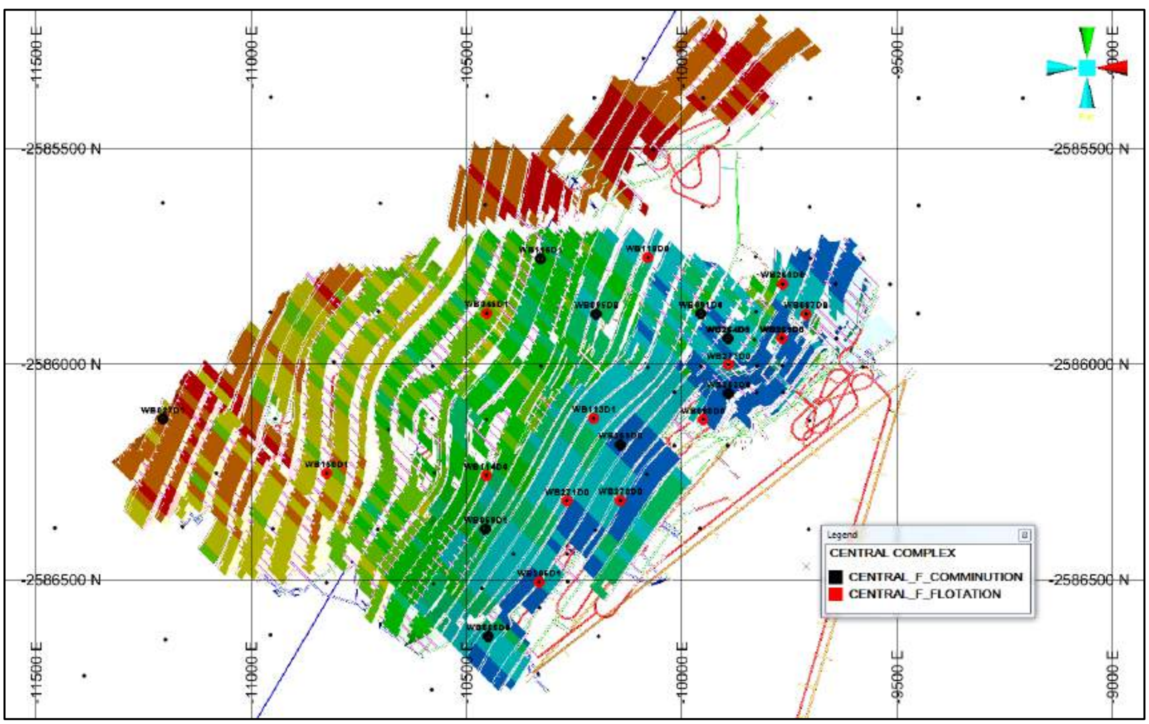
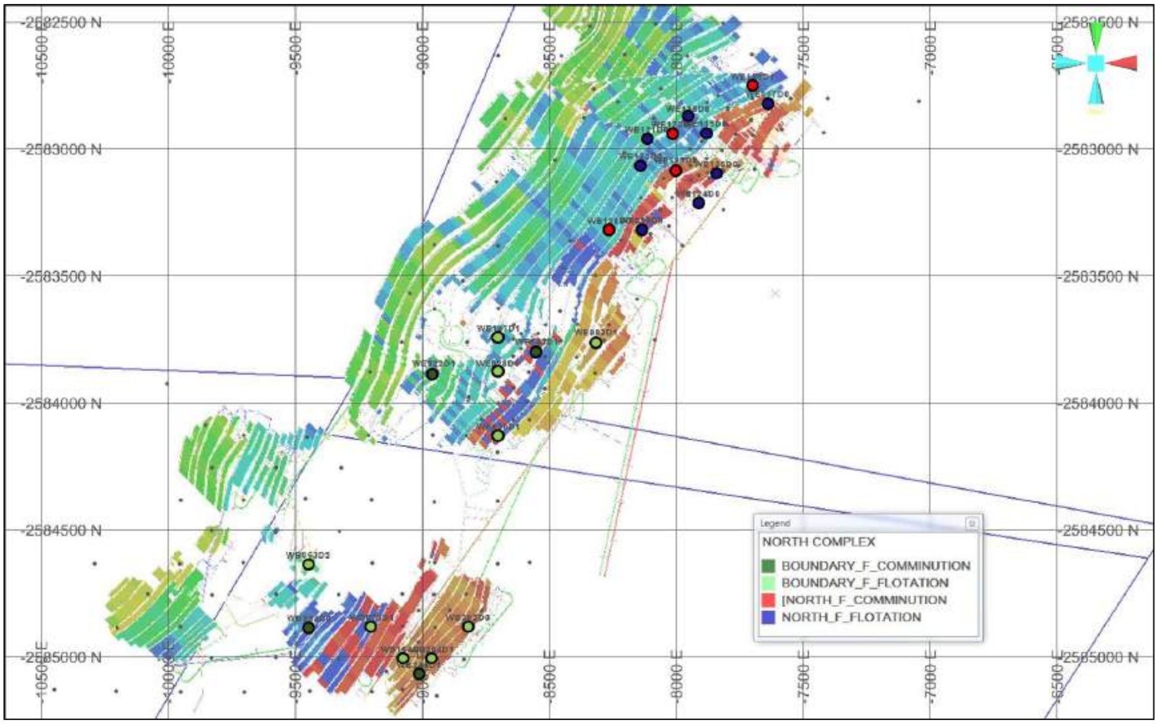


Figure 13-3: North Complex Sample Location Map



13.3.2.1 T-South

A total of 18 different, ¾ NQ drill core samples were used for testing of the T-South material comminution and flotation characteristics. The samples selected from the T-South material ranged from 2.44 to 5.57 g/t 4E. Refer to Table 13-5 and Table 13-6 for a summary of the comminution and flotation samples, respectively.

Table 13-5: Summary of T-South Comminution Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E[3] (g/t)	Cu (%)	Ni (%)
COM TZ VAR 1	WB216D0	231.00	235.50	0.53	0.50	0.02	1.06	2.10	0.25	0.11
COM TZ VAR 1	WB217D2	223.00	225.97	0.82	0.24	0.02	1.36	2.44	0.36	0.14
COM TZ VAR 1	WB219D1	267.95	271.00	0.73	0.57	0.02	1.53	2.86	0.32	0.13
COM TZ VAR 1	WB234D0	223.50	225.50	1.28	2.61	0.06	0.69	4.63	0.16	0.09
COM TZ VAR 2	WB214D2	251.62	260.50	0.65	0.89	0.02	0.24	1.81	0.05	0.04
COM TZ VAR 3	WB224D0	370.50	385.50	1.07	1.69	0.04	0.72	3.52	0.11	0.06
COM TZ VAR 4	WB227D0	321.50	324.00	0.81	1.51	0.04	0.34	2.70	0.07	0.06
COM TZ VAR 4	WB233D1	501.72	507.00	2.67	4.84	0.12	3.61	11.24	0.20	0.10
COM TZ VAR 5	WB237D1	237.00	253.12	1.94	3.43	0.09	1.21	6.66	0.21	0.09

Table 13-6: Summary of T-South Flotation Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
FT TZ VAR 1	WB228D1	431.00	436.15	0.83	1.34	0.04	0.23	2.44	0.03	0.02
FT TZ VAR 2	WB217D1	223.50	226.50	1.01	0.31	0.02	1.73	3.07	0.41	0.17
FT TZ VAR 3	WB226D0	322.50	329.76	0.82	2.07	0.05	0.26	3.20	0.04	0.02
FT TZ VAR 4	WB219D2	268.00	271.15	0.89	0.99	0.03	1.80	3.72	0.46	0.20
FT TZ VAR 5	WB229D0	450.00	455.50	1.23	2.03	0.05	0.88	4.19	0.07	0.04
FT TZ VAR 6	WB222D0	295.33	305.50	1.08	2.42	0.06	0.76	4.33	0.24	0.13
FT TZ VAR 7	WB215D2	239.00	245.00	1.33	1.93	0.05	1.20	4.52	0.13	0.06
FT TZ VAR 8	WB220D0	178.00	182.20	1.37	2.83	0.07	0.47	4.74	0.07	0.03
FT TZ VAR 9	WB233D2	501.10	508.50	1.48	3.04	0.07	0.98	5.57	0.10	0.05

[3] Anticipated grade from geology sampling.

13.3.2.2 F-South

A total of nine different, ¾ NQ drill core samples were used for testing of the F-South material comminution and flotation characteristics. Refer to Table 13-7 and Table 13-8 for a summary of the comminution and flotation samples, respectively.

Table 13-7: Summary of F-South Comminution Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
COM SF VAR 1	WB157D0	419.67	456.36	0.82	1.64	0.04	0.15	2.65	0.02	0.10
COM SF VAR 2	WB126D0	624.50	642.07	Not Determined (ND)	ND	ND	ND	2.69	ND	ND
COM SF VAR 3	WB017D1	1 033.00	1 042.50	1.24	2.40	0.06	0.23	3.92	0.10	0.18
COM SF VAR 4	WB149D0	718.65	730.00	1.40	2.68	0.07	0.16	4.31	0.02	0.12

Table 13-8: Summary of F-South Flotation Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
FT SF VAR 1	WB131D1	693.50	696.68	1.05	1.98	0.05	0.09	3.17	0.02	0.11
FT SF VAR 2	WB156D0	750.96	771.00	1.36	2.59	0.06	0.22	4.24	0.04	0.11
FT SF VAR 3	WB026D0	912.25	922.75	1.41	2.61	0.06	0.26	4.34	0.07	0.11
FT SF VAR 4	WB096D3	1 005.00	1 007.50	2.06	3.74	0.20	0.23	6.24	0.03	0.17
FT SF VAR 5	WB013D0	663.00	679.00	2.07	4.04	0.10	0.30	6.51	0.08	0.18

13.3.2.3 F-Central

A total of 28 different, ¾ NQ drill core samples were used for testing of the F-Central material comminution and flotation characteristics. The samples selected from the F-Central material ranged from 2.42 to 7.60 g/t 4E. Refer to Table 13-9 and Table 13-10 for a summary of the comminution and flotation samples, respectively.

Table 13-9: Summary of F-Central Comminution Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
COM SFC VAR 1	WB027D1	1 165.0	1 172.0	1.41	3.37	0.07	0.19	5.04	0.12	0.28
COM SFC VAR 2	WB264D0	471.8	478.8	1.43	3.41	0.08	0.23	5.15	0.16	0.21
COM SFC VAR 3	WB116D1	619.0	628.0	1.41	2.60	0.06	0.08	4.16	0.02	0.14
COM SFC VAR 4	WB069D1	567.0	581.5	1.18	1.93	0.07	0.13	3.32	0.09	0.19
COM SFC VAR 5	WB095D0	601.5	609.0	0.89	2.14	0.02	0.16	3.20	0.07	0.20
COM SFC VAR 6	WB269D0	418.0	430.0	1.55	2.48	0.07	0.12	4.21	ND	ND
COM SFC VAR 7	WB091D0	486.3	493.3	0.94	2.27	0.06	0.15	3.42	0.10	0.21
COM SFC VAR 8	WB259D1	380.6	385.2	0.79	1.92	0.04	0.12	2.88	0.10	0.24
COM SFC VAR 9	WB263D0	403.0	409.8	1.37	3.47	0.08	0.24	5.15	0.11	0.22
COM SFC VAR 10	WB085D0	412.0	427.5	1.55	3.28	0.08	0.18	5.11	0.05	0.19

Table 13-10: Summary of F-Central Flotation Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
FT SFC VAR 1	WB271D0	453.7	457.6	0.70	1.58	0.04	0.11	2.42	0.10	0.23
FT SFC VAR 2	WB114D0	654.5	661.5	0.81	1.82	0.04	0.14	2.81	0.13	0.28
FT SFC VAR 3	WB277D0	367.5	372.6	0.83	1.97	0.05	0.15	3.00	0.06	0.20
FT SFC VAR 4	WB113D1	553.0	559.0	0.84	2.00	0.05	0.15	3.04	0.07	0.19
FT SFC VAR 5	WB259D0	447.0	454.5	0.91	2.05	0.05	0.13	3.13	0.06	0.16
FT SFC VAR 6	WB118D0	568.0	579.5	0.93	1.99	0.05	0.24	3.21	0.07	0.19
FT SFC VAR 7	WB263D1	439.6	446.1	0.86	2.30	0.05	0.16	3.37	0.10	0.21
FT SFC VAR 8	WB090D0	336.0	343.0	0.99	2.28	0.03	0.15	3.46	0.07	0.20
FT SFC VAR 9	WB091D1	548.5	550.5	0.97	2.39	0.04	0.18	3.58	0.11	0.17
FT SFC VAR 10	WB206D1	403.5	409.5	1.15	2.35	0.08	0.06	3.65	0.02	0.12
FT SFC VAR 11	WB087D0	329.5	332.3	1.08	2.48	0.03	0.16	3.76	0.04	0.19
FT SFC VAR 12	WB150D1	906.0	925.5	1.10	2.87	0.06	0.21	4.25	0.12	0.22
FT SFC VAR 13	WB260D0	391.7	401.9	1.21	2.84	0.07	0.18	4.30	0.07	0.20
FT SFC VAR 14	WB095D2	600.0	605.0	1.19	2.91	0.07	0.26	4.42	0.10	0.18
FT SFC VAR 15	WB264D0	442.7	452.5	1.28	3.19	0.07	0.22	4.76	0.11	0.26
FT SFC VAR 16	WB046D1	802.0	815.5	1.49	3.59	0.10	0.24	5.42	0.10	0.24
FT SFC VAR 17	WB087D2	329.0	336.0	1.51	3.67	0.08	0.22	5.48	0.11	0.27
FT SFC VAR 18	WB270D0	352.8	363.1	1.69	4.17	0.09	0.34	6.30	0.14	0.22
FT SFC VAR 19	WB085D1	416.0	429.0	2.39	4.81	0.12	0.28	7.60	0.08	0.22

13.3.2.4 F-Boundary

A total of 13 different, ¾ NQ drill core samples were used for testing of the F-Boundary material comminution and flotation characteristics. The samples selected from the F-Boundary material ranged from 2.59 to 5.70 g/t 4E. Refer to Table 13-11 and Table 13-12 for a summary of the comminution and flotation samples, respectively.

Table 13-11: Summary of F-Boundary Comminution Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
COM SFB VAR 1	WB093D0	718.00	731.00	0.97	2.15	0.03	0.18	3.34	0.10	0.22
COM SFB VAR 2	WB249D1	282.00	293.00	1.22	2.89	0.07	0.25	4.43	0.15	0.26
COM SFB VAR 3	WE022D1	579.00	625.00	0.76	1.68	0.04	0.14	2.62	0.10	0.26
COM SFB VAR 4	WE143D1	383.00	404.00	0.86	1.85	0.04	0.11	2.86	0.10	0.22

Table 13-12: Summary of F-Boundary Flotation Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
FT SFB VAR 1	WB079D1	527.00	543.00	0.77	1.66	0.04	0.13	2.59	0.05	0.22
FT SFB VAR 2	WE083D1	247.00	263.00	0.90	1.96	0.05	0.13	3.05	0.11	0.20
FT SFB VAR 3	WE030D1	326.00	353.00	1.07	2.08	0.05	0.11	3.32	0.08	0.23
FT SFB VAR 4	WB053D2	810.00	829.00	0.98	2.28	0.03	0.17	3.45	0.14	0.26
FT SFB VAR 5	WB154D0	378.00	390.00	1.23	2.27	0.06	0.25	3.81	0.11	0.22
FT SFB VAR 6	WE028D0	411.00	414.00	1.24	2.62	0.06	0.15	4.07	0.09	0.25
FT SFB VAR 7	WE147D1	472.00	483.00	1.35	2.93	0.07	0.24	4.59	0.20	0.34
FT SFB VAR 8	WB204D1	274.50	285.00	1.96	3.40	0.06	0.28	5.70	0.16	0.28
FT SFB VAR 9	WB202D0	333.96	336.48	0.90	1.85	0.07	0.16	2.99	0.12	0.24

13.3.2.5 F-North

A total of 13 different ¾ NQ drill core samples were used for testing of the F-North material comminution and flotation characteristics. The samples selected from the F-North material ranged from 1.46 to 5.62 g/t 4E. Refer to Table 13-13 and Table 13-14 for a summary of the comminution and flotation samples, respectively.

Table 13-13: Summary of F-North Comminution Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
COM SFN VAR 1	WE119 D1	309.75	343.35	1.11	2.98	0.07	0.22	4.37	0.18	0.30
COM SFN VAR 2	WE120 D1	396.50	444.74	1.09	2.74	0.06	0.21	4.11	0.16	0.25
COM SFN VAR 3	WE125 D1	314.00	361.10	0.44	0.94	0.02	0.06	1.46	0.05	0.17
COM SFN VAR 4	WE128 D0	348.00	355.96	1.04	2.79	0.06	0.18	4.07	0.20	0.34
COM SFN VAR 5	WE129 D0	280.17	307.21	1.84	3.47	0.09	0.22	5.62	0.12	0.23

Table 13-14: Summary of F-North Flotation Samples

Sample No.	Drill Hole ID	From (m)	To (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)
FT SFN VAR 1	WE099D0	236.50	283.00	0.91	1.87	0.05	0.24	3.06	0.08	0.19
FT SFN VAR 2	WE117D0	249.00	296.00	0.63	1.63	0.04	0.13	2.42	0.10	0.22
FT SFN VAR 3	WE118D0	389.00	424.50	0.89	2.23	0.05	0.16	3.34	0.12	0.21
FT SFN VAR 4	WE119D0	308.60	342.96	0.98	2.44	0.06	0.21	3.68	0.16	0.27
FT SFN VAR 5	WE129D1	279.33	307.45	1.66	3.61	0.08	0.24	5.59	0.14	0.27
FT SFN VAR 6	WE122D0	378.00	403.50	1.08	2.41	0.06	0.19	3.75	ND	ND
FT SFN VAR 7	WE121D0	451.13	459.96	1.00	2.53	0.06	0.19	3.78	0.13	0.19
FT SFN VAR 8	WE124D0	188.50	193.74	1.05	1.96	0.05	0.15	3.21	0.09	0.22
FT SFN VAR 9	WE135D0	211.20	226.60	0.84	2.11	0.05	0.14	3.14	0.09	0.22

13.3.3 Comminution Variability Testwork

Variability testwork on the Waterberg material comminution parameters were conducted on a total of 28 samples across the various lithology units / mining areas. Refer to DFS metallurgical testwork, Table 13-4, Table 13-6, Table 13-8, Table 13-10, and Table 13-12 for more details on the samples tested.

Refer to Table 13-15 for a summary of the testwork conducted as part of the DFS.

Table 13-15: Summary of Comminution Variability Results

Sample ID	Drop Weight Index (DWi)	Mia	Mih	Mic	SG	ta	A*b	Ai	BBWi
	kWh/m^3	kWh/t	kWh/t	kWh/t	t/m^3	-	-	g	kWh/t
COM TZ VAR1	4.04	12.10	8.10	4.20	2.89	0.64	71.50	0.16	19.50
COM TZ VAR2	3.88	11.80	7.90	4.10	2.87	0.67	73.80	0.18	18.40
COM TZ VAR3	4.72	13.20	9.10	4.70	3.00	0.55	63.70	0.17	20.10
COM TZ VAR4	4.20	12.40	8.30	4.30	2.92	0.62	69.70	0.19	18.30
COM TZ VAR5	5.00	14.80	10.30	5.30	2.81	0.52	56.20	0.16	19.10
COM TZ 85th Percentile	**4.83**	**13.84**	**9.58**	**4.94**	**2.95**	**0.65**	**60.70**	**0.18**	**19.74**
COM SFN VAR1	6.04	16.30	11.80	6.10	2.96	0.43	49.40	0.09	23.50
COM SFN VAR2	6.80	17.30	12.80	6.60	3.08	0.38	45.40	0.18	21.30
COM SFN VAR3	7.10	18.60	13.90	7.20	2.95	0.36	41.50	0.03	23.90
COM SFN VAR4	5.44	15.60	11.00	5.70	2.86	0.47	52.30	0.03	22.10
COM SFN VAR5	5.65	15.70	11.20	5.80	2.93	0.46	51.90	0.07	20.50
COM SFN 85th Percentile	**6.92**	**17.82**	**13.24**	**6.84**	**3.01**	**0.46**	**46.80**	**0.13**	**23.66**
COM SFB VAR1	8.03	19.60	15.00	7.80	3.10	0.32	38.50	0.18	23.60
COM SFB VAR2	7.62	20.40	15.50	8.00	2.86	0.34	37.60	0.13	21.00
COM SFB VAR3	6.42	16.50	12.10	6.30	3.07	0.40	47.60	0.21	23.20
COM SFB VAR4	6.47	17.10	12.50	6.50	2.99	0.40	46.20	0.08	22.50
COM SFB 85th Percentile	**7.85**	**20.04**	**15.28**	**7.91**	**3.09**	**0.40**	**39.55**	**0.20**	**23.42**
COM SF VAR1	6.28	17.00	12.40	6.40	2.93	0.41	46.70	0.09	22.10
COM SF VAR2	8.12	20.20	15.50	8.00	3.03	0.32	37.20	0.12	22.60
COM SF VAR3	6.33	16.90	12.40	6.40	2.96	0.41	46.60	0.08	21.30
COM SF VAR4	6.33	16.90	12.40	6.40	2.97	0.41	47.20	0.13	24.30
COM SF 85th Percentile	**7.31**	**18.76**	**14.11**	**7.28**	**3.00**	**0.41**	**37.92**	**0.13**	**23.54**
COM SFC VAR1	9.98	24.50	19.60	10.10	2.95	0.26	29.40	0.07	23.00
COM SFC VAR2	10.04	24.90	19.90	10.30	2.92	0.26	29.00	0.26	19.90
COM SFC VAR3	10.10	25.20	20.20	10.50	2.90	0.26	29.00	0.03	23.90
COM SFC VAR4	6.92	19.10	14.20	7.30	2.83	0.37	40.90	0.08	25.50
COM SFC VAR5	7.51	19.20	14.50	7.50	3.00	0.35	40.00	0.15	23.50
COM SFC VAR6	10.02	25.30	20.20	10.50	2.87	0.26	29.00	0.06	26.10
COM SFC VAR7	7.35	19.20	14.40	7.50	2.95	0.35	39.80	0.06	21.40
COM SFC VAR8	11.20	26.00	21.30	11.00	3.05	0.23	27.00	0.16	24.40
COM SFC VAR9	10.30	24.70	19.90	10.30	3.00	0.25	29.00	0.10	22.40
COM SFC VAR10	8.13	20.50	15.80	8.20	2.99	0.32	36.50	0.12	23.70
COM SFC 85th Percentile	**10.23**	**25.27**	**20.20**	**10.50**	**3.00**	**0.35**	**29.00**	**0.16**	**25.12**

13.3.3.1 Drop Weight Index

The SMC test generates a DWi which is a measure of the rock strength when broken under impact. The DWi is directly related to the JK impact breakage parameters A and b as shown in Table 13-15, which are used in the JK SAG mill models to predict throughput, power draw and product size distribution.

Figure 13-4 summarises the DWi data for each ore zone. It is noted that the T-South material has a significantly lower DWi value compared to the F-Central material. The F-Central samples reported the highest variability and spread of DWi data, ranging from 7.5 kWh/m^3 to 11.2 kWh/m^3.

Figure 13-4: Drop Weight Index Summary for Waterberg Ore Zones

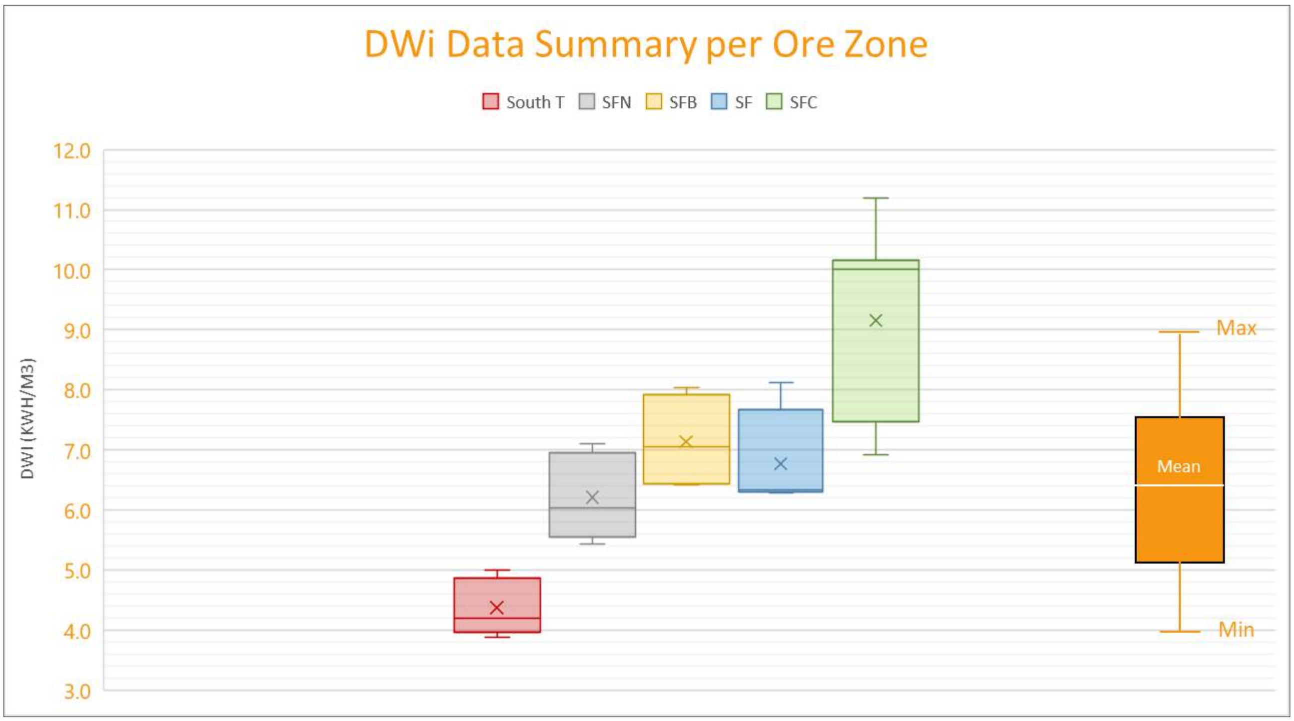


Table 13-16 gives an indication of the Axb parameter classification.

Table 13-16: Classification of Axb Parameter

Axb Range	127 +	67 - 127	56 - 67	43 - 56	39 - 43	30 - 39	0 - 30
Classification	Very Soft	Soft	Moderately Soft	Medium	Moderately Hard	Hard	Very Hard

Based on the classification in Table 13-16, the Waterberg ores can be classified as listed below.

- T-South samples are moderately soft to soft with Axb values ranging from 56.2 to 73.8. These samples are slightly softer compared to PFS composite sample which reported an Axb value of 51.6.
- F-North samples are moderately hard to medium with Axb values ranging from 41.5 to 52.3.

- F-Boundary samples are moderately hard to medium with Axb values ranging from 37.6 to 47.6.
- F-South samples are moderately hard to medium with Axb values ranging from 37.2 to 47.2.
- F-Central samples are moderately hard to very hard with Axb values ranging from 27.0 to 40.9. The average of the samples tested are slightly harder compared to the PFS composite samples, which reported Axb values of 30.8 and 32.1.

Figure 13-5 presents the Ai summary for the Waterberg ore zones.

Figure 13-5: Abrasion Index Summary for Waterberg Ore Zones

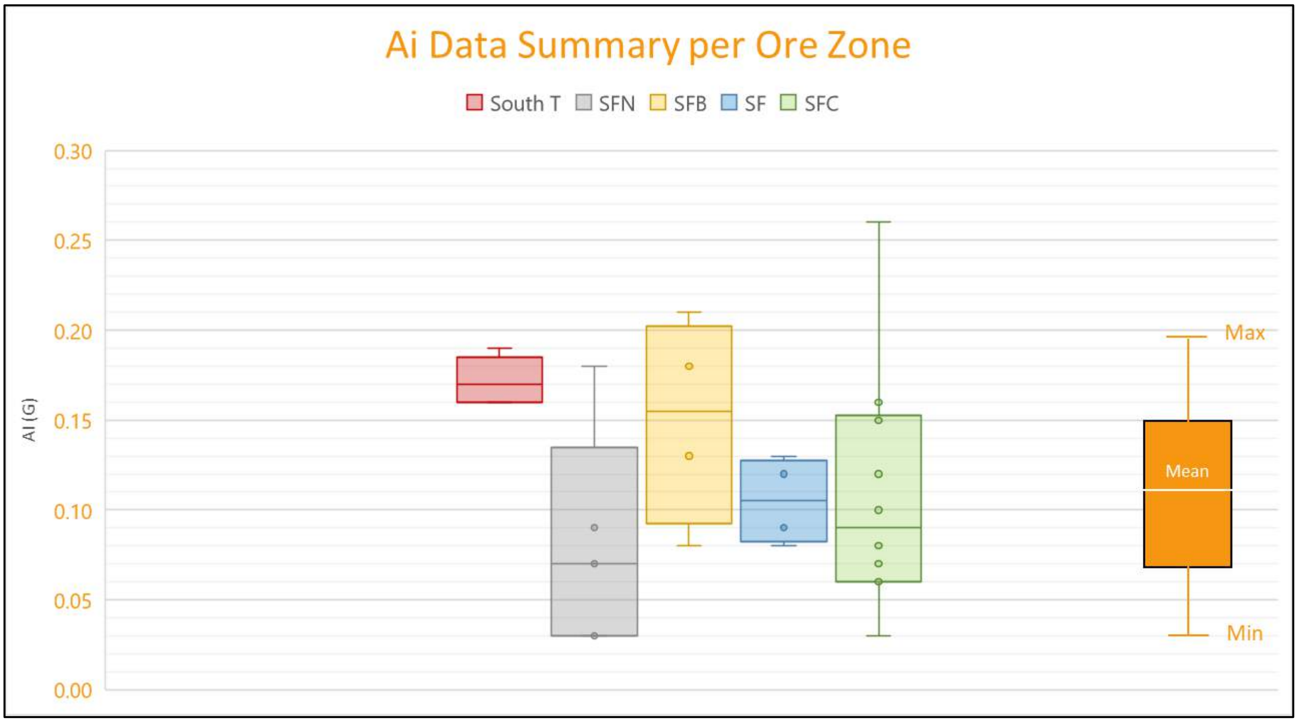


Table 13-17 gives an indication of the Ai parameter classification.

Table 13-17: Classification of Bond Abrasion Index

Ai Range	<0.2	0.2 - 0.5	0.5 - 0.75	0.75 - 1	>1
Classification	Low	Medium	Abrasive	Very Abrasive	Extremely Abrasive

Based on the above classification, the Waterberg ores can be classified as listed below.

- T-South samples presents a low abrasiveness with Ai values ranging from 0.16 g to 0.19 g, compared to the PFS composite sample which reported an Ai value of 0.19 g.
- F-North samples presents a low abrasiveness with Ai values ranging from 0.03 g to 0.18 g.
- F-Boundary samples presents a low to medium abrasiveness with Ai values ranging from 0.08 g to 0.21 g, compared to the PFS composite sample which reported an Ai value of 0.20 g.

- F-South samples presents a low abrasiveness with Ai values ranging from 0.08 g to 0.13 g.
- F-Central samples presents a low abrasiveness with Ai values ranging from 0.03 g to 0.26 g. The average value of the samples tested (0.11 g) are less abrasive compared to the PFS composite samples which reported Ai values of 0.16 g and 0.18 g.

13.3.3.2 Bond Ball Work Index

Figure 13-6 summarises the BBWi data (at a 106 µm closing screen) for each ore zone, while Table 13-18 gives an indication of the BBWi classification. Figure 13-6 confirms the variability of the work index for the Waterberg ores, specifically the F-Central material.

Figure 13-6: Bond Ball Work Index Summary for Waterberg Ore Zones

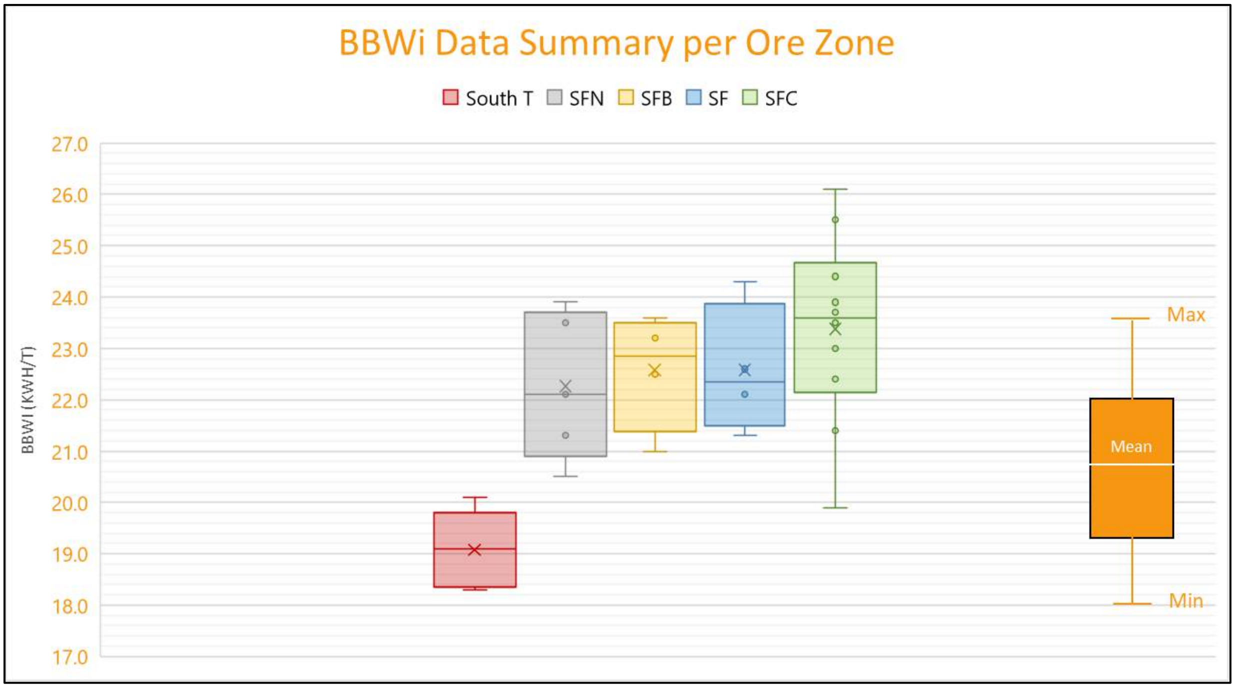


Table 13-18: Classification of Bond Work Index

BBWi (kWh/t)	7-9	10-14	15 - 20	> 20
Classification	Soft	Medium	Hard	Very Hard

Based on the above classification, the Waterberg ores can be classified as listed below.

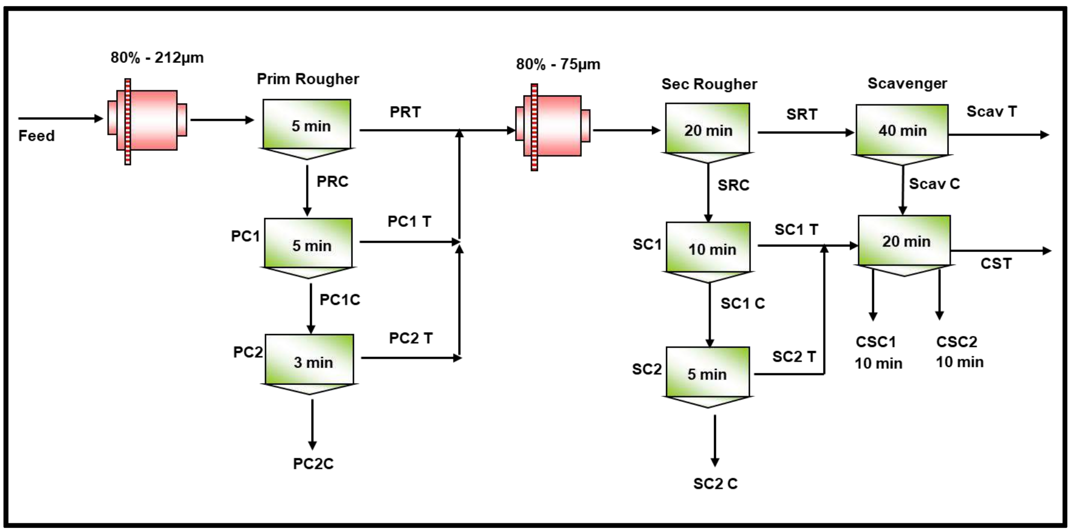
- T-South samples are hard with BBWi values ranging from 18.3 kWh/t to 20.1 kWh/t. These samples compare well to the PFS composite sample, which reported a BBWi value of 19.5 kWh/t.
- F-North samples are very hard with BBWi values ranging from 20.5 kWh/t to 23.9 kWh/t. These samples are harder compared to the PFS composite sample which reported a BBWi value of 20.2 kWh/t.

- F-Boundary samples are very hard with BBWi values ranging from 21.0 kWh/t to 23.6 kWh/t. These samples compare well to the PFS composite sample which reported a BBWi value of 22.7 kWh/t.
- F-South samples are very hard with BBWi values ranging from 21.3 kWh/t to 24.3 kWh/t.
- F-Central samples are very hard with BBWi values ranging from 19.9 kWh/t to 26.1 kWh/t. The average of the samples tested are slightly harder compared to the PFS composite samples, which reported BBWi values of 24.4 kWh/t and 22.0 kWh/t.

13.3.4 Flotation Variability Testwork

The DFS flotation testwork campaign included open circuit bench scale flotation testing using individual drill core samples, as per Section 13.3.2, and subjecting them to the flotation flowsheet as developed during the PFS campaign. The open circuit variability flowsheet is presented in Figure 13-7.

Figure 13-7: Open Circuit Variability Testing Flowsheet



13.3.4.1 Flotation Variability Sample Assays

The measured head grades of the variability samples used in the flotation testwork are summarised in Table 13-19.

13.3.4.2 Summary of Flotation Variability Results

A summary of the recorded concentrate grades and associated recoveries are presented in Table 13-20.

Table 13-19: Flotation Variability Samples Measured Head Assays

Sample Ref	Drill Hole ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
T-South									
FT TZ VAR 1	WB228D1	0.43	0.72	0.05	0.14	1.29	0.59	0.05	0.03
FT TZ VAR 2	WB217D1	1.13	0.49	0.03	1.63	3.27	0.83	0.39	0.14
FT TZ VAR 3	WB226D0	1.47	3.21	0.03	0.49	5.20	0.17	0.09	0.04
FT TZ VAR 4	WB219D2	1.02	0.78	0.01	1.69	3.50	1.03	0.36	0.13
FT TZ VAR 5	WB229D0	1.08	1.81	0.06	0.59	3.53	0.16	0.06	0.03
FT TZ VAR 6	WB222D0	1.08	2.65	0.02	0.57	4.31	0.25	0.13	0.07
FT TZ VAR 6	WB222D0	1.08	2.65	0.02	0.57	4.31	0.25	0.13	0.07
FT TZ VAR 7	WB215D2	1.43	2.21	0.03	1.37	5.03	0.26	0.18	0.06
FT TZ VAR 8	WB220D0	0.43	0.22	0.01	0.75	1.41	0.21	0.13	0.06
FT TZ VAR 9	WB233D2	1.15	2.56	0.02	0.96	4.70	0.22	0.11	0.05
F-South									
FT SF VAR 1	WB131D1	0.36	0.50	0.04	0.03	0.92	<0.005	0.01	0.10
FT SF VAR 2	WB156D0	1.32	2.86	0.10	0.21	4.48	<0.005	0.04	0.12
FT SF VAR 3	WB026D0	1.435	2.79	0.10	0.22	4.54	0.22	0.07	0.13
FT SF VAR 4	WB096D3	3.49	5.62	0.26	0.20	9.57	<0.005	0.03	0.19
FT SF VAR 5	WB013D0	2.12	3.78	0.13	0.29	6.32	0.34	0.08	0.17
F-Central									
SFC FT VAR 1	WB271D0	0.75	1.80	0.06	0.13	2.73	0.41	0.12	0.25
SFC FT VAR 2	WB114D0	0.82	1.88	0.06	0.16	2.92	0.69	0.14	0.31
SFC FT VAR 3	WB277D0	0.82	2.09	0.06	0.15	3.11	0.26	0.06	0.21
SFC FT VAR 4	WB113D1	0.87	2.26	0.04	0.18	3.34	0.37	0.08	0.21
SFC FT VAR 5	WB259D0	0.68	1.79	0.06	0.14	2.67	0.01	0.06	0.15
SFC FT VAR 6	WB118D0	0.89	2.21	0.06	0.12	3.27	<0.005	0.07	0.21
SFC FT VAR 7	WB263D1	0.90	2.54	0.08	0.17	3.68	0.40	0.10	0.22
SFC FT VAR 8	WB090D0	0.90	2.24	0.06	0.14	3.34	0.03	0.07	0.20
SFC FT VAR 9	WB091D1	0.92	2.55	0.07	0.15	3.69	0.46	0.09	0.17
SFC FT VAR 11	WB206D1	0.66	1.05	0.06	0.09	1.86	<0.005	0.03	0.12
SFC FT VAR 12	WB087D0	0.92	2.30	0.08	0.15	3.45	<0.005	0.05	0.21
SFC FT VAR 10	WB150D1	1.17	3.21	0.10	0.20	4.68	0.75	0.13	0.24
SFC FT VAR 14	WB260D0	1.32	2.89	0.11	0.15	4.45	0.01	0.06	0.19
SFC FT VAR 15	WB095D2	1.23	2.85	0.08	0.22	4.37	0.36	0.10	0.19
SFC FT VAR 13	WB264D0	1.36	3.52	0.10	0.24	5.22	0.40	0.12	0.28
SFC FT VAR 16	WB046D1	1.61	4.12	0.13	0.23	6.08	0.36	0.10	0.24
SFC FT VAR 17	WB087D2	1.54	4.14	0.12	0.49	6.28	0.01	0.13	0.31
SFC FT VAR 18	WB270D0	1.54	4.89	0.17	0.31	6.90	0.71	0.16	0.27
SFC FT VAR 19	WB085D1	2.95	5.77	0.21	0.30	9.21	0.41	0.09	0.24
F-Boundary									
SFB FT VAR 1	WB053D2	0.98	2.31	0.08	0.21	3.57	0.67	0.15	0.28
SFB FT VAR 2	WB154D0	0.99	2.65	0.08	0.35	4.07	0.50	0.12	0.25
SFB FT VAR 3	WE030D1	1.04	2.20	0.10	0.10	3.43	0.49	0.08	0.24
SFB FT VAR 4	WE083D1	0.75	1.67	0.05	0.14	2.60	0.01	0.09	0.20
SFB FT VAR 5	WE028D0	1.21	3.50	0.08	0.22	5.01	0.52	0.12	0.31
SFB FT VAR 6	WB079D1	0.73	1.67	0.06	0.12	2.57	0.10	0.05	0.21
SFB FT VAR 7	WE147D1	1.38	3.25	0.10	0.21	4.93	0.67	0.19	0.33
F-North									
SFN FT VAR 1	WE099D0	1.07	2.47	0.08	0.22	3.84	0.62	0.11	0.24
SFN FT VAR 2	WE117D0	0.90	2.81	0.09	0.18	4.13	0.71	0.17	0.28
SFN FT VAR 3	WE118D0	0.93	2.53	0.07	0.18	3.70	0.51	0.13	0.24
SFN FT VAR 4	WE119D0	1.10	2.68	0.07	0.19	4.36	0.89	0.16	0.27
SFN FT VAR 5	WE129D1	2.24	3.99	0.13	0.28	6.65	0.88	0.14	0.30
SFN FT VAR 6	WE122D0	1.08	2.64	0.08	0.17	3.97	0.67	0.13	0.26
SFN FT VAR 7	WE121D0	1.04	2.66	0.07	0.18	3.95	0.74	0.13	0.19
SFN FT VAR 8	WE124D0	0.97	1.69	0.08	0.12	2.85	0.54	0.08	0.20
SFN FT VAR 9	WE135D0	0.77	2.48	0.07	0.16	3.48	0.46	0.10	0.22

Table 13-20: Flotation Variability Testing Results Summary

Sample Ref	Drill Hole ID	Grind	Mass Pull (%)	Product Grade							Recovery						
		% -75µm		Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)	Pt (%)	Pd (%)	Rh (%)	Au (%)	4E (%)	Cu (%)	Ni (%)
FT TZ VAR 1	WB228D1	Test not conducted due to sample grade being below cutoff.															
FT TZ VAR 2	WB217D1	86.3	4.4	14.8	4.2	0.0	13.9	33.0	8.1	2.6	69.5	65.4	17.5	41.1	53.3	91.3	66.6
FT TZ VAR 3	WB226D0	95.5	3.4	38.6	77.0	0.2	13.1	128.9	1.9	NR	94.0	93.3	22.3	90.8	92.8	93.1	NR
FT TZ VAR 4	WB219D2	90.3	4.4	22.4	12.6	0.1	18.7	53.8	6.7	2.2	83.7	75.8	18.0	48.8	65.4	90.6	63.5
FT TZ VAR 5	WB229D0	89.8	4.7	20.4	34.2	0.6	10.9	66.1	1.2	NR	90.9	90.6	48.0	80.1	88.2	90.1	NR
FT TZ VAR 6	WB222D0	89.2	4.0	24.3	54.9	0.1	10.9	90.1	NR	NR	88.8	84.3	30.8	73.8	83.8	NR	NR
FT TZ VAR 6	WB222D0	80.0[4]	3.9	25.3	55.8	0.0	12.5	93.6	NR	NR	86.0	82.2	7.1	72.6	81.5	NR	NR
FT TZ VAR 7	WB215D2	90.1	4.0	30.4	45.9	0.1	26.4	102.8	4.3	NR	90.7	88.0	11.6	81.1	86.3	95.6	NR
FT TZ VAR 8	WB220D0	92.9	4.0	24.3	54.9	0.1	10.9	90.1	NR	NR	88.8	84.3	30.8	73.8	83.8	NR	NR
FT TZ VAR 9	WB233D2	91.5	2.6	43.7	75.2	0.2	29.7	148.8	4.0	NR	91.6	89.4	16.8	88.0	89.2	94.6	NR
FT SF VAR 1	WB131D1	81.5	2.9	9.1	13.1	0.1	0.5	22.8	0.4	0.4	73.3	76.2	11.0	53.1	72.0	72.9	11.5
FT SF VAR 2	WB156D0	77.2	1.7	51.5	111.5	3.5	9.3	175.8	2.3	2.7	68.7	70.8	64.8	65.0	69.7	84.1	30.6
FT SF VAR 3	WB026D0	74.2	2.8	40.4	73.0	1.8	6.1	121.3	2.1	2.2	78.8	80.3	66.7	69.7	79.0	87.4	44.8
FT SF VAR 4	WB096D3	78.6	8.2	40.6	52.4	2.4	1.6	97.0	0.3	0.9	89.3	81.2	68.4	64.4	83.7	87.5	40.4
FT SF VAR 5	WB013D0	71.0	3.0	55.7	122.5	2.2	8.1	188.4	2.2	2.2	78.7	88.2	55.9	75.3	84.0	87.4	38.2
SFC FT VAR 1	WB271D0	87.7	Not submitted for assaying due to too high mass pull														
SFC FT VAR 2	WB114D0	72.8	3.0	19.0	46.2	1.2	4.9	71.4	NR	NR	72.6	77.8	60.9	76.4	75.9	NR	NR
SFC FT VAR 3	WB277D0	69.5	2.7	27.8	74.3	1.4	4.6	108.1	NR	NR	83.2	88.9	75.0	76.6	86.5	NR	NR
SFC FT VAR 4	WB113D1	77.2	3.0	19.6	56.7	1.3	3.3	80.9	1.9	2.7	69.6	81.8	65.5	72.6	77.8	81.6	38.6
SFC FT VAR 5	WB259D0	94.3	Not reported due to poor test accountability														
SFC FT VAR 6	WB118D0	94.2	4.1	16.9	42.9	1.0	2.4	63.2	NR	NR	82.0	82.4	66.2	80.1	81.87	NR	NR
SFC FT VAR 7	WB263D1	90.4	3.1	24.2	60.7	1.5	3.4	89.9	2.2	3.1	67.7	79.0	73.3	69.2	75.1	77.6	41.3
SFC FT VAR 8	WB090D0	96.1	3.2	19.4	39.5	0.8	2.3	62.0	1.6	1.9	65.8	58.7	45.2	50.5	60.1	80.3	30.1
SFC FT VAR 9	WB091D1	88.0	Not reported due to poor test accountability														
SFC FT VAR	WB206D1	89.4	1.9	20.1	42.3	1.3	3.2	67.0	1.2	1.5	67.2	75.1	42.9	71.5	71.4	79.9	20.8
SFC FT VAR	WB087D0	81.4	2.2	44.1	104.5	2.4	5.2	156.2	NR	NR	85.0	86.4	69.3	78.2	85.4	NR	NR
SFC FT VAR	WB150D1	83.8	4.9	16.5	44.7	1.1	3.0	65.3	2.3	2.8	71.0	77.3	54.2	77.4	75.1	91.2	55.5
SFC FT VAR	WB260D0	73.4	4.4	23.1	44.7	1.5	2.3	71.5	1.4	2.2	82.2	76.1	73.8	65.1	77.5	86.7	43.3
SFC FT VAR	WB095D2	78.3	3.2	31.1	66.5	1.4	3.7	102.7	NR	NR	87.1	81.7	66.3	68.3	82.4	NR	NR
SFC FT VAR	WB264D0	90.1	2.6	38.8	107.2	2.7	5.6	154.2	3.3	4.0	67.7	85.2	80.1	80.1	79.7	74.9	37.8
SFC FT VAR	WB046D1	77.8	3.3	37.5	107.3	3.2	5.4	153.4	2.4	3.7	75.7	87.2	84.1	75.7	83.6	82.3	46.8
SFC FT VAR	WB087D2	87.3	4.3	31.1	79.9	1.9	8.3	121.2	2.2	3.1	81.1	82.5	71.2	81.0	81.8	87.3	47.1
SFC FT VAR	WB270D0	94.0	3.6	44.2	127.0	3.3	6.6	181.1	4.1	5.2	85.5	89.5	81.6	82.1	88.1	89.9	68.1
SFC FT VAR	WB085D1	89.1	3.3	71.3	147.6	5.1	7.0	231.0	2.4	4.0	78.8	83.4	81.7	77.0	81.7	80.3	44.8
SFB FT VAR 1	WB053D2	74.8	2.3	27.3	69.3	1.8	7.2	105.7	4.7	5.4	67.4	81.5	57.1	71.1	76.1	86.9	47.9

[4] Milling time reduced for test to reach target grind of 80% -75µm

Sample Ref	Drill Hole ID	Grind	Mass Pull (%)	Product Grade							Recovery						
		% -75µm		Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	Cu (%)	Ni (%)	Pt (%)	Pd (%)	Rh (%)	Au (%)	4E (%)	Cu (%)	Ni (%)
SFB FT VAR 2	WB154D0	83.7	2.5	27.8	79.4	1.5	6.1	114.8	3.8	5.3	69.2	84.9	48.6	66.8	78.7	89.0	53.9
SFB FT VAR 3	WE030D1	90.3	3.1	23.6	50.0	1.2	3.8	78.5	2.2	4.2	68.5	72.6	41.6	79.8	70.8	87.8	50.4
SFB FT VAR 4	WE083D1	80.9	3.2	12.0	35.2	0.7	2.7	50.6	2.5	3.3	67.1	77.9	43.6	84.7	74.6	90.7	51.6
SFB FT VAR 5	WE028D0	77.7	4.6	18.3	47.1	0.7	3.7	69.8	1.7	2.5	74.7	73.4	46.7	74.6	73.4	78.9	39.9
SFB FT VAR 6	WB079D1	68.2	1.9	11.5	28.9	53.8	1.8	96.0	NR	2.1	80.8	73.6	72.7	61.9	73.6	NR	20.7
SFB FT VAR 7	WE147D1	75.4	4.7	23.0	53.4	1.1	3.9	81.4	3.4	4.3	75.0	81.4	56.8	74.2	78.7	81.4	59.5
SFN FT VAR 1	WE099D0	89.7	3.5	28.4	49.6	1.4	5.0	84.3	2.8	4.5	87.2	82.2	60.1	76.9	83.0	88.0	61.0
SFN FT VAR 2	WE117D0	86.9	3.4	21.1	58.7	1.4	5.3	86.6	3.8	4.8	77.6	81.2	55.5	78.2	79.5	85.8	58.3
SFN FT VAR 3	WE118D0	80.3	3.4	21.4	53.9	1.3	4.7	81.4	3.1	3.9	76.7	80.1	63.1	76.2	78.6	87.7	54.7
SFN FT VAR 4	WE119D0	85.4	3.4	17.2	53.1	0.01	4.4	74.7	4.0	4.8	64.7	73.0	1.6	67.1	70.2	82.0	56.2
SFN FT VAR 5	WE129D1	83.0	4.1	30.3	66.7	1.6	4.7	103.3	2.8	3.8	74.5	76.5	60.2	73.8	75.5	85.6	51.3
SFN FT VAR 6	WE122D0	81.4	2.5	38.0	78.1	1.3	4.8	122.2	4.2	6.0	78.7	80.0	48.8	67.9	78.5	85.7	53.1
SFN FT VAR 7	WE121D0	84.6	4.2	15.6	47.2	0.8	2.7	66.2	2.4	3.0	68.9	83.8	55.5	60.0	78.1	86.1	66.7
SFN FT VAR 8	WE124D0	93.3	Not reported due to poor test accountability														
SFN FT VAR 9	WE135D0	87.5	3.6	15.6	38.4	1.0	2.6	57.5	2.3	3.3	70.4	72.5	55.9	67.5	71.3	84.8	48.8

Figure 13-8 and Figure 13-9 present the 4E head grade – recovery curves, and 4E head grade – concentrate grade curves, respectively. The anticipated range of mill feed grades is shaded for reference.

Figure 13-8: Open Circuit Variability 4E Head Grade-Recovery Curves

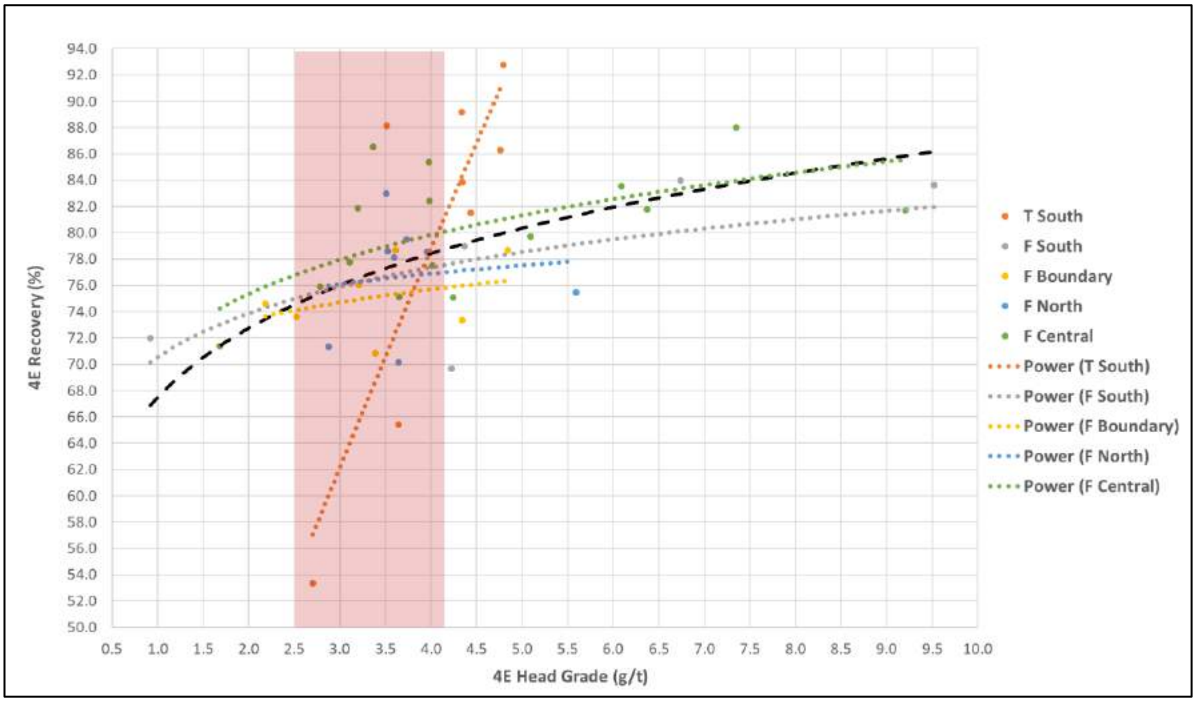
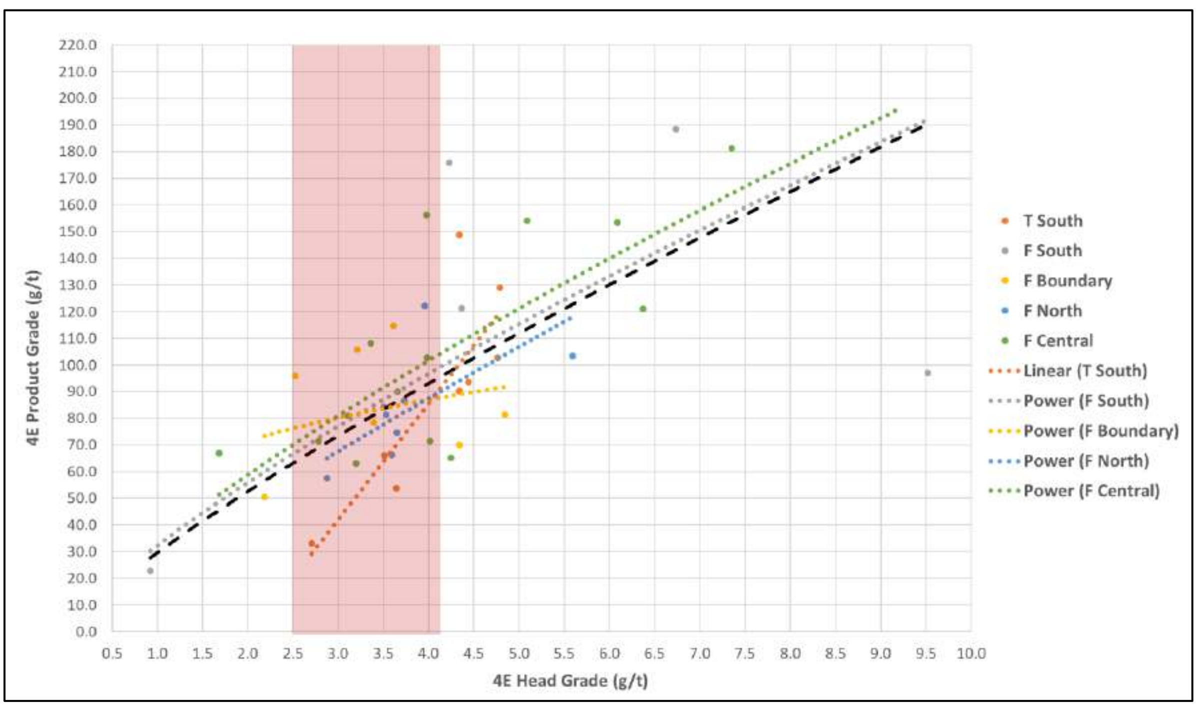


Figure 13-9: Open Circuit Variability 4E Head Grade-Concentrate Grade Curves



The following items were noted.

- An increase in 4E recovery and concentrate grade with increasing head grade was noted across each of the lithology units.
- The grinding times were kept constant for each lithology unit, based on the grinding times measured in the PFS for each composite sample, resulting in the variance in secondary grinds.
- In general, the secondary grind for the T-South samples were finer than the target grind of 80% passing 75 µm, resulting in higher PGE and Ni recoveries compared to the PFS testwork.
- It appears that finer grinds on the F Zone materials resulted in a reduction in recoveries.
- The F-North material presented an inferior flotation response when considering product grade and associated recovery.

Summaries of the Cu and Ni head grade-recovery curves are presented in Figure 13-10 and Figure 13-11, respectively.

Figure 13-10: Open Circuit Variability Copper Head Grade-Recovery Curves

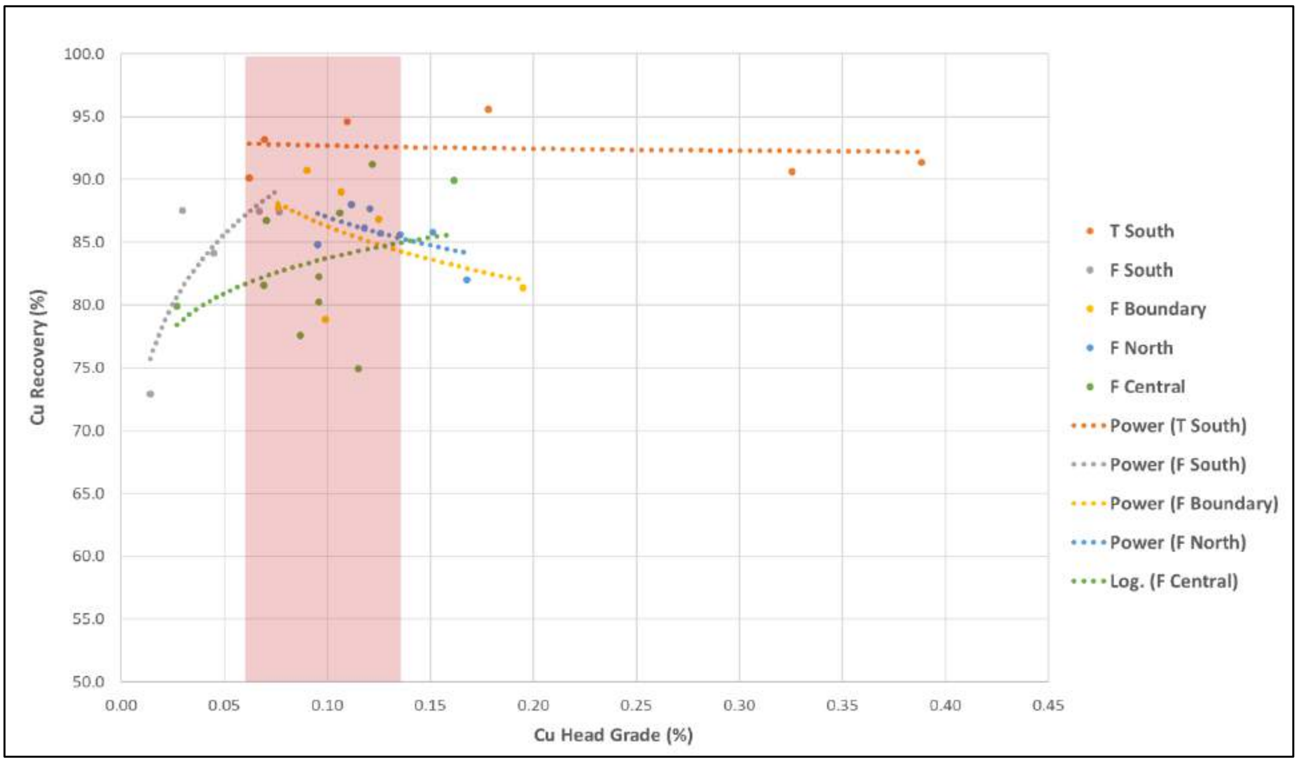
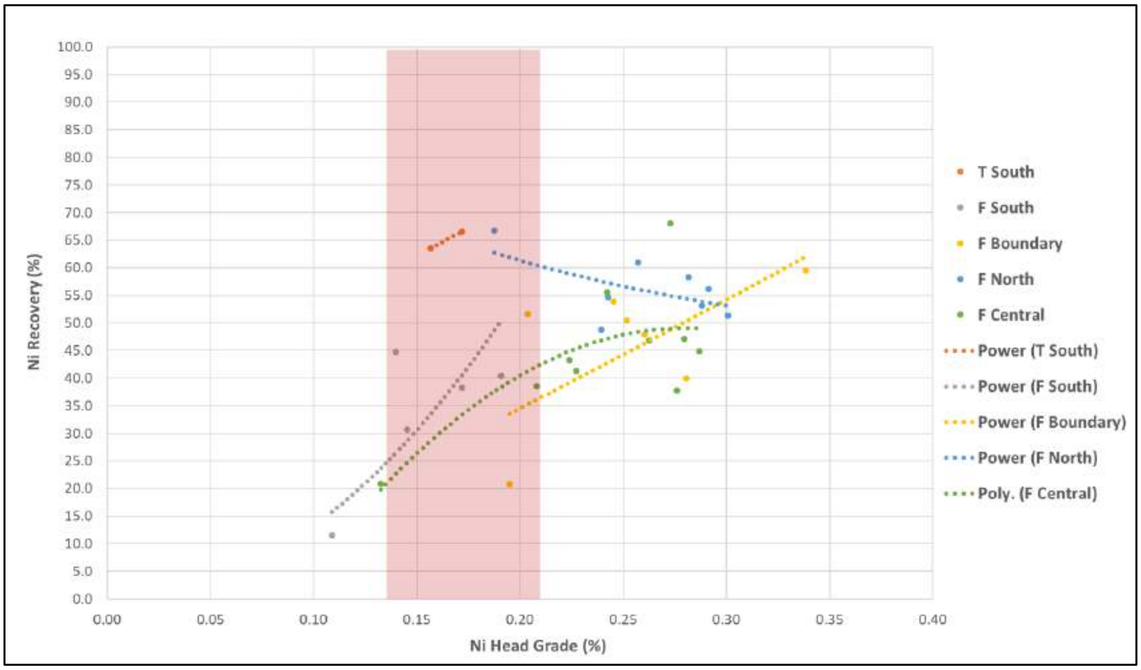


Figure 13-11: Open Circuit Variability Nickel Head Grade-Recovery Curves



When considering the Cu and Ni recoveries, the following items were noted.

- The fine grind on the T-South samples resulted in high base metal recoveries.
- The F-North samples reported superior Ni recoveries, compared to other lithology units at similar head grades.
- Base metal recoveries are sensitive to grind.

13.3.5 Mine Blend Flotation Testwork

Once the variability testing was completed, focus was placed on the flotation response of likely Mine Blends. The following blends were tested.

- Mine Blend 1: Mine Blend 1: 15% T-South: 40% F-Central: 25% F-North: 20% F-Boundary
- Mine Blend 4: 20% T-South: 35% F-Central: 20% F-North: 25% F-Boundary
- Mine Blend 5: 50% T-South: 50% F-Central
- Mine Blend 6: 30% T-South: 70% F-Central

13.3.5.1 Mine Blend 1

Mine Blend 1 was produced from a composite of the following drill holes: WB228D1, WB219D2, WB229D0, WB222D0, WB222D0, WB215D2, WB220D0, WB233D2, WB233D2, WB271D0, WB114D0, WB259D0, WB118D0, WB263D1, WB090D0, WB206D1, WB260D0, WE099D0, WE135D0, WB154D0 and WE030D1. The individual masses of each of the drill holes were based on sample availability and grade to get the resulting blend within the expected grade. Refer to Table 13-21 for a summary of the measured head grade of Mine Blend 1.

Table 13-21: Mine Blend 1 Sample Head Assays

Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
0.81	2.20	0.05	0.30	3.36	0.34	0.06	0.18

The Mine Blend 1 sample was subjected to the flowsheet as developed in the PFS in open circuit mode and achieved a 4E recovery of 75.5% at a final product grade of 95 g/t 4E (3.4% mass pull) as shown in Figure 13-7. It is noted that the test accountability was not within acceptable limits and that the back calculated head grade was higher than measured at 4.2 g/t 4E (i.e. final product grade and recovery is possibly overstated).

13.3.5.2 Mine Blend 4

Mine Blend 4 was produced from a composite of the following drill holes: WB228D1, WB219D2, WB229D0, WB222D0, WB222D0, WB215D2, WB220D0, WB233D2, WB233D2, WB271D0, WB114D0, WB259D0, WB118D0, WB263D1, WB090D0, WB206D1, WB260D0, WE119D0, WE122D0, WE124D0, WB154D0 and WE030D1. The individual masses of each of the drill holes were based on sample availability and grade to get the resulting blend within the expected grade. Refer to Table 13-22 for a summary of the measured head grade of Mine Blend 4.

Table 13-22: Mine Blend 4 Sample Head Assays

Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
1.00	2.29	0.03	0.65	3.97	0.42	0.11	0.17

The sample was subjected to the flowsheet as developed in the PFS in open circuit mode and achieved a 4E recovery of 77.5% at a final product grade of 82 g/t 4E (3.5% mass pull) as shown in Figure 13-7. It is noted that the test accountability was not within acceptable limits and that the back calculated head grade was lower than measured (i.e. final product grade and recovery is possibly understated).

13.3.5.3 Mine Blend 5

Mine Blend 5 was produced from a composite of the following drill holes: WB228D1, WB219D2, WB229D0, WB222D0, WB222D0, WB215D2, WB220D0, WB233D2, WB233D2, WB271D0, WB114D0, WB259D0, WB118D0, WB263D1, WB090D0, WB206D1 and WB260D0. The individual masses of each of the drill holes were based on sample availability and grade, to get the resulting blend within the expected grade. Refer to Table 13-23 for a summary of the measured head grade of Mine Blend 5.

Table 13-23: Mine Blend 5 Sample Head Assays

Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
0.91	2.47	0.02	0.54	3.94	0.35	0.12	0.14

The sample was subjected to the flowsheet as developed in the PFS and achieved a 4E recovery of 81.2% at a final product grade of 112 g/t 4E (3.2% mass pull), in open circuit mode as shown in Figure 13-7.

13.3.5.4 Mine Blend 6

Mine Blend 6 was produced from a composite of the following drill holes: WB228D1, WB219D2, WB229D0, WB222D0, WB222D0, WB215D2, WB220D0, WB233D2, WB233D2, WB271D0, WB114D0, WB259D0, WB118D0, WB263D1, WB090D0, WB206D1 and WB260D0. The individual masses of each of the drill holes were based on sample availability and grade, to get the resulting blend within the expected grade. Refer to Table 13-24 for a summary of the measured head grade of Mine Blend 6.

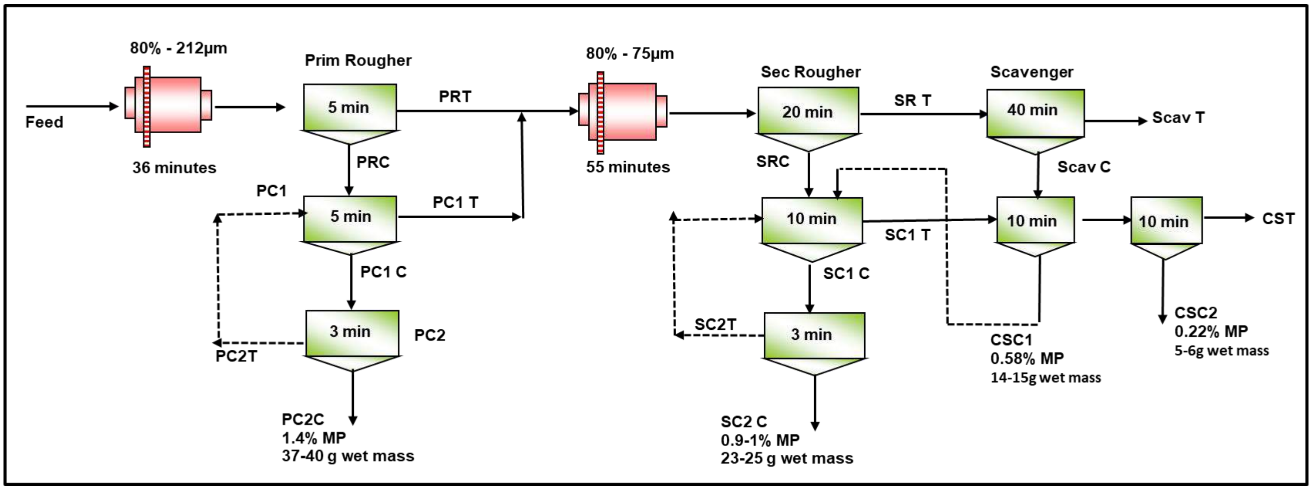
Table 13-24: Mine Blend 6 Sample Head Assays

Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
0.94	2.26	0.05	0.34	3.58	0.34	0.12	0.17

Mine Blend 6 was tested in open and locked cycle mode. Two open circuit tests were conducted using the PFS flowsheet as per Figure 13-7. In the first test a 4E recovery of 77.8% was achieved while producing a 103 g/t 4E product at a 3.1% mass pull. During the repeat test, a 4E recovery of 79.5% was achieved while producing a 102 g/t 4E product at a 2.8% mass pull. Associated Cu and Ni recoveries were 82.5% and 46.1%, respectively.

An 8-cycle locked cycle test was further conducted on this blend to test the performance in a continuous mode, as per the flowsheet presented in Figure 13-12.

Figure 13-12: Locked Cycle Flowsheet for Mine Blend 6



In locked cycle mode, a final product of 91 g/t 4E was produced at a 4E recovery of 80.9%, and a mass pull of 3.1%. Cu recovery was recorded as 84.9% with an associated Ni recovery of 46.2%. The final product had a S level of 7.9% and a Fe level of 13.7%. Magnesium oxide (MgO) and silicon dioxide (SiO_2) was reported at 16.4% and 39.6%, respectively.

13.3.5.5 Mine Blend 6 Using Waterberg Site Water

An open circuit test on the Mine Blend 6 sample was further conducted to test the impact of using Waterberg groundwater as a process water source. Refer to Table 13-25 for a summary of the measured water quality of the sample (H04-1317) used.

Table 13-25: Waterberg Groundwater Sample H04-1317 Details

Parameter	Value	Parameter	Value
pH	7.7	Manganese (Mn)	0.01 mg/l
Conductivity	204.5 mS/m	Potassium (K)	22.0 mg/l
TDS (mg/l)	1 230 mg/l	Sodium (Na)	292.2 mg/l
Total Hardness	374.1 mg/l Calcium Carbonate ($CaCO_3$)	Chloride	317.7 mg/l
Calcium (Ca) Hardness	132.3 mg/l $CaCO_3$	Fluoride	0.83 mg/l
Magnesium (Mg) Hardness	242.6 mg/l $CaCO_3$	Ammonium	<0.20 mg/l
Aluminium (Al)	0.01 mg/l	Nitrate	16.23 mg/l
Arsenic (As)	<0.03 mg/l	Nitrite	0.01 mg/l
Ca	53.0 mg/l	Orthophosphate	<0.05 mg/l
Cu	0.01 mg/l	Sulphate	62.2 mg/l
Fe	0.01 mg/l	Silica	42.0 mg/l
Mg	58.9 mg/l		

Compared to the previous open circuit and locked cycle tests conducted on the same sample, this specific test reported a lower 4E recovery of 76.1% at a final product grade of 92.1 g/t 4E and a 2.9% mass pull. Cu recovery was calculated at 80.8% with an associated Ni recovery of 45.2%. When considering the test sample head grade of 3.44 g/t 4E in Figure 13-8 and Figure 13-9, it is noted that the final product grade achieved was higher than noted during the variability testing (based on a blend of 70% F-Central and 30% T-South). The achieved 4E recovery of 76.1% compares well to a calculated recovery (i.e. a weighted average of T-South and F-Central 4E recoveries at 3.44 g/t 4E head grade) of roughly 76% based on the variability testing.

The water sample used for testing was the sample with the highest level of chlorides and nitrates, which is known to negatively affect PGE recoveries. The sample also presented with a high hardness, which can negatively affect reagent activities. The water from the various sources would be blended prior to use in the circuit, which was not reflected in the testing.

Based on the results achieved during the variability testing and the fact that only a single test was conducted using site water, there is not enough proof at this point to suggest that the Waterberg groundwater will have a negative impact on the flotation performance. It is recommended that further work be conducted to determine if, and to what level, the groundwater needs to be treated prior to use in the flotation circuit.

13.3.6 Backfill Sample Preparation (MF1 Testwork)

The following Waterberg samples were delivered to MINTEK in April 2019 to prepare final tailings samples to be used for backfill testing.

- 112 kg F-Central
- 34 kg T-South
- 72 kg F-Boundary
- 77 kg F-North

The following composite samples were prepared.

- Early Mine Blend consisting of 25% T-South: 75% F-Central.
- Late Mine Blend consisting of a 50% F-North: 50% F-Boundary.

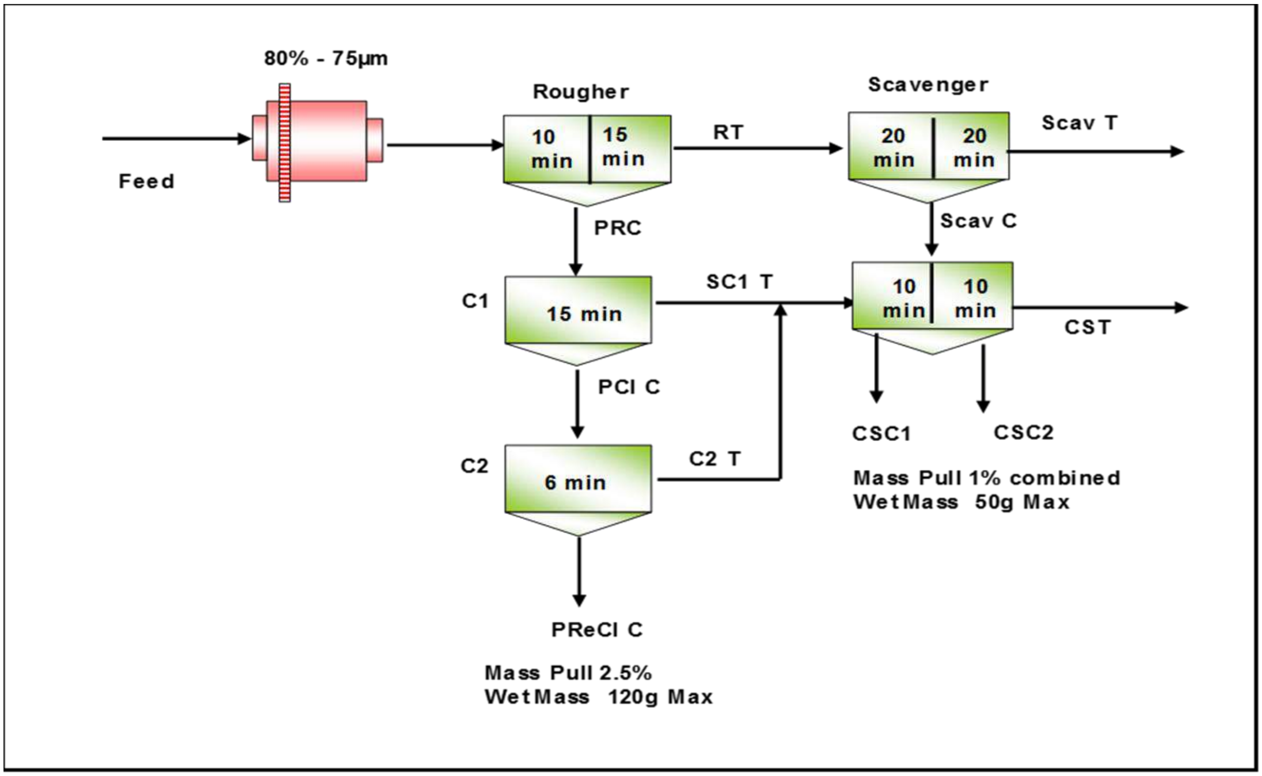
The head assays of the two composite samples are presented in Table 13-26.

Table 13-26: Backfill Tailings Sample Head Assays

Sample	Pd (g/t)	Pd (g/t)	Au (g/t)	4E (g/t)	S (%)	Cu (%)	Ni (%)
Early Mine Blend	1.84	3.99	0.48	6.31	0.50	0.15	0.21
Late Mine Blend	1.07	2.71	0.19	3.96	0.58	0.12	0.27

A MF1 flowsheet, as per Figure 13-13, was used for sample preparation.

Figure 13-13: MF1 Flowsheet Used in Backfill Tailings Sample Preparation



Refer to Table 13-27 for a summary of the MF1 circuit response of the two samples used for backfill sample preparation. These tests were not optimised for PGE recovery but were based upon producing a representative backfill product for evaluation.

Table 13-27: MF1 Circuit Performance for Mine Blend Samples

Sample Ref	Mass Pull (%)	Product Grade			Recovery		
		4E (g/t)	Cu (%)	Ni (%)	4E (%)	Cu (%)	Ni (%)
Early Mine Blend	3.3	144.0	3.7	3.2	73.9	79.3	45.7
Late Mine Blend	3.9	81.6	2.3	3.2	73.0	74.3	46.7

13.3.7 PGE & Nickel Entitlement Study

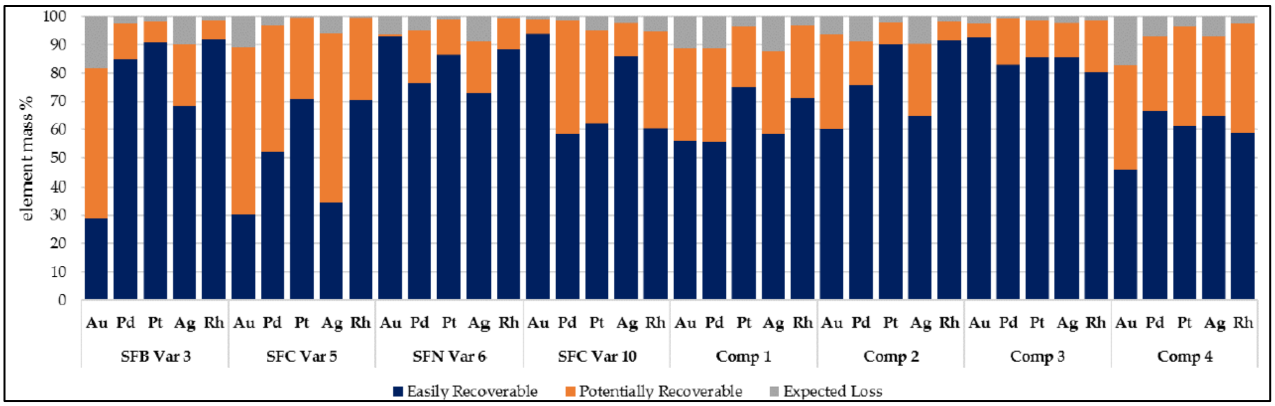
XPS Canada was contracted by PTM in April 2019 to conduct a PGE and Ni mineralogy and entitlement study on the following eight individual core samples.

- WE030D1 (SFB FT VAR 3)
- WB259D0 (SFC FT VAR 5)
- WE122D0 (SFN FT VAR 6)
- WB150D1 (SFC FT VAR 10)
- T-South Composite 1 (Intersection O222818 - O222827)
- T-South Composite 2 (Intersection O222705 - O222714)

- T-South Composite 3 (Intersection O227733 - O227745)
- T-South Composite 4 (Intersection O252959 - O252970)

The PGE mineralogy consisted of tellurides, arsenides, and alloys. Pd-rich mineralogy was consistent across the samples tested; however, the Pt mineralogy showed a difference with Pt arsenides dominating in the F zones while Pt tellurides dominated the T-South material. Expected PGM losses were noted as between 1% to 18%. Refer to Figure 13-14 for a summary of the PGE entitlement across the various samples.

Figure 13-14: PGE Entitlement Study Summary



Ni mineralogy consisted of primarily pentlandite with some Ni occurring in solid solution in pyrrhotite and several gangue species (olivine, serpentine, and pyroxenes). Trace levels of Ni arsenides were identified.

Ni entitlement was calculated based on Ni deportment, liberation, and grain size; and varied between 39% to 78% across the samples tested. Low Ni entitlement showed some correlation to low total sulphide content. Refer to Table 13-28 for a summary of the Ni entitlement findings.

Table 13-28: XPS Nickel Entitlement Study Summary

	SFB Var 3	SFC Var 5	SFN Var 6	SFC Var 10	Comp 1	Comp 2	Comp 3	Comp 4
Ni Grade %	0.24	0.15	0.27	0.24	0.10	0.21	0.07	0.04
Ni Grade in Non-sulphide %	0.07	0.06	0.06	0.05	0.02	0.03	0.03	0.02
% Ni in Sulphide	72.20	59.40	76.20	77.00	76.40	87.20	59.40	47.80
% Pn not Locked (>10 µm	80.40	67.40	81.50	84.00	84.90	90.30	79.50	82.00
% Ni considered Unrecoverable	41.90	59.90	37.90	35.40	35.10	21.30	52.80	60.80
% Ni Entitlement @P80 75 µm	**58.10**	**40.10**	**62.10**	**64.60**	**64.90**	**78.70**	**47.20**	**39.20**

Cu mineralogy is almost all chalcopyrite (Cpy). Cu entitlement was calculated based on Cu deportment, liberation and grain size and was roughly 80% for all samples except Composite 3 (which was 70% due to poorer liberation). Refer to Table 13-29 for a summary of the Cu entitlement findings.

Table 13-29: XPS Copper Entitlement Study Summary

	SFB Var 3	SFC Var 5	SFN Var 6	SFC Var 10	Comp 1	Comp 2	Comp 3	Comp 4
Cu Grade %	0.08	0.07	0.14	0.13	0.19	0.39	0.09	0.05
% Cu in Chalcopyrite	>99%	>99%	>99%	>99%	>99%	>99%	>99%	>99%
% Cpy not Locked (>10 µm)	81.70	83.60	79.60	85.40	77.10	81.00	69.90	82.30
% Cu Considered Unrecoverable	18.30	16.40	20.40	14.60	22.90	19.00	30.10	17.70
% Cu Entitlement @P80 75 µm	**81.70**	**83.60**	**79.60**	**85.40**	**77.10**	**81.00**	**69.90**	**82.30**

13.3.8 Concentrate Specification

A full chemical analysis was conducted on the concentrate products from the Mine Blend 6 locked cycle test. The results are presented in Table 13-30.

Table 13-30: Mine Blend 6 Locked Cycle Test Concentrate Analysis

Element	Unit	Value	Element	Unit	Value	Element	Unit	Value
4E	g/t	90.8	Germanium (Ge)	ppm	2.0	Scandium	ppm	7.7
Ag	ppm	6.7	Holmium	ppm	0.1	Silicon	%	18.8
Al	%	2.6	Indium (In)	ppm	<0.2	Silicon Oxide	%	40.1
As	ppm	89.3	K	%	<0.1	Samarium	ppm	0.2
Barium (Ba)	ppm	29.6	Lanthanum (La)	ppm	1.3	Tin (Sn)	ppm	6.8
Beryllium	ppm	<5.0	Lithium (Li)	ppm	<10	Strontium (Sr)	ppm	51.2
Bismuth	ppm	8.2	Lutetium	ppm	0.1	Tantalum (Ta)	ppm	1.0
Ca	%	3.0	Mg	%	10.5	Terbium	ppm	0.1
(Cadmium) Cd	ppm	1.8	MgO	%	16.7	Thorium (Th)	ppm	1.0
Cerium (Ce)	ppm	2.7	Mn	%	0.1	Titanium (Ti)	%	0.1
Co	ppm	1 262.8	Mo	ppm	10.1	Thallium (Tl)	ppm	0.6
Cr	ppm	443.6	Niobium (Nb)	ppm	1.8	Thulium	ppm	<0.05
Cesium (Cs)	ppm	0.5	Neodymium	ppm	1.0	U	ppm	0.5
Cu	%	3.3	Ni	%	2.9	V	ppm	28.6
Dysprosium	ppm	0.3	Phosphorus	%	<0.01	Tungsten (W)	ppm	2.7
Erbium	ppm	0.2	Pb	ppm	49.3	Yttrium (Y)	ppm	1.9
Europium	ppm	0.1	Praseodymium	ppm	0.2	Ytterbium	ppm	0.2
Fe	%	14.0	Rubidium (Rb)	ppm	2.5	Zn	ppm	462.7
Gallium (Ga)	ppm	4.3	S	%	8.0			
Gadolinium	ppm	0.2	Antimony (Sb)	ppm	1.2			

13.3.9 Process Plant Recovery Estimate

The process plant recovery estimate was derived using both open and closed-circuit data obtained from MF2 testwork during the PFS and DFS on the various main Waterberg deposit lithology units. All data was obtained using proven, laboratory scale, testing techniques.

13.3.9.1 Recovery Correlation Testwork

The testwork presented in Table 13-31 was used in the regression models for the recoveries.

13.3.9.2 Plant Feed Schedule

The mill feed schedule is aligned with the mining production schedule and is planned to start in January 2024. A plot of the preliminary plant feed schedule and 4E feed grades is presented in Figure 13-15.

Following are items noted from the mill feed schedule.

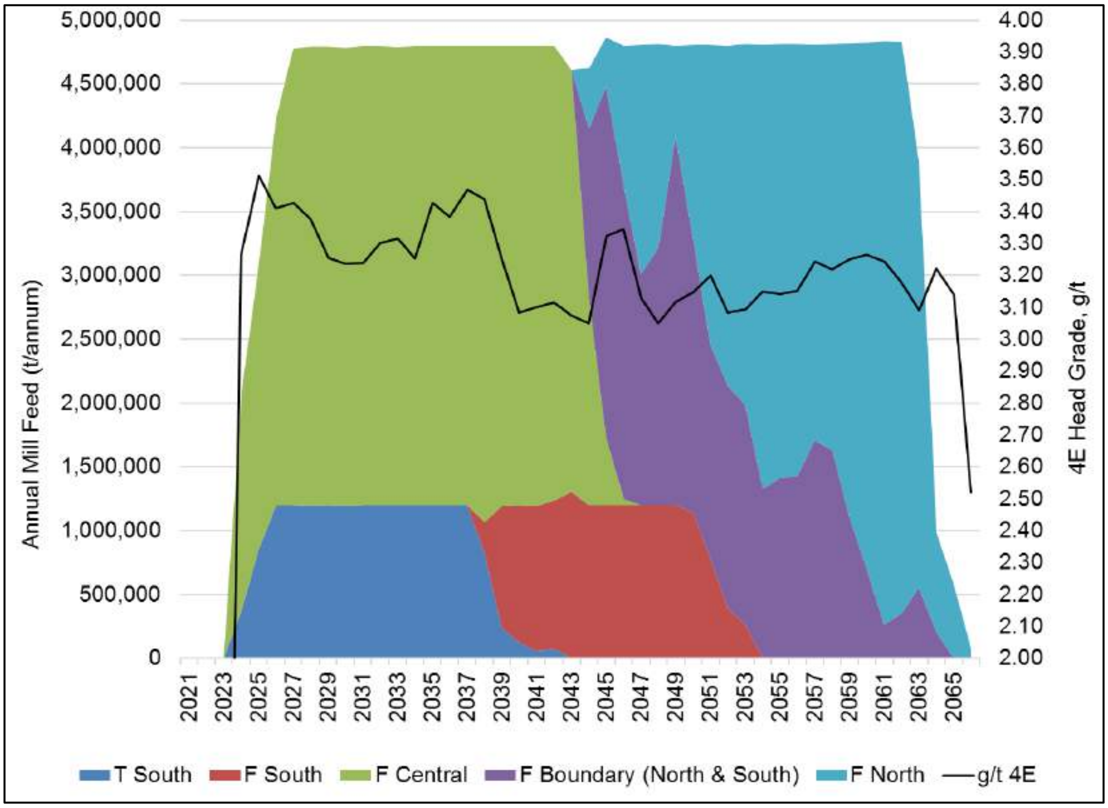
- The lithologies being treated are listed below.
  - T-South
  - F-South
  - F-Central
  - F-Boundary
  - F-North
- The 4E mill feed grade is expected to vary between 2.52 g/t and 3.77 g/t with a LOM average value of 3.23 g/t.
- The Cu mill feed grade is expected to vary between 0.06% and 0.12% with a LOM average value of 0.09%.
- The Ni mill feed grade is expected to vary between 0.14% and 0.21% with a LOM average value of 0.18%
- The blend being processed during the first 13 years of production includes roughly 25% of T-South and 75% F-Central (similar to Mine Blend 6 tested).

Table 13-31: Testwork Data Used for Recovery Modelling

Ore Type	Study Phase	Testwork Phase	Test Description (Phase-Sample-Circuit-Test ID)	Test Type	Notes
T-South	PFS	Phase 4	PH4 T2c MF2 T1	Open Circuit	
	PFS	Phase 4	PH4 T2c MF2 leachate concentration test (LCT)	Locked Cycle	
	DFS	Variability	WB222D0 - FT TZ VAR 6 Repeat	Open Circuit	Remainder of the T-South DFS variability tests had too fine grind and was not included in the recovery model.
F-South	DFS	Variability	WB156D0 - SF_FT Var 2 WB026D0 - SF_FT Var 3 WB096D3 - SF_FT Var 4 WB013D0 - SF_FT Var 5 WB156D0 - SF_FT Var 2 Repeat WB026D0 - SF_FT Var 3 Repeat	Open Circuit	WB131D1 - SF_FT Var 1 was not included in the recovery modelling as the sample head grade was below cutoff grade.
F-Central	PFS	Phase 1b	PH1 F4 MF2 New Test 6	Open Circuit	
	PFS	Phase 1b	PH1 F4 MF2 LCTNo.1	Locked Cycle	
	DFS	Variability	WB114D0 -SFC FT Var 2 WB114D0 -SFC FT Var 2 Repeat WB277D0 -SFC FT Var 3 WB113D1 -SFC FT Var 4 WB113D1 -SFC FT Var 4 Repeat WB118D0 -SFC FT Var 6 WB263D1 -SFC FT Var 7 WB150D1 -SFC FT Var 10 WB150D1 -SFC FT Var 10 Repeat WB087D0 -SFC FT Var 12 WB264D0 -SFC FT Var 13 WB095D2 -SFC FT Var 15 WB046D1 -SFC FT Var 16 WB087D2 -SFC FT Var 17 WB270D0 -SFC FT Var 18 WB085D1 -SFC FT Var 19	Open Circuit	SFC FT Var 1 – WB271D0 was not submitted for assaying due to a higher than targeted mass pull. SFC FT Var 5 – WB259D0 was not included in the recovery modelling as the sample head grade was below cutoff grade. SFC FT Var 8 – WB090D0 was not included in the recovery modelling as the grind was too fine. The following results were not included in the recovery modelling due to test accountabilities not being within required limits: SFC FT Var 9 -WB091D1, SFC FT Var 14 -WB260D0, and SFC FT Var 14 rpt -WB260D0.
F-Boundary[5]	PFS	Phase 4	PH4 F-Boundary Test 1	Open Circuit	
	PFS	Phase 1b	PH1 F-North MF2 LCT	Locked Cycle	
	DFS	Variability	WB053D2-SFB Var 1 WB154D0-SFB Var 2 WE028D0-SFB Var 5 WB079D1-SFB Var 6 WE147D1-SFB Var 7 WE030D1-SFB Var 3 Repeat WE083D1-SFB Var 4 Repeat	Open Circuit	
F-North[6]	PFS	Phase 3	PH3 EDF MF2 T7	Open Circuit	
	PFS	Phase 3	PH3 EDF MF2 LCT	Locked Cycle	
	DFS	Variability	WE099D0 - SFN 1 WE117D0 - SFN 2 WE118D0 - SFN 3 WE119D0 - SFN 4 WE129D1 - SFN 5 WE122D0 - SFN 6 Repeat WE121D0 - SFN 7 WE135D0 - SFN 9 Repeat	Open Circuit	WE124D0 - SFN 8 was not included in the recovery modelling due to test accountabilities not being within required limits.
Mine Blend 6 25% T-South:75 F-Central	DFS	Mine Blend 6 Test	Mine Blend 6 Repeat OCT	Open Circuit	
			Mine Blend 6 LCT	Locked Cycle	

[5] F-Boundary material were referred to as "F-North" in earlier phases of the PFS testwork
[6] F-North material were referred to as "Early Dawn F" in earlier phases of the PFS testwork

Figure 13-15: Life-of-Mine Mill Feed Profile



13.3.9.3 Basis of Recovery Estimate

PGE UGR (ratio between mill feed grade and final concentrate grade) versus mass pull was used as a basis to model the expected recoveries from the testwork results per ore type.

For each of the lithologies, a correlation between concentrate mass pull and Pt UGR was derived, using the test results from the tests presented in Table 13-31.

The process plant recovery estimate was derived using both open and closed-circuit data obtained from MF2 testwork during the PFS and DFS on the various main Waterberg deposit lithology units. All data was obtained using proven, laboratory scale, testing techniques and accredited analytical laboratories.

Each of the test results were weighted equally if the accountability was within expected limits (i.e. none of the test results were discounted apart from as stated). If for any tests, low accountabilities were noted for certain metals, those data points were excluded from the model (for the affected metal). The Pt UGR was used as the basis since the testwork accountabilities for the Pt results were more consistent when compared to Pd, Rh, and Au.

Once the correlation between concentrate mass pull and Pt UGR was established, correlations between the Pt UGR and the other individual PGEs (Pd, Au, and Rh) 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 Cu and Ni 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 to produce a final product with of least 80 g/t 4E.

During months where the monthly blend was similar to the Mine Blend 6 composition (30% T-South: 70% F-Central), the correlations for the Mine Blend 6 model was applied. For the remaining months, during which the Mine Blend varied, the monthly blend's PGE recoveries were calculated based on weighted averages of the individual recoveries modelled for each lithology.

The resulting recovery equations for a 30% T-South: 70% F-Central (Mine Blend 6), as well as the different Waterberg lithologies are presented in Table 13-32.

Table 13-32: Recovery Correlations for Waterberg Recovery Modelling

Mine Blend 6 (Early LOM)	
Description	**Equation**
Mass pull %	= 0.9636*(4E Head Grade)$^{1.0465}$
Pt Recovery	= 84.609*(Pt Head Grade)$^{0.0398}$
Pd Recovery	= 79.51*(Pd Head Grade)$^{0.0473}$
Au Recovery	= 74.126*(Au Head Grade)$^{0.0623}$
Rh Recovery	= 90.065*(Rh Head Grade)$^{0.07}$
Cu Recovery	= 78.739*(Mass Pull %)$^{0.032}$
Ni Recovery	= 41.062*(Mass Pull %)$^{0.136}$
T-South	
Description	**Equation**
Mass pull %	= 0.894*(4E Head Grade)$^{1.0867}$
Pt Recovery	= 85.236*(Pt Head Grade)$^{0.0772}$
Pd Recovery	= 75.939*(Pd Head Grade)$^{0.0879}$
Au Recovery	= 75.884*(Au Head Grade)$^{0.0957}$
Rh Recovery	= 120.6*(Rh Head Grade)$^{0.4757}$
Cu Recovery	= 77.073*(Mass Pull %)$^{0.063}$
Ni Recovery	= 39.771*(Mass Pull %)$^{0.119}$
F-South	
Description	**Equation**
Mass pull %	= 0.8905*(4E Head Grade)$^{1.0649}$
Pt Recovery	= 77.382*(Pt Head Grade)$^{0.0703}$
Pd Recovery	= 74.306*(Pd Head Grade)$^{0.0641}$
Au Recovery	= 74.533*(Au Head Grade)$^{0.064}$
Rh Recovery	= 62.4
Cu Recovery	= 82.693*(Mass Pull %)$^{0.033}$
Ni Recovery	= 27.618*(Mass Pull %)$^{0.229}$
F-Central	
Description	**Equation**

Mine Blend 6 (Early LOM)	
Description	**Equation**
Mass pull %	$= 0.8883*(\text{4E Head Grade})^{1.0866}$
Pt Recovery	$= 77.133*(\text{Pt Head Grade})^{0.0804}$
Pd Recovery	$= 75.056*(\text{Pd Head Grade})^{0.0862}$
Au Recovery	$= 87.499*(\text{Au Head Grade})^{0.1049}$
Rh Recovery	$= 109.17*(\text{Rh Head Grade})^{0.0744}$
Cu Recovery	$= 71.878*(\text{Mass Pull \%})^{0.062}$
Ni Recovery	$= 30.57*(\text{Mass Pull \%})^{0.225}$
F-Boundary	
Description	**Equation**
Mass pull %	$= 0.6848*(\text{4E Head Grade})^{1.2779}$
Pt Recovery	$= 70.137*(\text{Pt Head Grade})^{0.3144}$
Pd Recovery	$= 65.859*(\text{Pd Head Grade})^{0.2555}$
Au Recovery	$= 113.32*(\text{Au Head Grade})^{0.2521}$
Rh Recovery	$= 481.67*(\text{Rh Head Grade})^{0.5761}$
Cu Recovery	$= 76.083*(\text{Mass Pull \%})^{0.088}$
Ni Recovery	$= 32.464*(\text{Mass Pull \%})^{0.343}$
F-North	
Description	**Equation**
Mass pull %	$= 0.8761*(\text{4E Head Grade})^{1.0954}$
Pt Recovery	$= 77.311*(\text{Pt Head Grade})^{0.1019}$
Pd Recovery	$= 75.374*(\text{Pd Head Grade})^{0.0892}$
Au Recovery	$= 82.389*(\text{Au Head Grade})^{0.124}$
Rh Recovery	$= 107.25*(\text{Rh Head Grade})^{0.2017}$
Cu Recovery	$= 81.366*(\text{Mass Pull \%})^{0.044}$
Ni Recovery	$= 46.286*(\text{Mass Pull \%})^{0.134}$

13.3.9.4 DFS Plant Recovery Estimate

The recovery estimate for the early years as well as the total LOM is presented in Table 13-33 and Table 13-34, respectively, and is based on the following inputs.

- 1 x 400 ktpm MF2 Concentrator Plant.
- Mill feed schedule as per Section 13.3.9.2.
- PGE, Ni, and Cu recoveries calculated as detailed in Section 13.3.9.3.
- Ramp-up and commissioning losses are included on each of the individual 4E elements, as well as Cu and Ni, for each concentrate module, as listed below.
  - 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

Table 13-33: Discounted Recoveries for Early Years (2024 – 2037)

Element	Mill Feed Grade	Mass Pull	Final Product Grade	Discounted Recovery (%)
4E	3.35 g/t	3.41%	79.9 g/t	81.4%
Pt	0.96 g/t	3.41%	23.8 g/t	84.3%
Pd	2.04 g/t	3.41%	49.2 g/t	82.2%
Au	0.30 g/t	3.41%	6.0 g/t	68.1%
Rh	0.04 g/t	3.41%	0.9 g/t	72.6%
Cu	0.19%	3.41%	2.3 %	81.7%
Ni	0.16%	3.41%	2.2 %	47.8%

Table 13-34: Discounted Recoveries over Life of Mine

Element	Mill Feed Grade	Mass Pull	Final Product Grade	Discounted Recovery (%)
Pt	0.94 g/t	3.19%	23.0 g/t	78.4%
Pd	2.04 g/t	3.19%	51.4 g/t	80.4%
Au	0.21 g/t	3.19%	4.5 g/t	68.6%
Rh	0.05 g/t	3.19%	1.0 g/t	65.8%
Cu	0.09%	3.19%	2.3%	83.0%
Ni	0.18%	3.19%	2.7%	48.0%

The flotation concentrate final product target specification is a 4E grade of 80 g/t. The expected mass pull to achieve an 80 g/t 3E product is 3.19 % based on a LOM mill feed grade of 3.23 g/t 4E.

It is evident from the testwork on the various ore types that the recoveries are very sensitive to changes in mass pull.

13.4 Recommended Future Testwork

The following testwork is recommended prior to execution of the project.

- Further flotation testwork to confirm the effect of the available groundwater on flotation performance and to determine what adjustments to the raw water circuit would be required (if any).
- Concentrate thickening and filtration testwork.

13.5 Risks and Opportunities

The testwork programmes undertaken for the Waterberg DFS was of a suitable standard for an FS and were conducted at a reputable institution. Analytical results were determined at an accredited laboratory with necessary QA/QC protocols in place.

Data obtained from the various testwork campaigns (PFS and DFS), and subsequent modelling and simulation allowed the following design activities to take place.

- Confirmation of the PFS selected flowsheet and reagent suite.
- Mass and water balance development for a 400 ktpm concentrator.
- Sizing of major mechanical equipment.
- Estimation of plant operating cost over LOM.

Portions of the plant operating costs and expected overall plant recoveries were derived from the laboratory test results. Based on the testwork and engineering design performed as part of the DFS, several processing risks and opportunities were identified.

13.5.1 Flowsheet

The flowsheet developed during the PFS phase was tested during the variability testwork on each of the Waterberg lithologies over a range of head grades and confirmed to be valid during the DFS. The response on each of the ore types are captured within the recovery estimation.

The flowsheet allows for sufficient flexibility to treat each of the Waterberg ore types individually or as a blend.

13.5.2 Assaying

During the PFS, head-grade analysis using a variety of analytical methods resulted in notable assay variability despite several re-assay checks. This was most likely attributable to coarse nugget effects mostly noted on the Au and Pd assays.

During the DFS, to minimise the impact of the assay variability, a round-robin was held between several reputable assaying laboratories to determine how the head assays correlated between the various laboratories. Known Waterberg sample standards were also included as part of this exercise. A laboratory was selected based on the outcome of these results and used for all assaying during the DFS testing.

During the assaying of the DFS campaign samples, a known Waterberg sample was included for every 10 samples. For example, if the batch had less than 10 samples, 1 standard was included; and a batch of 33 samples typically included 4 Waterberg standards. In addition to these standards, the laboratory further included laboratory specific standards (typically AMIS) and blank samples, as part of their QA/QC. A total of 128 check samples (Waterberg standards, AMIS standards, and blanks) were reported. Refer to Table 13-35 for a summary of the variances noted between the measured and certified values.

Table 13-35: Variances between Measured and Certified Assays on Check Samples

	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Cu (%)	Ni (%)
Average Variance	2.7%	2.3%	4.8%	8.9%	33.3%	7.4%
90th P of Variance	5.2%	3.4%	9.1%	4.8%	5.9%	5.0%

A plot of the measured vs certified values for the PGMs is presented in Figure 13-16 (for lower-grade samples) and Figure 13-17 (for medium to high-grade samples).

Figure 13-16: PGMs Check Sample Summary for Lower-grade Samples

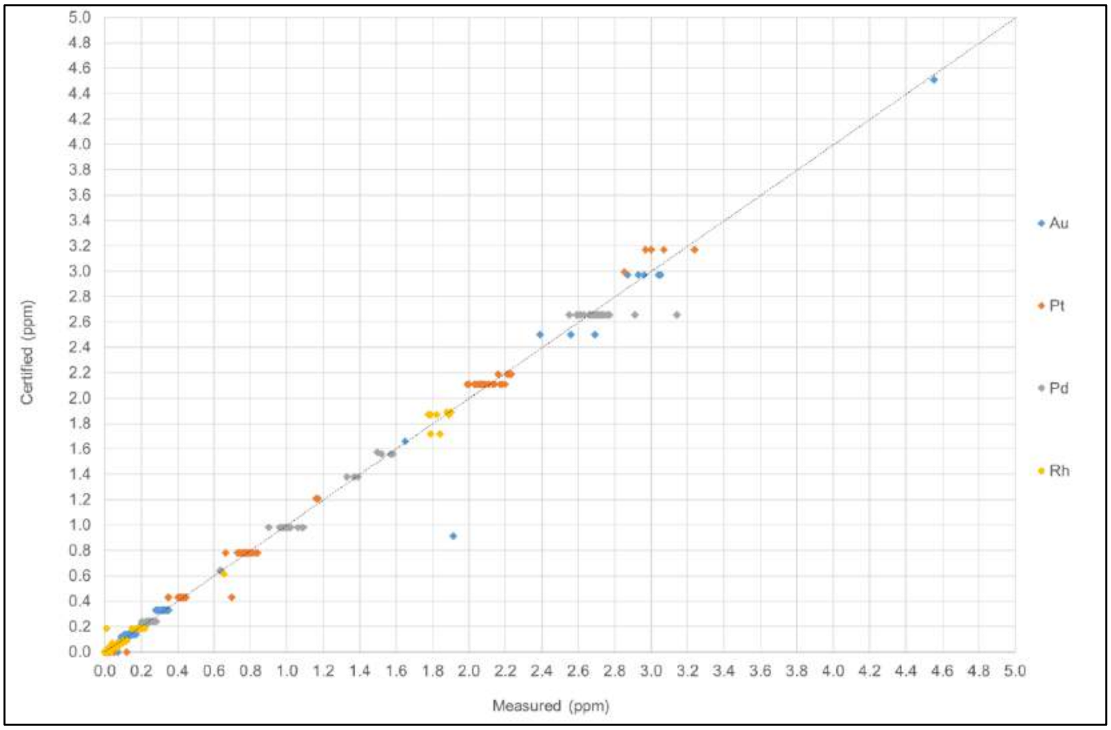
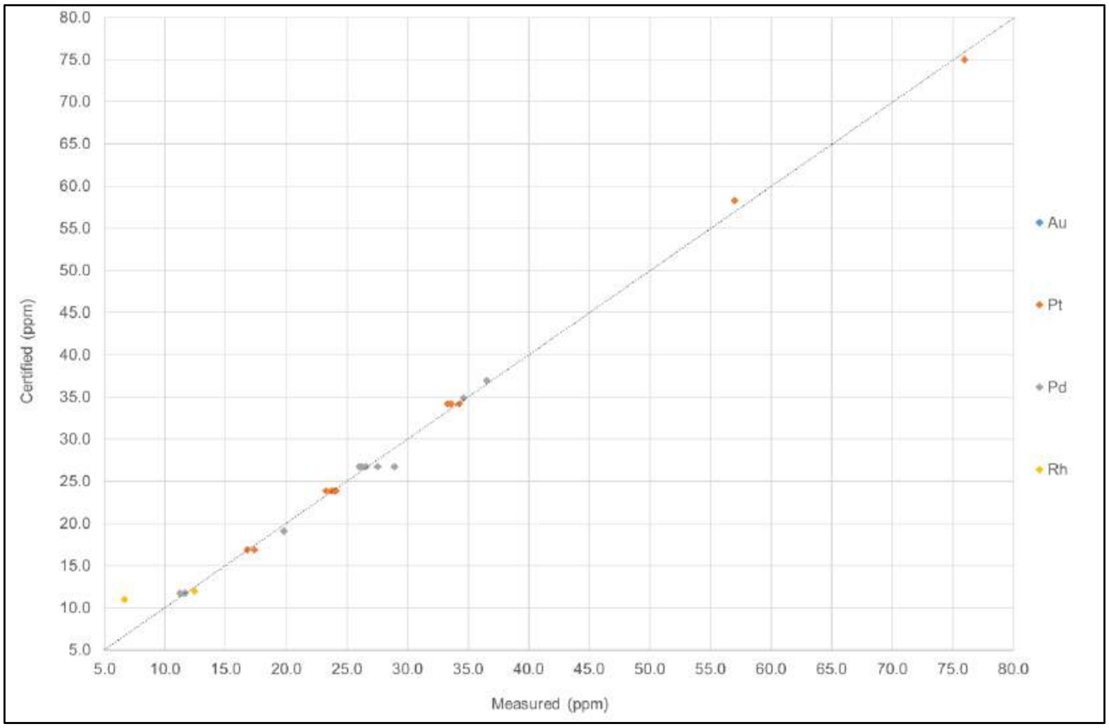
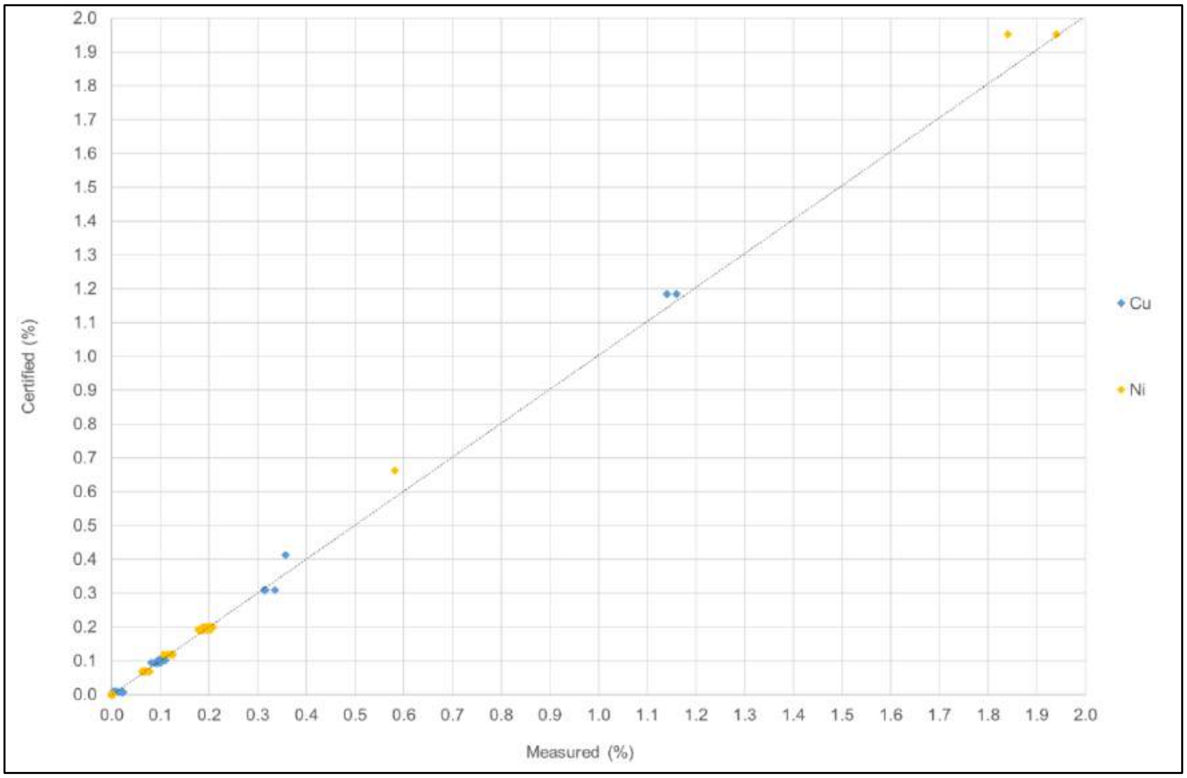


Figure 13-17: PGMs Check Sample Summary for Medium to High-grade Samples



A plot of the measured vs certified values for Cu and Ni is presented in Figure 13-18. The average variance of the measured Cu versus certified Cu assays was high at 33.3%; however, this is attributed to the low-grade spectrum where a total of 16 samples measured 0.02% compared to a measured value of 0.01%. When considering the 90th percentile of the group, the variance on the Cu assays was only 5.9%.

Figure 13-18: Copper and Nickel Check Sample Summary



13.5.3 Recovery Estimate

The recovery estimate derived for the DFS as presented in Section 13.3.9.4 was based on the results achieved from various open circuit and some locked cycle tests conducted during the PFS and DFS. It also included results from the variability testing campaign.

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 µm) 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 influence 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 as a result of bad operational practices.

To address as many of these problems as possible the plant design allows for 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 as to reduce the occurrence of losses due to bad operational practices.

13.6 Comments on Mineral Processing and Metallurgical Testing

In the opinion of the QP for Section 13 of this Technical Report, there was sufficient metallurgical evaluation completed during both the 2016 PFS and this 2019 DFS to support the process design selected, namely the MF2 circuit. The concentrator should be able to produce a concentrate containing 80 g/t 4E with a 4E recovery of about 80% from the ore being mined as per the production profile. The PFS work was completed on selected blends of ore while the DFS work was completed on variability samples across the multiple zones of the ore body as well as the expected production mine blend between T-South and F-Central, based on the mining schedule. The grade-recovery relationship for each of the 4E metals as well as Cu and Ni were established to the satisfaction of the QP.

The ore was confirmed to be hard and not compatible with SAG Milling; therefore, the three-stage crushing followed by two stages of ball milling is appropriately selected.

The use of site water (rather than Johannesburg water) for selected material during the DFS programme did not indicate significant variation in the grade or the recovery of 4E or base metals. Further evaluation is required; however, this is not expected to change the grade-recovery relationship materially.

14 MINERAL RESOURCE ESTIMATES

14.1 Estimation and Modelling Techniques

14.1.1 Key Assumptions and Parameters

The following methodology was used to produce the final Mineral Resource Models for both the F Zone and T Zone.

- Import all received information from Waterberg JV Resources into Datamine.
  - Collars.
  - Assays.
  - Downhole surveys.
  - Stratigraphic information.
  - Geological parameters.
  - Perimeters – farm boundaries, project area.
  - Aeromagnetic images.
- Detailed checks on imported data.
- Flag overall mineralised zones (F Zone, T Zone) using lithological constraints and 1 g/t 4E cutoff (separate mineralised vs disseminated, scattered and barren values).
- Create structural and overall mineralised envelope wireframes.
- Delineate geological domains based on full mineralised zones considering total vertical thickness, average grade, contained metal content and grade relationship of the geological profile (continuous, scattered etc.).
- Wireframes, drill holes and perimeters (domains) are rotated to a best fit horizontal plane.
- The drill holes are projected to an elevation datum – top contact is made flat / horizontal.
  - Create a probability model.
  - Code samples as indicators where samples above 1 g/t 4E is assigned a value of 1 and below a value of 0. A 2 m inclusive waste is considered representing internal dilution that will never be selectively stripped and forms part of the mineralised envelope to ensure a continuous ore envelope.
  - Composite indicators (1 and 0) on a 1 m basis.
  - Create an empty start model on a 5 m x 5 m x 1 m basis.
  - Estimate the 1 and 0 indicator values into the start model, which indicate the probability of a cell being ore or waste.
  - Calculate the expected ore versus waste proportion that should be applied to delineate the ore envelope from the composite samples.
  - Produce a table with proportions at various probability cutoffs.
  - Apply the expected proportion establish from the probability cutoff table to the probability model. Number of samples, distance to the estimated cells and visual checks are also considered.
  - Create a final start model for the grade estimation process.
- Flag drill hole samples using the start block model created from the probability model.

- Conduct adjustments of edge samples to compensate for block centres versus sample centres.
- Perform descriptive statistics for Pt, Pd, Rh, Au, Cu, Ni, 4E, and density, for respective geological domains.
- Compile histogram and probability plots (PP plots).
- Apply top capping (outliers), using the histograms and PP plots.
- Descriptive statistics for top cap values.
- Perform exploratory data analysis and variography on the 1 m composites within the indicator model envelope. Variography is conducted in the flattened and rotated coordinate system.
- Create a 25 m x 25 m x 1 m block model, using the start model, for grade estimation process.
- Produce a global mean model for SK.
- Grade estimation – ordinary and SK.
- Perform various model validations.
- Create a waste model.
- Convert the 25 x 25 x 1 m krig model to a 5 x 5 x 1 m model (original start model).
- Project back to the rotated plane wireframe.
- Rotate the cell centres back to original 3D space.
- Classify model into measured, indicated and inferred.
- A final Mineral Resource Model is created at a 2.0 g/t (4E) and 2.5 g/t (4E) cutoff from the *in situ* model applying a minimum width (2 m), inclusive waste of 5 m and eliminate isolated scattered cells.
- The Mineral Resource was cutoff at 1 250 m vertical depth as a preliminary initial economic limit.
- Produced Mineral Resource tables at appropriate cutoffs.

14.1.2 Data Used

A total of 147 new drill holes were drilled in the project area since the April 2016 update, targeting both the T Zone and F Zone, with another 130 deflections drilled from original holes. The total combined new metres drilled is 63 755 m of which 26 713 samples were taken with 2 603 standard reference samples and 2 490 blank samples added for the QA/QC process. Of the 273 drill intersections (including deflections), 51 intersected the T Zone and 262 intersected the F Zone mineralisation.

Data used in this estimate comprised 441 original drill holes with 583 deflections as shown in Figure 14-1. Of these, 247 intersections occurred in the T Zone ranging from approximately 200 m to 1 500 m in depth below surface as shown in Figure 14-2. Figure 14-3 shows that a total of 573 intersections in the F Zone were used ranging from approximately 200 m to 1 500 m in depth. The drill holes and spacing were sufficient to delineate the mineralised zones and continuity. The drill holes are vertical and intersect the overall mineralised zone at an average angle of 37º. All drill hole thicknesses or widths of the mineralised zones are stated as vertical thicknesses or uncorrected.

Figure 14-1: Diagram Showing Drill Holes Drilled in the Waterberg Project Area

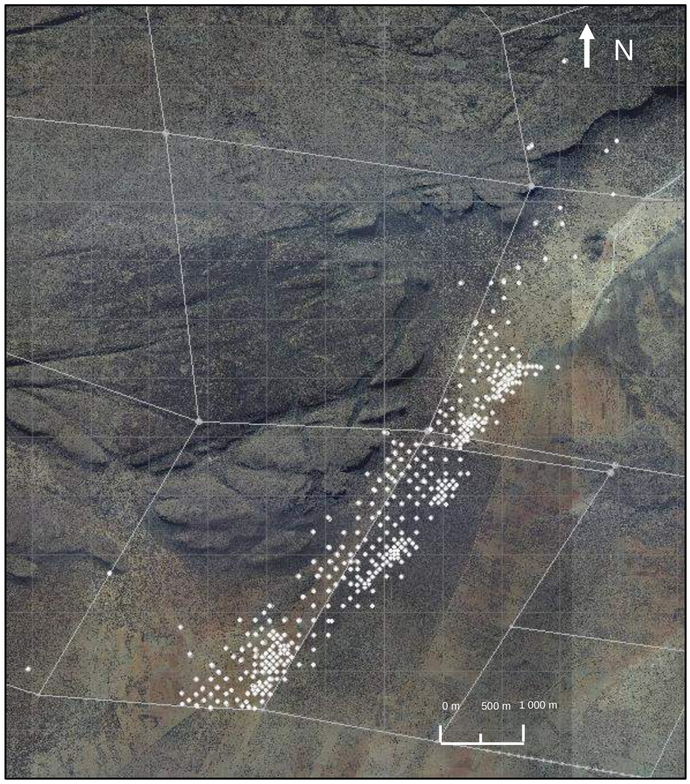


Figure 14-2: Drill Holes that Intersected the T Zone Mineralisation

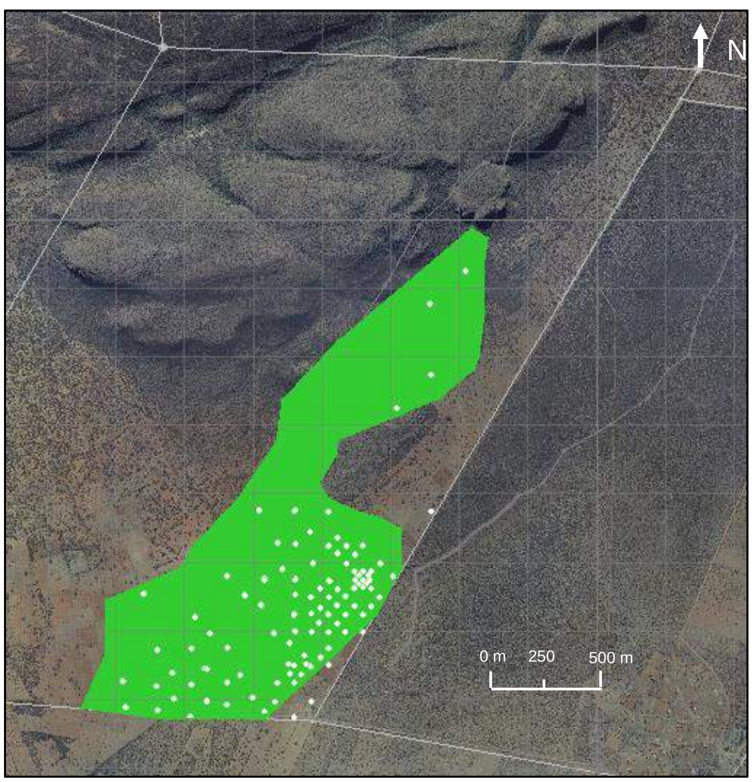
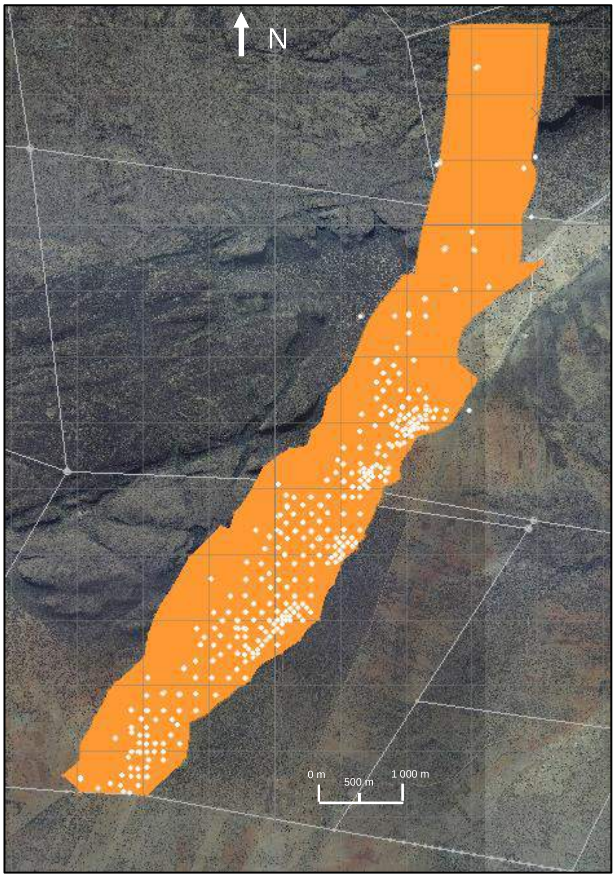


Figure 14-3: Drill Holes that Intersected the F Zone Mineralisation



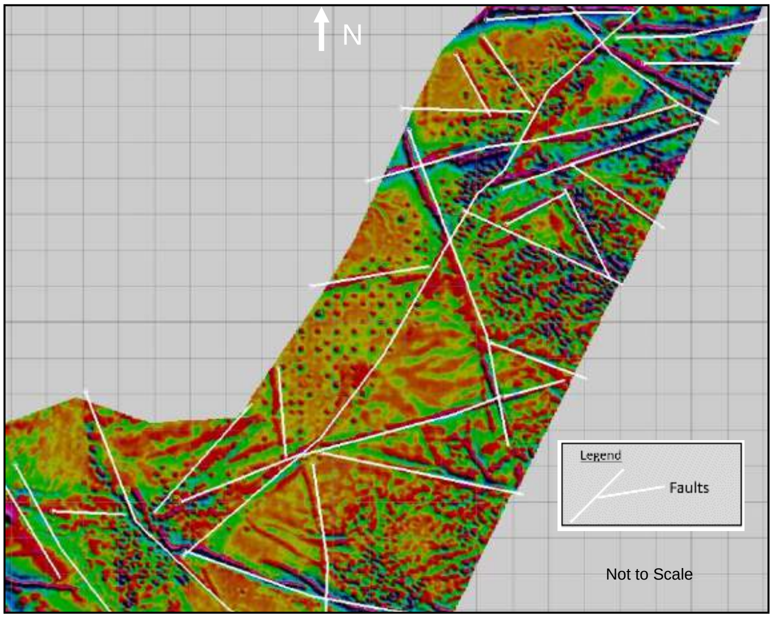
14.1.3 Structural Model

The geological understanding and relationships, including structural configuration, form the first phase and key aspect of the overall estimation process.

Aspects considered for the delineation of structural features were aeromagnetic data, stratigraphy, lithology, and mineralisation.

Figure 14-4 shows aeromagnetic data that was used as a first step in identifying the major structures. This is only an indication as these images show the structures that exist mainly in the disconformable Waterberg sediments that overlay the main mineralisation zones.

Figure 14-4: Initial Delineated Structures



The main consideration for delineating the structures are the stratigraphic units or lithological units. The Super F Zones are characterised by up to 100 m thick mineralisation that do occur as lenses on specific horizons that is not correlatable across the entire ore body, but along specific zones and directions. Depending on the section viewed, these lenses might appear to show faults, but in reality, it is different lenses along specific zones at different elevations. The mineralisation is not the best indication of faults, but rather the larger lithological units. Figure 14-5 shows that the major lithological units were used rather than the correlation of the mineralisation. The disconformable contact between the Waterberg sediments and the main mineralisation zone, base contact of the basement rocks serves as a first indication of potential faults as shown in Figure 14-6.

Figure 14-7 shows in yellow the final modelled structures. There are numerous intrusives found in the Waterberg sediments that do not extend into the mineralised zones below.

Figure 14-8 shows the top contact of the T Zone and Figure 14-9 shows the top contact for the F Zone.

Figure 14-5: Diagram Showing the Main Lithological Units used for Structural Interpretation

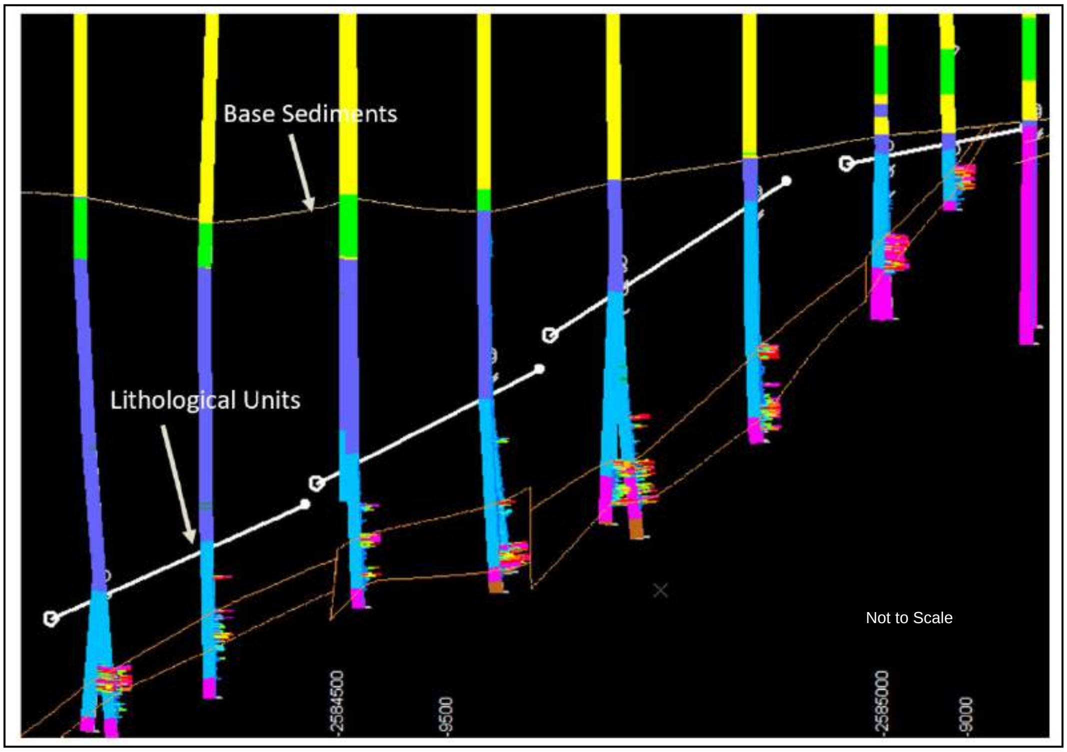


Figure 14-6: Diagram Showing Structural Relationships

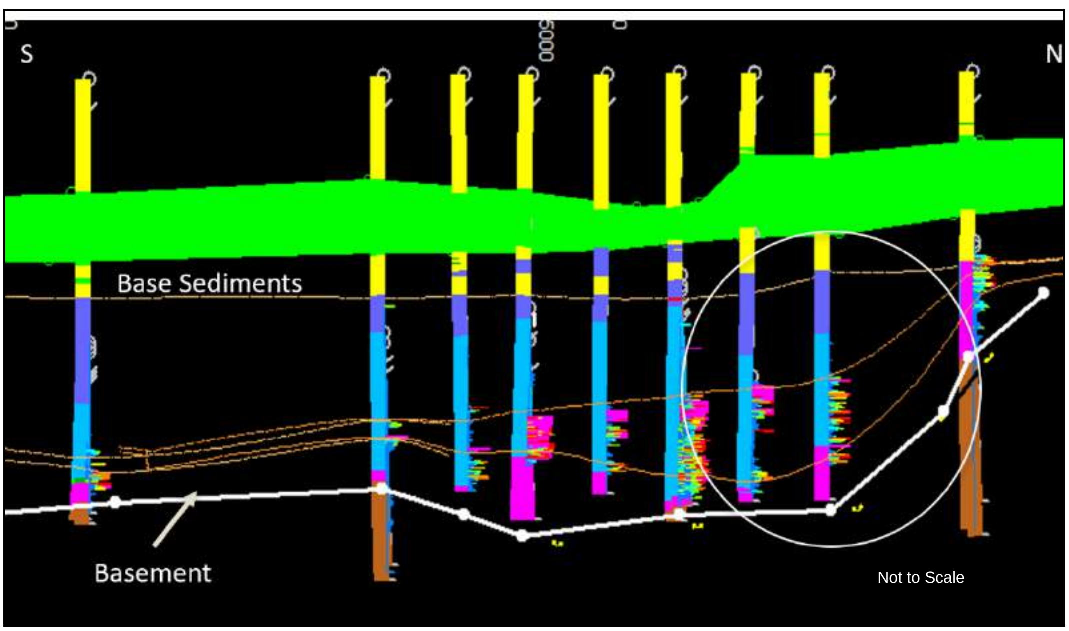


Figure 14-7: Diagram Showing the Delineated Faults for the Waterberg Project Area

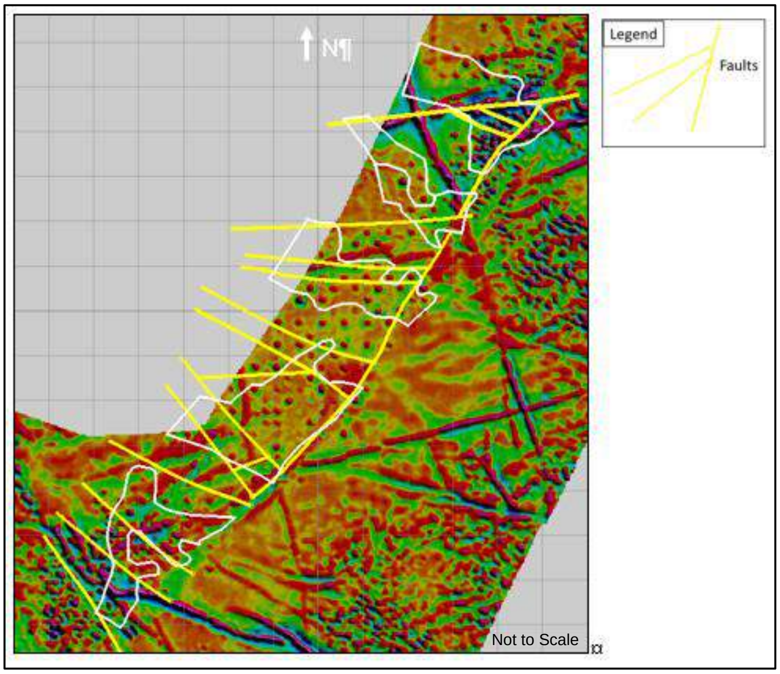


Figure 14-8: Wireframe Showing the Top of the T Zone

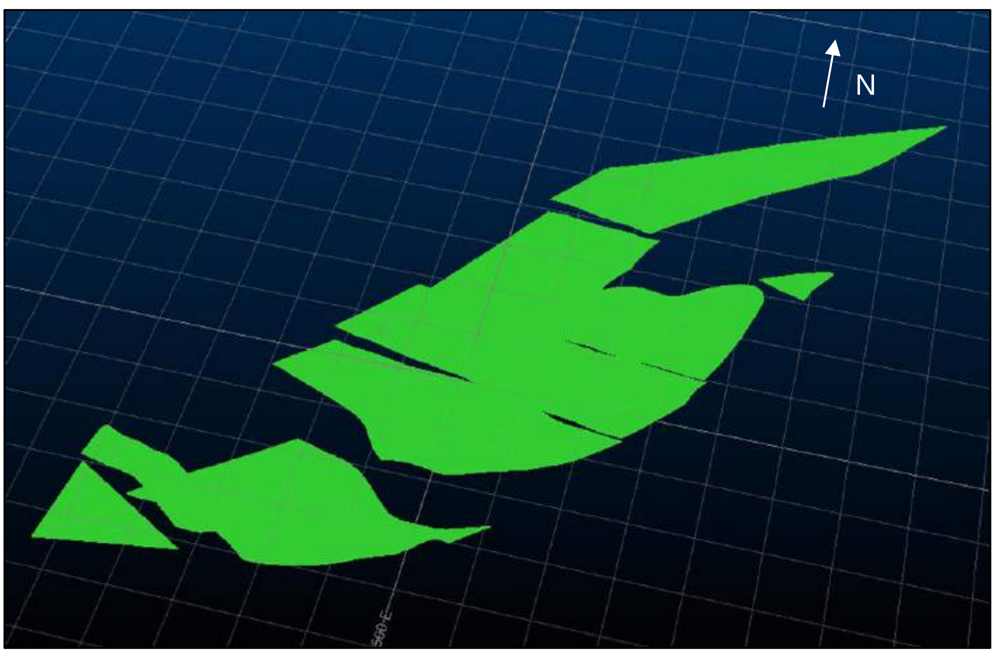


Figure 14-9: Wireframe Showing the Top of the F Zone

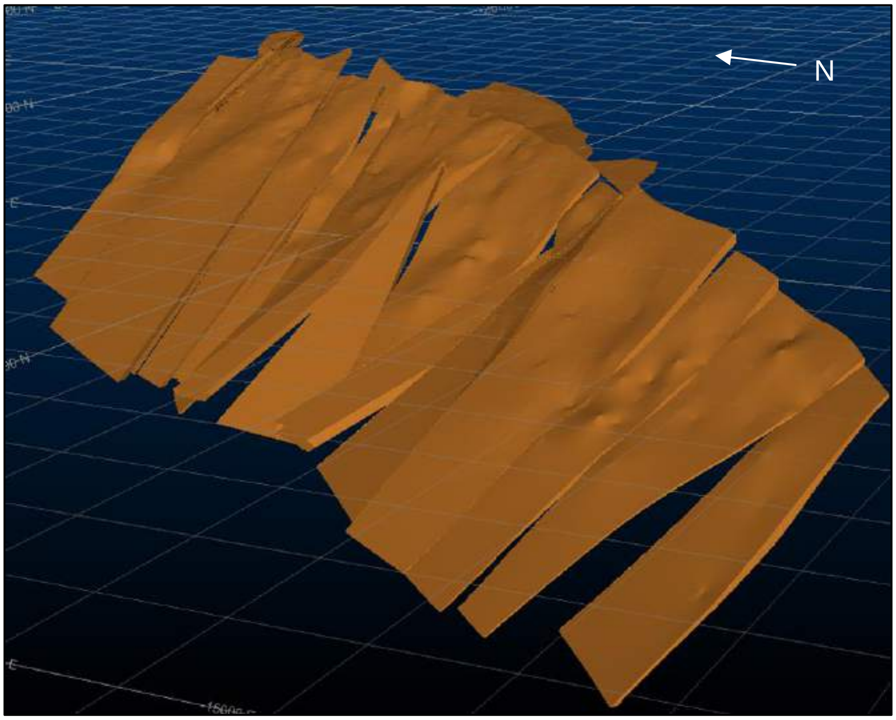


Figure 14-10 shows a strike section (southwest to northeast) of the spatial relationship between the T Zone and F Zone. The T Zone is on average 380 m above the F Zone. The TZ is at the base of the T Zone, with the T1 immediately above the TZ unit. The T0, along strike direction, is close to the T1 unit in the north-east and opens to as much as 100 m and closes again to the southwest as shown in Figure 14-10. The T0 is not developed in the southwestern portion (the down faulted block). Figure 14-11 shows that on a dip section the different units are parallel, maintaining similar distances apart.

Figure 14-10: Strike Section Showing the Spatial Relationship between T Zone (TZ/T1/T0) and F Zone

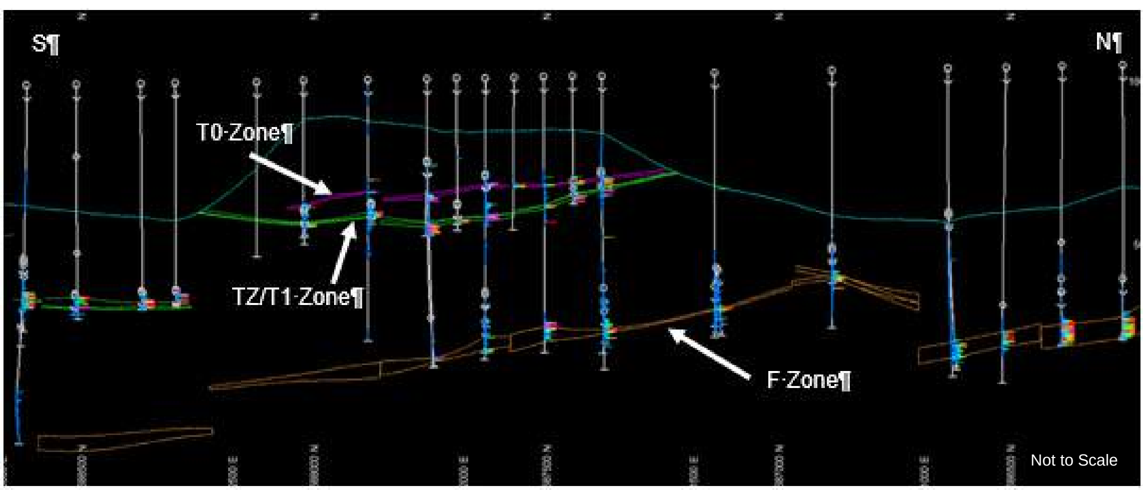
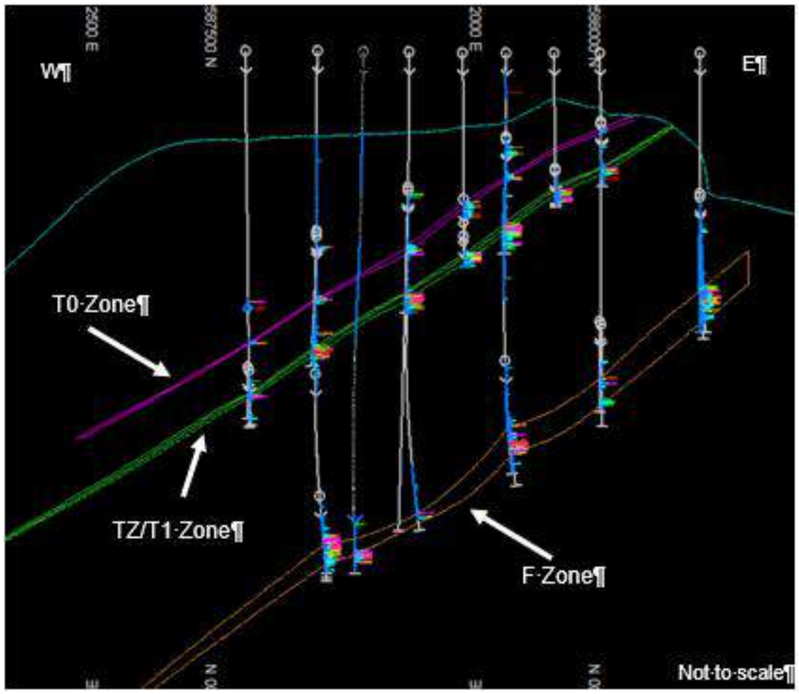


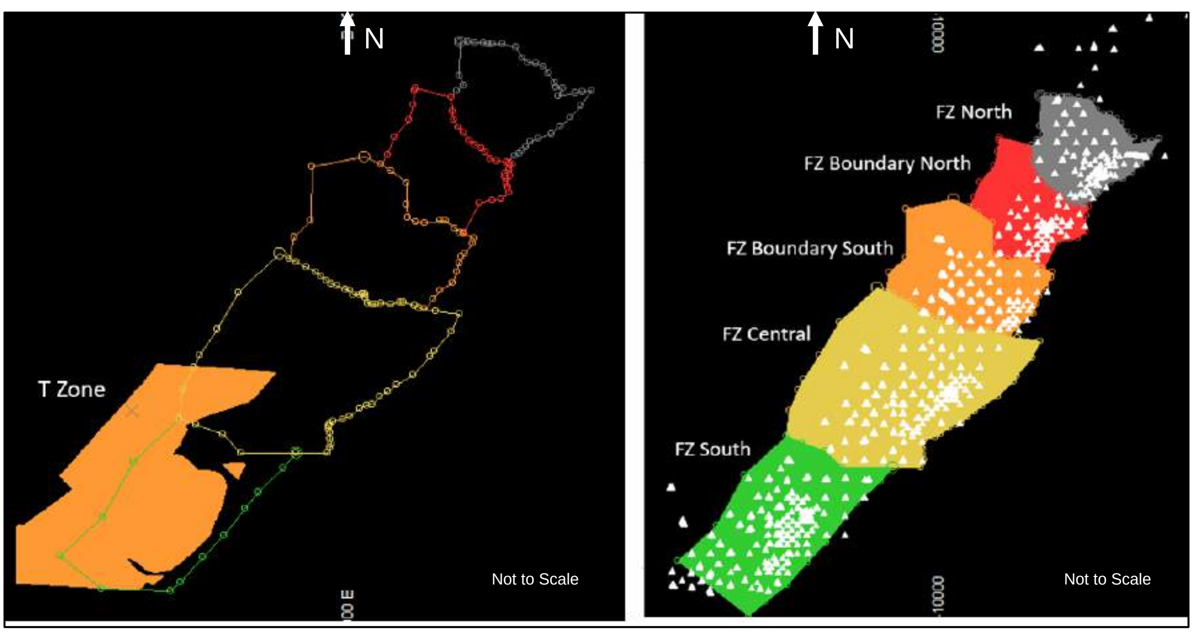
Figure 14-11: Dip Section (West – East) Showing the T Zone and F Zone Spatial Relationship



14.1.4 Project Areas

For practical reasons, the F Zone was divided into smaller project areas as can be seen in Figure 14-12, to handle the large spatial areas and block model size (number of cells etc.). The Waterberg Project boundaries were used as soft boundaries that include data from either side.

Figure 14-12: Diagram Showing the Respective Project Areas



14.1.5 Geological Domains

The F Zone, consisting of the FP and FH packages, was modelled as a single unit as no clear distinct individual units could be correlated across the project area. The T Zone has three distinct units, TZ, T1, and T0, based on mineralised and lithological characteristics.

The Waterberg Project area consists of distinct zones of mineralisation that vary in different parts of the project area. Geological domains based on various geological features including thickness of the mineralisation zones, mineralisation distribution within the zone, lithological changes and structural controls were defined.

The F Zone varies from thick (20 m – 60 m), well mineralised and continuous mineralisation (Super F Zones) to intermediate thickness (10 m – 20 m) less continuous to thin zones with scattered lower mineralisation. The T Zone is generally thinner (5 m – 10 m) with higher grades than the F Zone.

Table 14-1 shows the different parameters for respective domains for F Zone.

For the F Zone, a total of 17 domains were delineated and labeled 1 through 14 and 16 through 18 (there is no Domain 15).

Figure 14-13 shows the geological domains defined for the F Zone.

As a result, five domains were identified for the TZ unit, three domains for the T1 unit, and four domains for the T0 unit. Figure 14-14 to Figure 14-16 show these domains, respectively.

The thick well mineralised domains are referred to as Super F Zone Domains, which is also the main economic domains considered for mining as can be seen in Figure 14-17.

Table 14-1: F Zone Geological Domain Characteristics

Project Area	Domain	Vertical Thickness m	Grade 4E g/t	Metal 4E mg/t	Pt:Pd Ratio
North	1	37	0.85	29.2	0.52
North*	2	51	2.25	116.0	0.42
North	3	52	1.47	75.0	0.49
Boundary North	4	17	2.57	39.0	0.63
Boundary North	5	42	2.06	78.0	0.55
Boundary North*	6	65	1.81	131.0	0.49
Boundary North	7	35	1.82	60.0	0.46
Boundary South	8	31	1.40	27.0	0.54
Boundary South*	9	66	1.76	57.0	0.47
Boundary South	10	11	1.28	14.2	0.74
Central	11	55	0.97	55.2	0.54
Central*	12	97	2.10	196.4	0.43
Central	13	31	3.54	48.1	0.51
Central	14	11	1.21	12.7	0.54
South	16	17	1.17	21.3	0.61
South*	17	32	2.29	67.5	0.54
South	18	31	1.22	30.4	0.62

Notes:

- *Super F Zone – Domains
- There is no domain 15
- Grades are from composite drill hole intersections at 0 g/t cutoff

Figure 14-13: Geological Domains of the F Zone

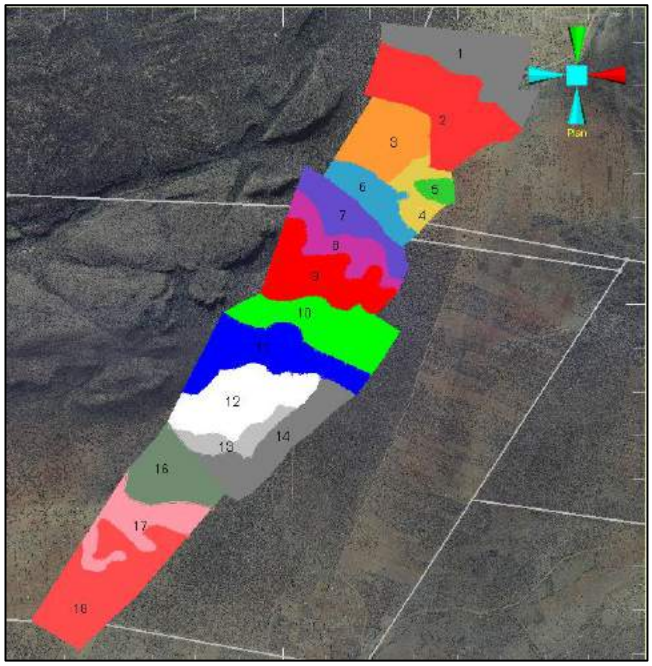
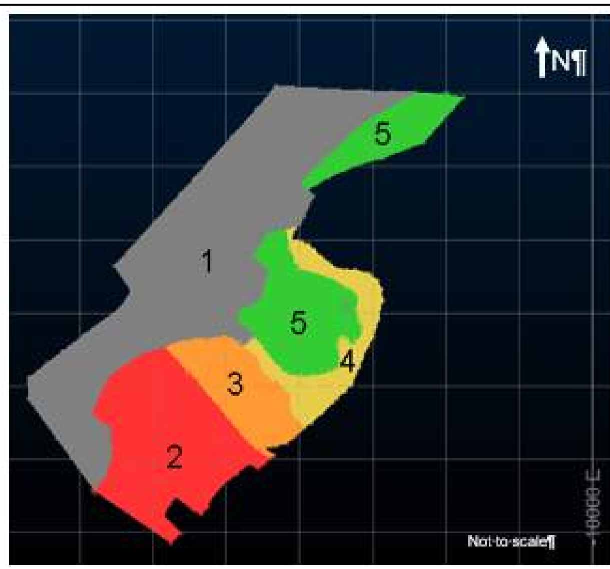


Figure 14-14: Geological Domains – TZ (Bottom Unit of the T Zone)



Domain	Vertical Thickness	Grade 4E	Metal 4E	PT:PD
	m	g/t	mg/t	
1	5.1	0.98	5.4	0.80
2	8.4	3.96	34.2	0.60
3	3.9	3.56	13.1	0.90
4	5.1	1.33	7.4	0.95
5	7.1	3.79	27.1	0.62

Note:

- Grades are from composite drill hole intersections at 0 g/t cutoff

Figure 14-15: Geological Domains – T1 (Unit Immediately above TZ)

Domain	Vertical Thickness	Grade 4E	Metal 4E	Pt:Pd
	m	g/t	mg/t	
1	4.5	0.49	2.8	0.69
2	17.5	1.09	19.3	0.65
3	12.1	0.47	6.3	0.92

Note:

- Grades are from composite drill hole intersections at 0 g/t cutoff

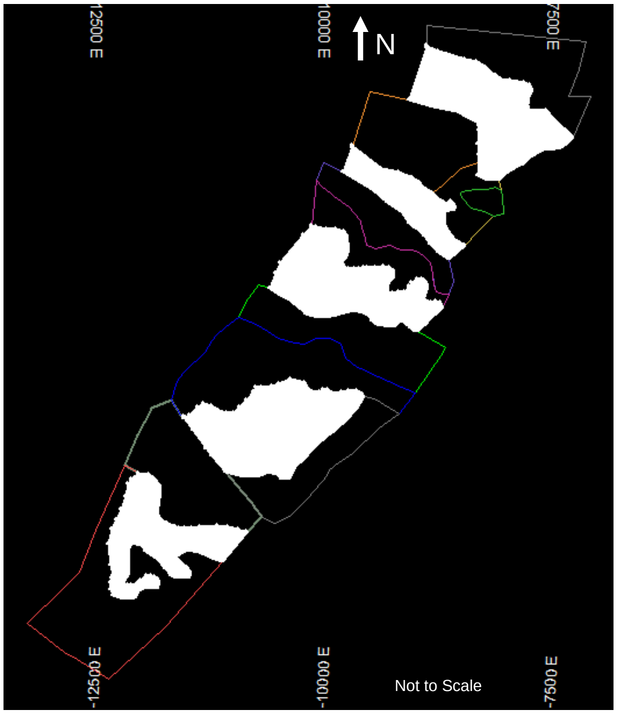
Figure 14-16: Geological Domains – T0 (Upper Unit of the T Zone)

Domain	Vertical Thickness	Grade 4E	Metal 4E	Pt:Pd
	m	g/t	mg/t	
1	3.7	1.63	6.7	0.71
2	12.8	2.28	25.1	0.67
3	4.4	1.20	5.3	0.75
4	9.1	1.92	17.2	0.64

Note:

- Grades are from composite drill hole intersections at 0 g/t cutoff

Figure 14-17: Diagram Showing the Super F Zone Domains



14.1.6 Probability Model

The first step is to delineate the overall mineralised envelope or zone in which mineralisation occurs. This was historically done by creating a wireframe on sections of the interpreted mineralised envelope. The current process uses indicators to delineate the mineralised envelope, basically on the same principles as a wireframe. From a Mineral Resource point of view, the first step is to separate mineralised material from disseminated and barren material. If higher grade portions exist and have clear continuity between drill holes, a second envelope inside the overall envelope can be delineated, etc.

It is important to understand the grade continuity of the ore body and the characteristics on all scales to eventually delineate and evaluate.

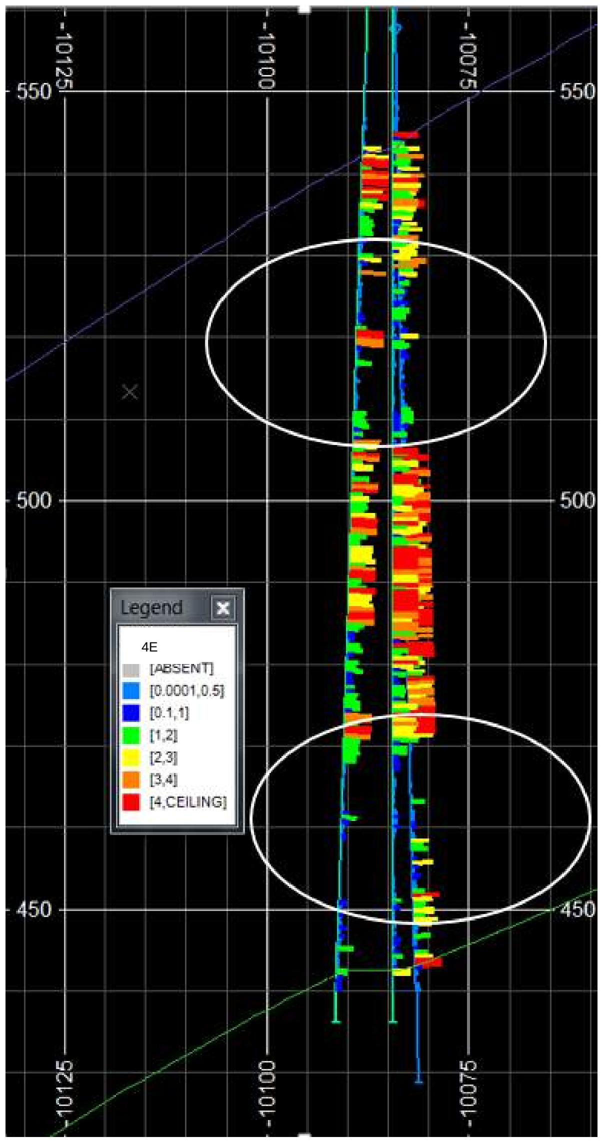
The initial drilling for the project area was on a 400 m drill spacing. Except for structural and other drill related issues all drill holes did intersect the mineralised zones over a strike length of more than 19 km. The current focus of the project extends over 8 km along strike and have more than 500 drill holes drilled. The variability of the mineralisation is the most important aspect to understand and then be modelled and evaluated.

As the mineralisation 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. Poorly mineralised or unmineralised 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 14-18 shows the discontinuous nature of the mineralisation.

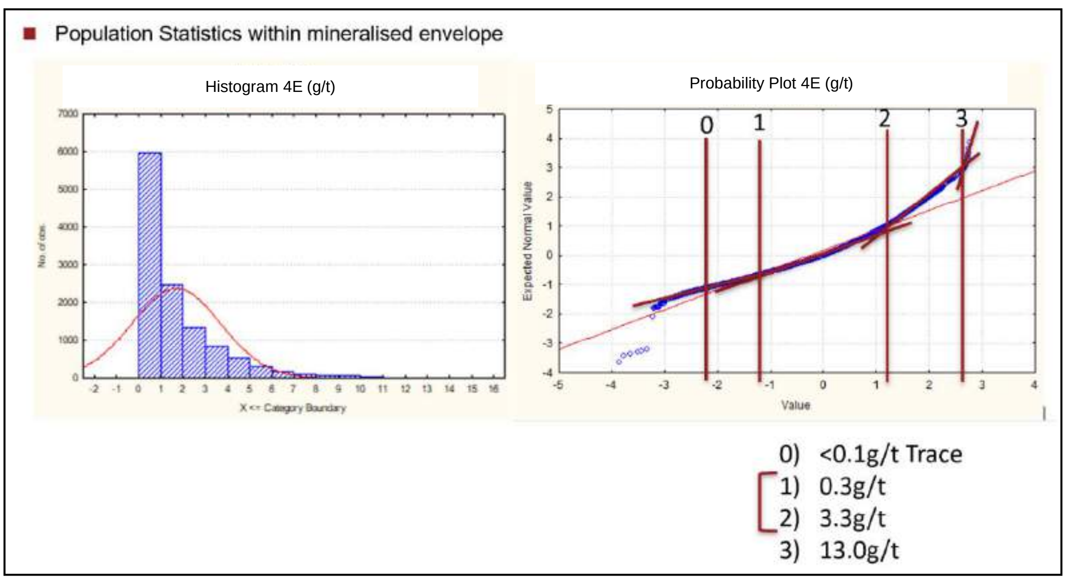
Figure 14-18: Discontinuous Nature of the Mineralised Zone



The grades show a large variability over short distances (i.e., deflection level of higher grades); therefore, selecting any high cutoff would result in incorrect delineation, especially having drill holes further apart. The reality is that the ore body cannot be drilled at 5 m intervals to capture the higher variability and the application of a high cutoff grade; therefore, the aim is to determine with wider spaced drilling the appropriate cutoff to ensure continuity if possible. The fact that there is a relatively high variability on a close space basis points to the fact that this ore body will never be evaluated, from a practical point of view (close drill spacing 10 m or less), with a high selectivity at a high cutoff grade. To isolate high grades and evaluate separately would overstate grades at the delineated volume. The high variability forces us to consider a wider range of grades to include and make it impossible to have isolated higher-grade portions delineated.

The second aspect of delineating the mineralised envelope is to consider a grade population that belongs together. If grade populations are split, there is a large risk that estimation between samples will be incorrect and not representative. The initial mineralised envelope should then represent a statistical population. Probability plots are useful to establish different populations of grade samples. Figure 14-19 shows a histogram and PP plot of grades. The PP plot shows at least five grade populations. The first one is the trace values below detection limit (left of the 0 line, < 0.1 g/t 4E). The second population is between 0.1 g/t and 0.3 g/t 4E and these represent most probably the disseminated grades. The third population is between 0.3 g/t and 3.3 g/t 4E represents most of the samples and the main mineralisation group. The fourth population is the 3.3 g/t to 13 g/t 4E and represents a smaller high-grade population within the overall population. The last population is a small number of samples and most probably the outliers. The selection of the cutoff for the delineation of the mineralised envelope should then be the 0.3 g/t. The 3.3 g/t 4E cutoff is not a continuous envelope and is contained within the larger 0.3 g/t 4E envelope. Further, the average grade of the mineralised samples is below 3 g/t 4E and selecting a cutoff close to the average would overstate grades for delineated volumes.

Figure 14-19: Histogram and Probability Plots of 4E Showing Different Grade Populations



A 1 g/t 4E cutoff grade was selected as representative of the mineralised envelope within each specific F and T Zones.

14.1.6.1 Coding of Indicators

All samples are flagged with either a 0 (waste) or 1 (mineralised) to indicate waste or mineralised zone, respectively. Samples greater than 1 g/t 4E are flagged as mineralised as shown in Table 6-11. The 0.111 value is below cutoff as shown in Table 14-2 and is included because on either side the samples are above cutoff and the lengths are less than the 2 m, which is the inclusive waste distance criteria or internal dilution that cannot be separately mined.

Table 14-2: Coding of Samples

BHID	From	To	4E	Flag
WB008D2	490.00	490.25	0.071	0
WB008D2	490.25	490.50	0.050	0
WB008D2	490.50	490.75	0.070	0
WB008D2	490.75	491.00	0.060	0
WB008D2	491.00	491.25	0.060	0
WB008D2	491.25	491.50	0.060	0
WB008D2	491.50	491.75	1.980	1
WB008D2	491.75	492.00	0.111	1
WB008D2	492.00	492.25	2.000	1
WB008D2	492.25	492.50	1.740	1
WB008D2	492.50	492.75	0.392	0
WB008D2	492.75	493.00	0.515	0
WB008D2	493.00	493.25	0.405	0
WB008D2	493.25	493.50	0.161	0
WB008D2	493.50	493.75	0.060	0

14.1.6.2 Density

Density was kriged for each block in the model similarly to grade. 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. On average the density values for the F Zone is 2.95 t/m^3, TZ is 2.91 t/m^3, T1 is 2.88 t/m^3, and T0 2.88 t/m^3.

The density values are considered by the QP to be appropriate for Bushveld type mineralisation.

14.1.6.3 Composite Indicators 1 Metre

The indicators (0 and 1) are composited on a 1 m basis to ensure they have the same support.

14.1.6.4 Create Start Model (5 x 5 x 1 Metre)

After compositing the indicators, an indicator start model is created. This has the same origin as the flattened block model with block sizes of 5 x 5 x 1 m in the X, Y, and Z direction, respectively.

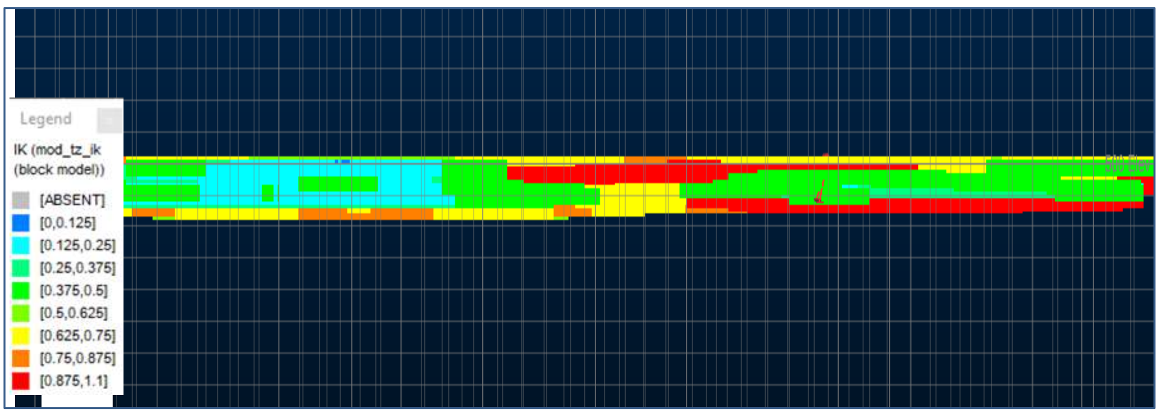
14.1.6.5 Setup Indicator Estimation Parameters

The indicator estimation uses an inverse distance squared algorithm as the data was already flagged as 0 and 1. The search ellipse was constrained to a single pass.

14.1.6.6 Estimate Indicators

The flagged indicators are estimated using inverse distant weight to obtain a mineralised envelope as shown in Figure 14-20.

Figure 14-20: Probability Model Example



14.1.6.7 Calculate Expected Percentage Ore in Envelope from Drill Hole Data

The expected amount of ore within the envelope is calculated from the composited drill hole data. This calculated figure is used in determining the most appropriate probability selection as shown in Table 14-3.

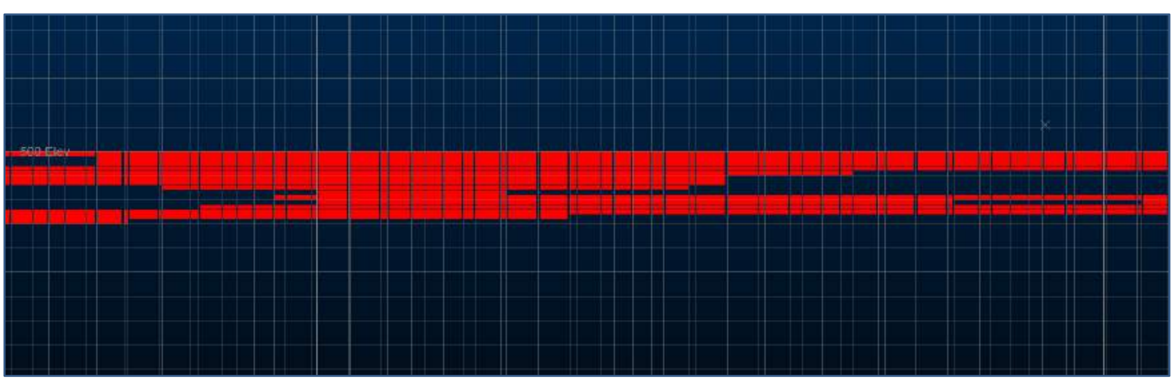
Table 14-3: Volume Relationship at Specific Probability Level Cutoffs

Probability	Tonnage	Percentage of Total Tonnage
0.00	22 330 950	
0.05	22 239 600	99.59%
0.10	22 152 975	99.20%
0.15	21 947 125	98.28%
0.20	21 632 000	96.87%
0.25	21 279 150	95.29%
0.30	20 779 250	93.05%
0.35	19 813 950	88.73%
0.40	19 183 875	85.91%
0.45	18 506 000	82.87%
0.50	17 126 400	76.69%
0.55	15 720 925	70.40%
0.60	14 509 025	64.97%
0.65	13 060 800	58.49%
0.70	10 966 350	49.11%
0.75	9 214 300	41.26%
0.80	7 386 000	33.08%
0.85	5 110 600	22.89%
0.90	3 498 175	15.67%
0.95	2 238 500	10.02%
1.00	1 097 475	4.91%

14.1.7 Estimation Start Model

After the indicators are estimated and a mineralised envelope obtain, an initial (start) model for estimation is created applying the appropriate probability level as shown in Figure 14-21.

Figure 14-21: Estimation Start Model Derived from the Probability Model Example



14.1.8 Flag Drill Hole with Final Start Model

Drill hole samples are coded using the Datamine "MOD2XYZ" process. The cells have the reef code and that is assigned to samples that lies within a specific cell.

14.1.9 Composite Ore Intersections

The drill hole intersections for both the F and T Zone intersections were composited for 4E, Pt, Pd, Au, Cu, Ni, and density on a 1 m interval. The compositing utilised the weighting of density and sample length.

14.1.10 Histograms and Probability Plots

A detailed statistical analysis showed typically skewed distributions for most of the elements to be assessed. The data was thus capped using probability and log probability plots to reduce the variability in the populations for each domain.

14.1.11 Outlier Analysis

The histogram and probability plots were used to determine the values to be top-cap (values greater than the top-cap value are set to the top-cap value) for the various domains. The maximum column in Table 14-4 represents the top-cut values applied for the T Zone and the F Zone.

Table 14-4: Top-cut Values (4E g/t) Applied for the T Zone and F Zone

Parameter	TZ	T1	T0	FZ North	FZ Boundary North	FZ Boundary South	FZ Central	FZ South
Density	3.22	3.24	3.15	3.71	3.36	3.25	3.48	3.30
Pt	6.00	2.80	5.50	4.50	4.50	3.40	6.00	4.80
Pd	10.00	6.00	8.00	8.00	7.00	7.80	11.00	9.70
Rh	0.20	0.12	0.25	0.22	0.25	0.17	0.35	0.36
Au	4.00	1.40	2.50	0.60	0.80	0.70	0.70	0.76
Ni	0.30	0.36	0.24	0.55	0.60	0.40	0.44	0.30
Cu	0.80	0.55	0.50	0.35	0.30	0.30	0.30	0.15
4E	12.00	10.00	15.00	14.00	13.00	9.50	16.00	14.50

14.1.12 Descriptive Statistics

Detailed descriptive statistics were completed on the composited data flagged within the start model as shown in Table 14-5. Each domain was analysed as well as the entire dataset for each mineralised layer.

Table 14-5: Descriptive Statistics for the T and F Zones

Parameter	Number of Samples	Min	Max	Av	Var	St Dev	Coefficient of Variation
T Zone – TZ							
Density (t/m^3)	1 105	2.616	3.22	2.91	0.006	0.075	0.03
Pt (g/t)	1 105	0.010	6.00	1.24	1.300	1.140	0.92
Pd (g/t)	1 105	0.010	10.00	2.10	4.461	2.112	1.00
Rh (g/t)	1 105	0.001	0.20	0.03	0.002	0.039	1.17
Au (g/t)	1 105	0.010	4.00	0.93	0.724	0.851	0.92
Ni (%)	1 078	0.006	0.30	0.10	0.005	0.072	0.74
Cu (%)	1 078	0.005	0.80	0.19	0.033	0.181	0.94
4E (g/t)	1 105	0.045	12.00	4.13	10.986	3.315	0.80
T Zone – T1							
Density (t/m^3)	496	2.707	3.24	2.88	0.009	0.093	0.03
Pt (g/t)	496	0.005	2.80	0.72	0.370	0.608	0.84
Pd (g/t)	496	0.006	6.00	1.24	1.306	1.143	0.92
Rh (g/t)	496	0.001	0.12	0.02	0.000	0.020	0.99
Au (g/t)	496	0.003	1.40	0.31	0.099	0.314	1.01
Ni (%)	494	0.003	0.36	0.06	0.003	0.058	0.98
Cu (%)	494	0.003	0.55	0.10	0.013	0.115	1.16
4E (g/t)	496	0.024	10.00	2.30	3.859	1.964	0.85
T Zone – T0							
Density (t/m^3)	486	2.677	3.15	2.88	0.004	0.065	0.02
Pt (g/t)	486	0.010	5.50	0.95	0.930	0.964	1.01
Pd (g/t)	486	0.020	8.00	1.53	2.430	1.559	1.02
Rh (g/t)	486	0.001	0.25	0.04	0.002	0.044	1.14
Au (g/t)	486	0.010	2.50	0.47	0.270	0.520	1.11
Ni (%)	463	0.004	0.24	0.08	0.003	0.056	0.73

Parameter	Number of Samples	Min	Max	Av	Var	St Dev	Coefficient of Variation
Cu (%)	463	0.001	0.50	0.16	0.018	0.133	0.85
4E (g/t)	486	0.061	15.00	2.98	8.574	2.928	0.98
FZ North							
Density (t/m^3)	4 350	2.515	3.71	2.96	0.003	0.059	0.02
Pt (g/t)	4 349	0.010	4.50	0.75	0.320	0.565	0.76
Pd (g/t)	4 349	0.007	8.00	1.79	1.846	1.359	0.76
Rh (g/t)	4 349	0.001	0.22	0.04	0.001	0.032	0.77
Au (g/t)	4 349	0.001	0.60	0.14	0.011	0.106	0.75
Ni (%)	4 263	0.009	0.55	0.19	0.007	0.083	0.44
Cu (%)	4 263	0.000	0.35	0.09	0.004	0.062	0.68
4E (g/t)	4 349	0.036	14.00	2.73	4.214	2.053	0.75
FZ Boundary North							
Density (t/m^3)	2 955	2.546	3.36	2.96	0.005	0.073	0.02
Pt (g/t)	2 955	0.010	4.50	0.68	0.361	0.601	0.89
Pd (g/t)	2 955	0.010	7.00	1.43	1.417	1.190	0.83
Rh (g/t)	2 955	0.001	0.25	0.04	0.001	0.033	0.91
Au (g/t)	2 955	0.001	0.80	0.13	0.013	0.112	0.87
Ni (%)	2 955	0.008	0.60	0.19	0.008	0.087	0.45
Cu (%)	2 955	0.001	0.30	0.09	0.003	0.057	0.65
4E (g/t)	2 955	0.040	13.00	2.28	3.664	1.914	0.84
FZ Boundary South							
Density (t/m^3)	3 544	2.645	3.25	2.95	0.005	0.073	0.02
Pt (g/t)	3 544	0.005	3.40	0.62	0.267	0.516	0.83
Pd (g/t)	3 544	0.005	7.80	1.36	1.248	1.117	0.82
Rh (g/t)	3 544	0.001	0.17	0.03	0.001	0.027	0.90
Au (g/t)	3 544	0.001	0.70	0.11	0.009	0.095	0.83

Parameter	Number of Samples	Min	Max	Av	Var	St Dev	Coefficient of Variation
Ni (%)	3 228	0.005	0.40	0.17	0.004	0.066	0.38
Cu (%)	3 228	0.001	0.30	0.07	0.003	0.051	0.72
4E (g/t)	3 544	0.021	9.50	2.12	2.761	1.661	0.78
FZ Central							
Density (t/m^3)	7 106	2.605	3.48	2.95	0.004	0.067	0.02
Pt (g/t)	7 103	0.005	6.00	0.72	0.441	0.664	0.92
Pd (g/t)	7 103	0.005	11.00	1.66	2.179	1.476	0.89
Rh (g/t)	7 103	0.001	0.35	0.04	0.002	0.039	1.00
Au (g/t)	7 103	0.001	0.70	0.11	0.010	0.099	0.87
Ni (%)	6 708	0.005	0.44	0.17	0.004	0.065	0.38
Cu (%)	6 708	0.000	0.30	0.06	0.002	0.049	0.82
4E (g/t)	7 103	0.021	16.00	2.53	4.970	2.229	0.88
FZ South							
Density (t/m^3)	1 459	2.699	3.30	2.97	0.006	0.079	0.03
Pt (g/t)	1 459	0.007	4.80	0.82	0.630	0.794	0.97
Pd (g/t)	1 459	0.005	9.70	1.48	2.420	1.556	1.05
Rh (g/t)	1 459	0.001	0.36	0.04	0.002	0.049	1.19
Au (g/t)	1 459	0.003	0.76	0.10	0.013	0.113	1.12
Ni (%)	1 459	0.002	0.30	0.12	0.002	0.044	0.38
Cu (%)	1 459	0.001	0.15	0.03	0.001	0.030	1.00
4E (g/t)	1 459	0.027	14.50	2.44	5.932	2.436	1.00

14.1.13 Variogram Modelling

Variograms are a useful tool for investigating the spatial relationships of samples. Variograms for 4E, Pt, Pd, Rh, Au, Ni, Cu, and density were modelled for the estimation process.

Downhole variograms are modelled to obtain the short distance spatial variance that is also an indication of the expected nugget that should be applied for the planar variograms. Figure 14-22 show an example of a downhole variogram for the F Zone.

Figure 14-22: Downhole Variogram Example

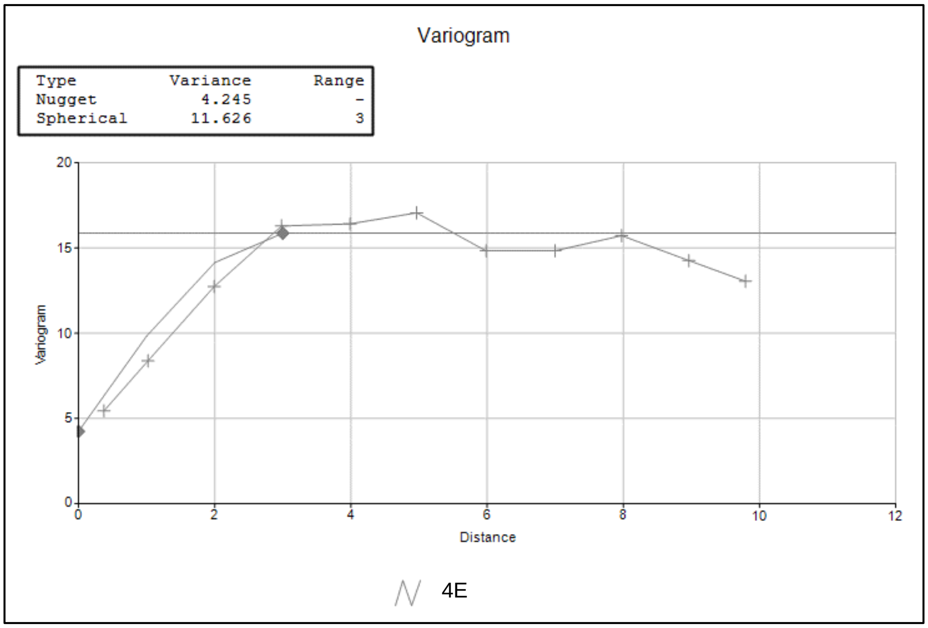


Figure 14-23 shows an example of an anisotropic planar variogram for the F Zone. Table 14-6 summarises the modelled variogram's parameters.

Figure 14-23: Example of a Variogram Model of the F Zone (4E)

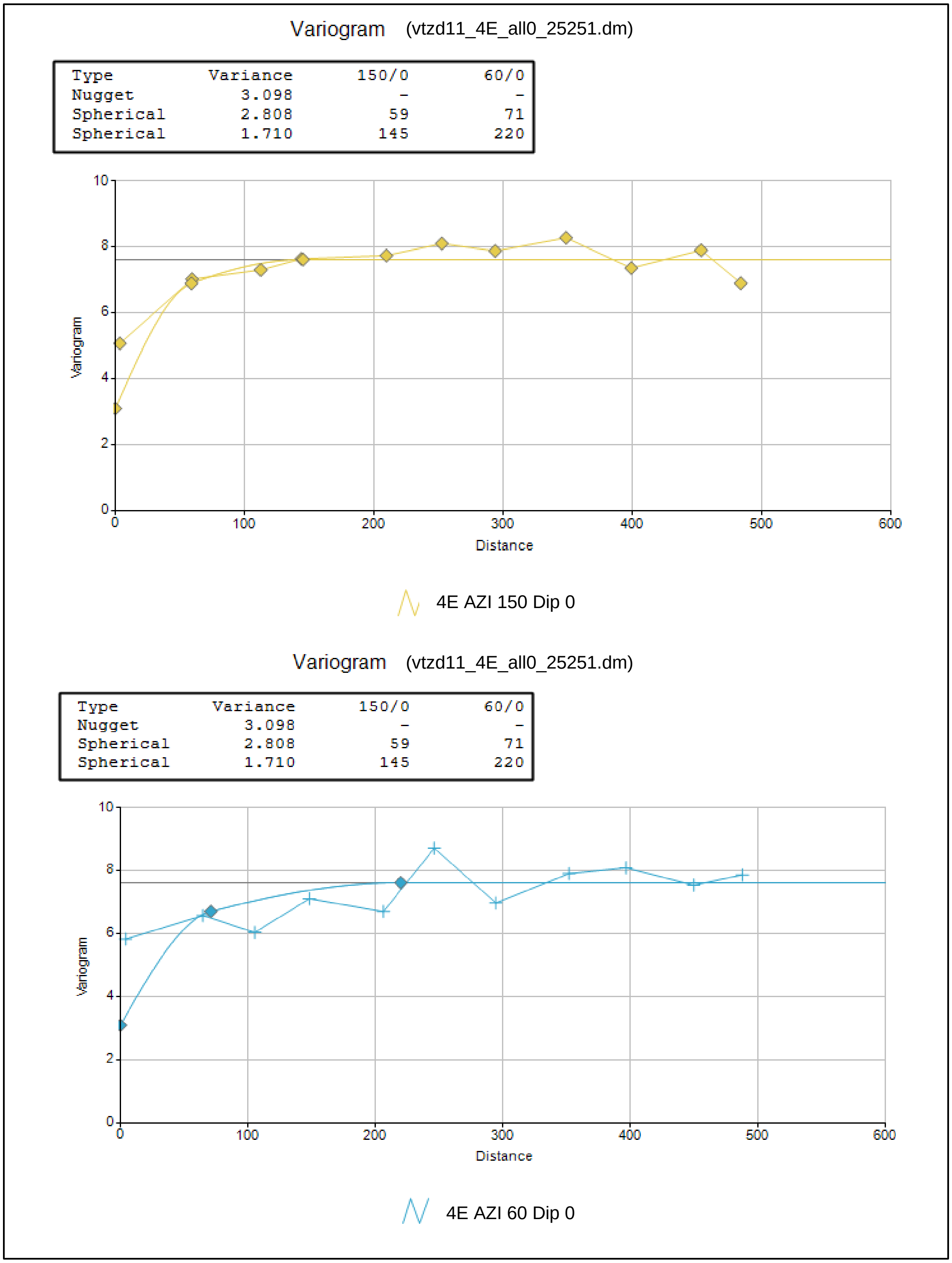


Table 14-6: Variogram Model Parameters

	Sill	Angle 1	Axis 1	Nugget (%)	Sill 1 (%)	X1 Range	Y1 Range	Z1 Range	Sill 2 (%)	X2 Range	Y2 Range	Z2 Range
T Zone – TZ												
Density (t/m^3)	0.0060	60	3	33	100	274	274	3	100	145	225	3
Pt (g/t)	1.1430	60	3	37	61	56	63	3	100	141	223	3
Pd (g/t)	3.9270	60	3	52	80	60	69	3	100	146	231	3
Rh (g/t)	0.0013	60	3	50	70	56	67	3	100	154	209	3
Au (g/t)	0.6100	60	3	46	83	39	53	3	100	118	240	3
Ni (%)	0.0040	60	3	25	25	71	88	3	100	143	235	3
Cu (%)	0.0290	30	3	34	41	50	88	3	100	218	236	3
4E (g/t)	7.6156	60	3	41	74	59	65	3	100	145	224	3
T Zone – T1												
Density (t/m^3)	0.0033	30	3	29	67	144	152	3	100	406	400	5
Pt (g/t)	0.1850	30	3	33	66	87	77	3	100	288	265	5
Pd (g/t)	0.7350	30	3	27	53	90	77	3	100	281	230	5
Rh (g/t)	0.0003	30	3	39	65	87	68	3	100	289	222	5
Au (g/t)	0.0620	30	3	29	69	85	80	3	100	281	255	5
Ni (%)	0.0005	30	3	37	77	133	91	3	100	336	288	5
Cu (%)	0.0016	30	3	34	60	116	148	3	100	289	350	5
4E (g/t)	1.8050	30	3	39	61	87	79	3	100	289	278	5
T Zone – T0												
Density (t/m^3)	0.0037	0	3	25	50	105	136	3	100	265	315	5
Pt (g/t)	0.1130	30	3	36	65	72	83	3	100	230	271	5
Pd (g/t)	0.1950	30	3	35	50	77	93	3	100	220	284	5
Rh (g/t)	0.0001	30	3	36	72	76	82	3	100	245	263	5
Au (g/t)	0.0170	30	3	29	65	69	84	3	100	218	272	5
Ni (%)	0.0013	30	3	33	73	74	89	3	100	217	254	5

	Sill	Angle 1	Axis 1	Nugget (%)	Sill 1 (%)	X1 Range	Y1 Range	Z1 Range	Sill 2 (%)	X2 Range	Y2 Range	Z2 Range
Cu (%)	0.0044	30	3	33	75	60	88	3	100	228	302	5
4E (g/t)	0.7610	30	3	33	67	75	84	3	100	214	271	5
F Zone – North												
Density (t/m^3)	0.0035	47	3	39	83	100	100	5	100	350	350	5
Pt (g/t)	0.3070	47	3	42	82	72	53	3	100	244	305	5
Pd (g/t)	1.7010	47	3	34	78	81	56	3	100	231	326	5
Rh (g/t)	0.0010	47	3	42	79	76	60	3	100	218	322	5
Au (g/t)	0.0100	47	3	40	80	73	84	3	100	225	306	5
Ni (%)	0.0070	47	3	43	71	65	86	3	100	227	308	5
Cu (%)	0.0040	47	3	25	75	71	101	3	100	221	348	5
4E (g/t)	4.0160	47	3	39	83	88	55	3	100	234	325	5
F Zone – Boundary North												
Density (t/m^3)	0.0053	30	3	40	80	100	100	3	100	314	335	5
Pt (g/t)	0.3120	30	3	42	64	97	86	3	100	286	252	5
Pd (g/t)	1.1722	30	3	36	86	101	90	3	100	291	254	5
Rh (g/t)	0.0009	30	3	42	79	76	60	3	100	285	270	5
Au (g/t)	0.0107	30	3	36	73	103	110	3	100	290	275	5
Ni (%)	0.0071	30	3	43	71	103	122	3	100	315	251	5
Cu (%)	0.0031	30	3	25	75	99	119	3	100	281	257	5
4E (g/t)	3.2466	30	3	39	68	104	100	3	100	291	245	5
F Zone – Boundary South												
Density (t/m^3)	0.0052	30	3	40	80	100	100	3	100	314	335	5
Pt (g/t)	0.2356	30	3	42	43	116	99	3	100	375	267	5
Pd (g/t)	1.1501	30	3	36	61	118	92	3	100	371	245	5
Rh (g/t)	0.0006	30	3	40	55	112	121	3	100	369	265	5
Au (g/t)	0.0084	30	3	38	75	103	110	3	100	369	252	5

	Sill	Angle 1	Axis 1	Nugget (%)	Sill 1 (%)	X1 Range	Y1 Range	Z1 Range	Sill 2 (%)	X2 Range	Y2 Range	Z2 Range
Ni (%)	0.0039	30	3	33	62	116	102	3	100	370	287	5
Cu (%)	0.0026	30	3	29	49	100	94	3	100	283	196	5
4E (g/t)	2.3209	30	3	39	63	114	100	3	100	369	245	5
F Zone – Central												
Density (t/m^3)	0.0045	0	3	25	25	150	94	3	100	255	244	5
Pt (g/t)	0.4019	0	3	31	58	96	93	3	100	248	194	5
Pd (g/t)	2.0830	0	3	34	68	100	93	3	100	261	225	5
Rh (g/t)	0.0014	0	3	34	59	109	92	3	100	250	209	5
Au (g/t)	0.0091	0	3	33	56	141	99	3	100	295	264	5
Ni (%)	0.0041	0	3	34	58	97	96	3	100	296	245	5
Cu (%)	0.0023	0	3	34	41	109	95	3	100	214	193	5
4E (g/t)	4.6709	0	3	34	51	97	91	3	100	257	215	5
F Zone – South												
Density (t/m^3)	0.0035	0	3	39	83	100	100	5	100	280	240	5
Pt (g/t)	0.4700	0	3	47	47	114	62	3	100	254	170	5
Pd (g/t)	1.8752	0	3	34	42	116	65	3	100	293	193	5
Rh (g/t)	0.0019	0	3	27	28	112	110	3	100	236	209	5
Au (%)	0.0108	0	3	27	27	134	72	3	100	281	236	5
Ni (%)	0.0020	0	3	50	50	115	95	3	100	262	253	5
Cu (g/t)	0.0008	0	3	42	43	103	143	3	100	240	267	5
4E (g/t)	4.4656	0	3	34	37	109	63	3	100	280	200	5

14.1.14 Global Mean Model

SK using a global mean was used to estimate in areas where there is insufficient data and the model needs to be extrapolated into these areas. The SK model was generally applied in the inferred Mineral Resource category. Global means were calculated for several block sizes / de-clustered data orientations. Based on this exercise an appropriate global mean was selected for use in the SK estimation.

SK was generally used for the second and third search radius while OK was used for the first search radius.

14.1.15 Grade Estimation

Estimation was completed using Datamine Studio^{TM} ver21 and Minesoft's geostatistical package 'RES ver4.'

Grade parameters estimated were estimated 4E, Pt, Pd, Rh, Au, Ni, Cu, and density using OK and SK.

The following applies to the Mineral Resource area and was undertaken using Minesoft (Pty) Ltd.'s 'RES' geostatistical programme. The following parameters were used in the kriging process.

- 25 m x 25 m x 1 m Block Size
- 3D Estimation was Conducted
- Search Ellipses Aligned with the Variogram Ranges
- Minimum Number of Samples = 18
- Maximum Number of Samples = 30
- Interpolation Methods – OK and SK

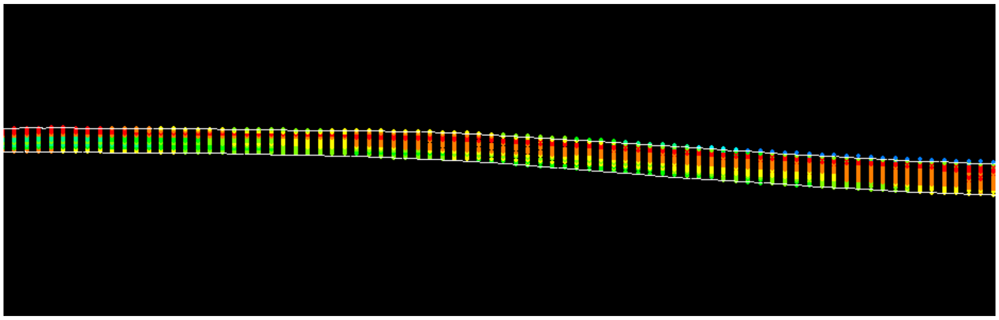
14.1.16 Model Validation

The models are validated based on several parameters. A visual validation comparing drill hole grades to block model grades, swath plots, search volumes, number of samples used in an estimate, distance from samples that represent the variogram ranges, kriging efficiency and slope of regression plots are all used to validate the estimation process.

14.1.17 Rotate Back to Rotated Plane

The kriged models are subdivided into smaller cells 5 x 5 x 1 m, maintaining the parent cell grades. These cell centres are projected back to the rotated plane as can be seen in Figure 14-24.

Figure 14-24: Example of Cell Centres Projected Back to Rotated Wireframe



14.1.18 Rotate Back to Original Three-dimensional Space

Figure 14-25 shows the 5 x 5 x 1 m cell centres back rotated to the original 3D plane. The cell centres are converted to a block model and represent the final *in situ* Mineral Resource Model as shown in Figure 14-26.

Figure 14-25: Example of the Back Rotated Cell Centres to Original Three-dimensional Space

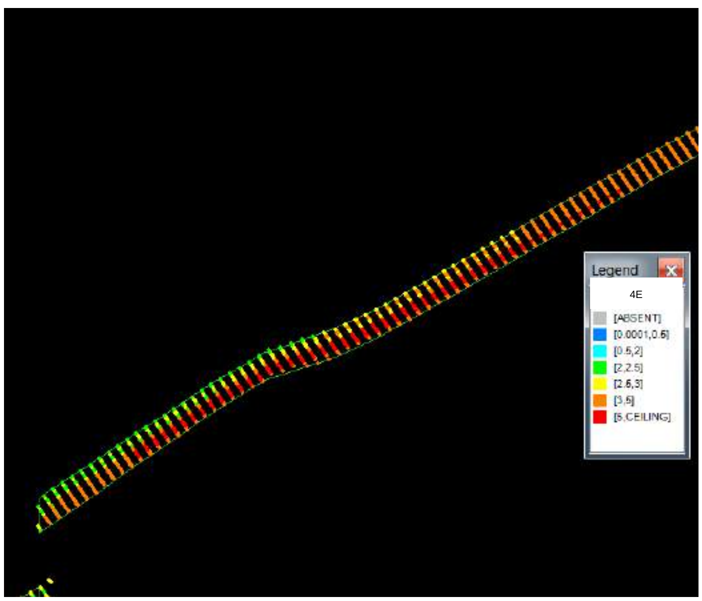
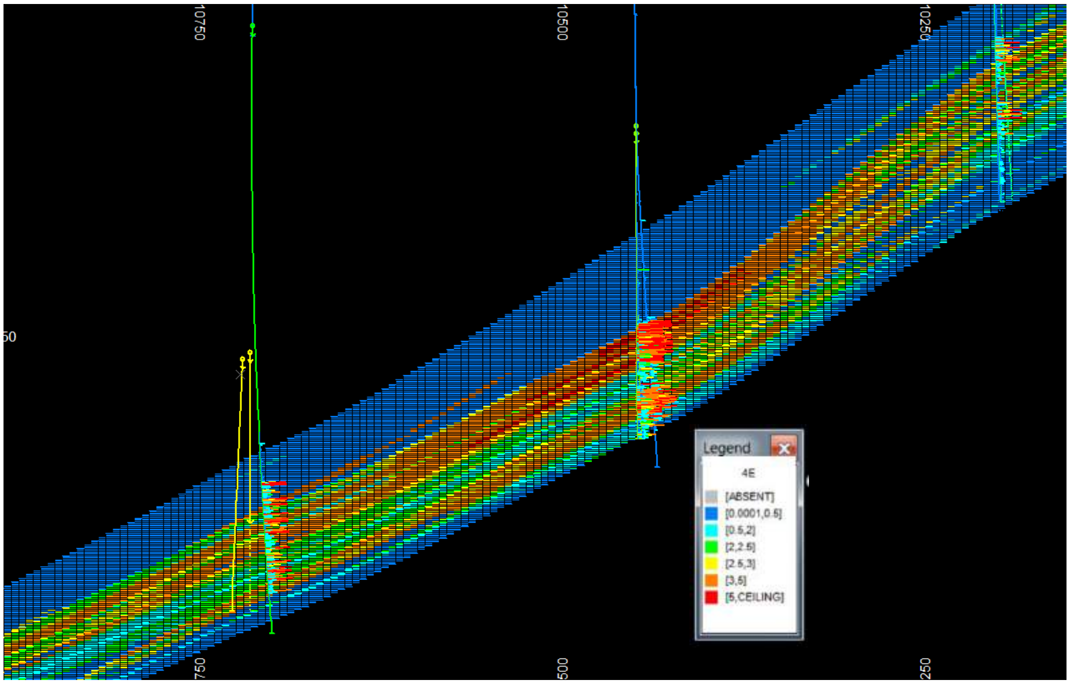


Figure 14-26: Example of the Final *In Situ* Mineral Resource Model



14.1.19 Conversion to Planned Mineral Resource Model

The *in situ* Mineral Resource Model has 1 m thick envelopes and some scattered cells that will not be mined. The Final Mineral Resource Model (Planned Mineral Resource Model) is finalised using specific criteria to eliminate thin slices and scattered mineralisation as well as ensure continuity.

The following parameters were considered creating the Planned Mineral Resource Model.

- A 2.0 g/t (4E) cutoff determined from economic parameters and a 2.5 g/t (4E) cutoff, which is the preferred option for the DFS.
- 2.5 m minimum width (vertical), actual corrected width is close to 2 m.
- Inclusive waste (internal dilution) grades need to be above the cutoff if waste portions are included. The T Zone units (TZ, T1, and T0) used 3 m and the thicker F Zones used 5 m.
- Isolated / scattered cells are eliminated.
- Subtract fault losses.

Figure 14-27 shows an example of the conversion from *in situ* resource model to the final planned resource model.

Figure 14-27 shows the initial overall vertical thickness of the delineated mineralised zones for the F and T Zones. Figure 14-28 through Figure 14-32 show the planned Mineral Resource

Model parameters at a 2.0 g/t cutoff (4E) and other applied parameters as discussed above. The plots represent a cumulative value in the vertical dimension for applied parameters.

Figure 14-27: Diagram Showing the *In Situ* versus Planned Mineral Resource Model

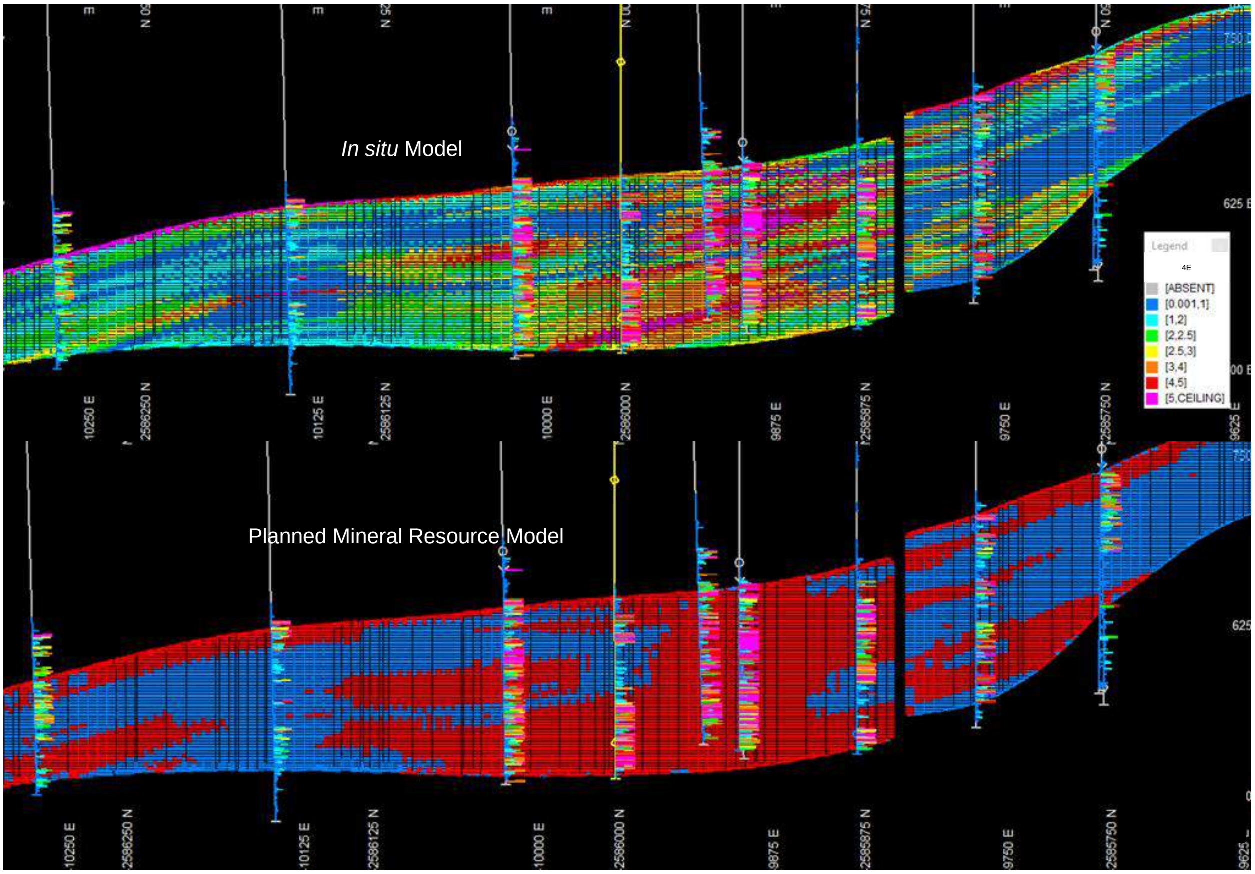


Figure 14-28: Initial Vertical Thickness of Respective Mineralised Zones

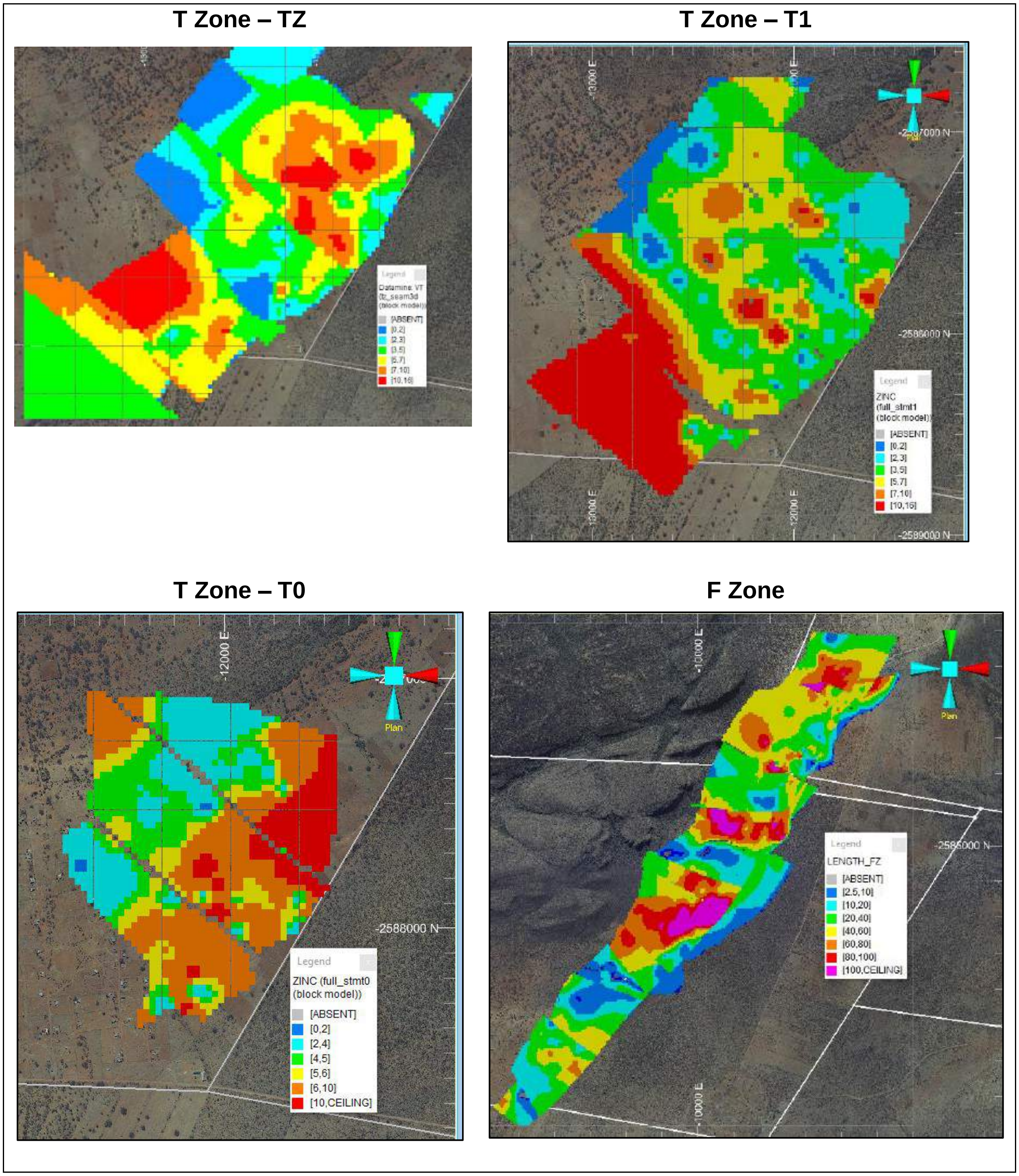


Figure 14-29: Planned Mineral Resource Model Plots (2.0 g/t Cutoff) for T Zone – TZ

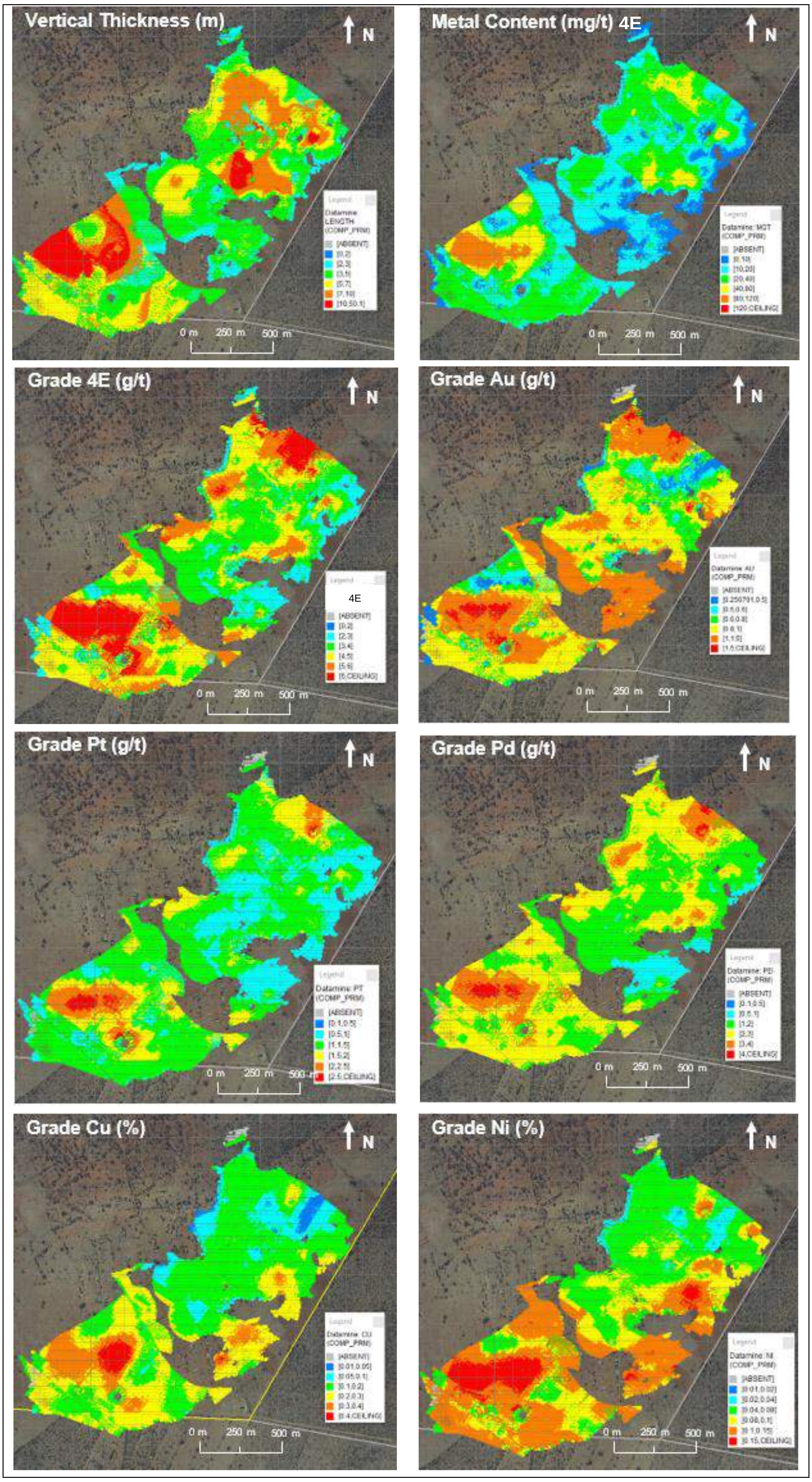


Figure 14-30: Planned Mineral Resource Model Plots (2.0 g/t Cutoff) for T Zone – T1

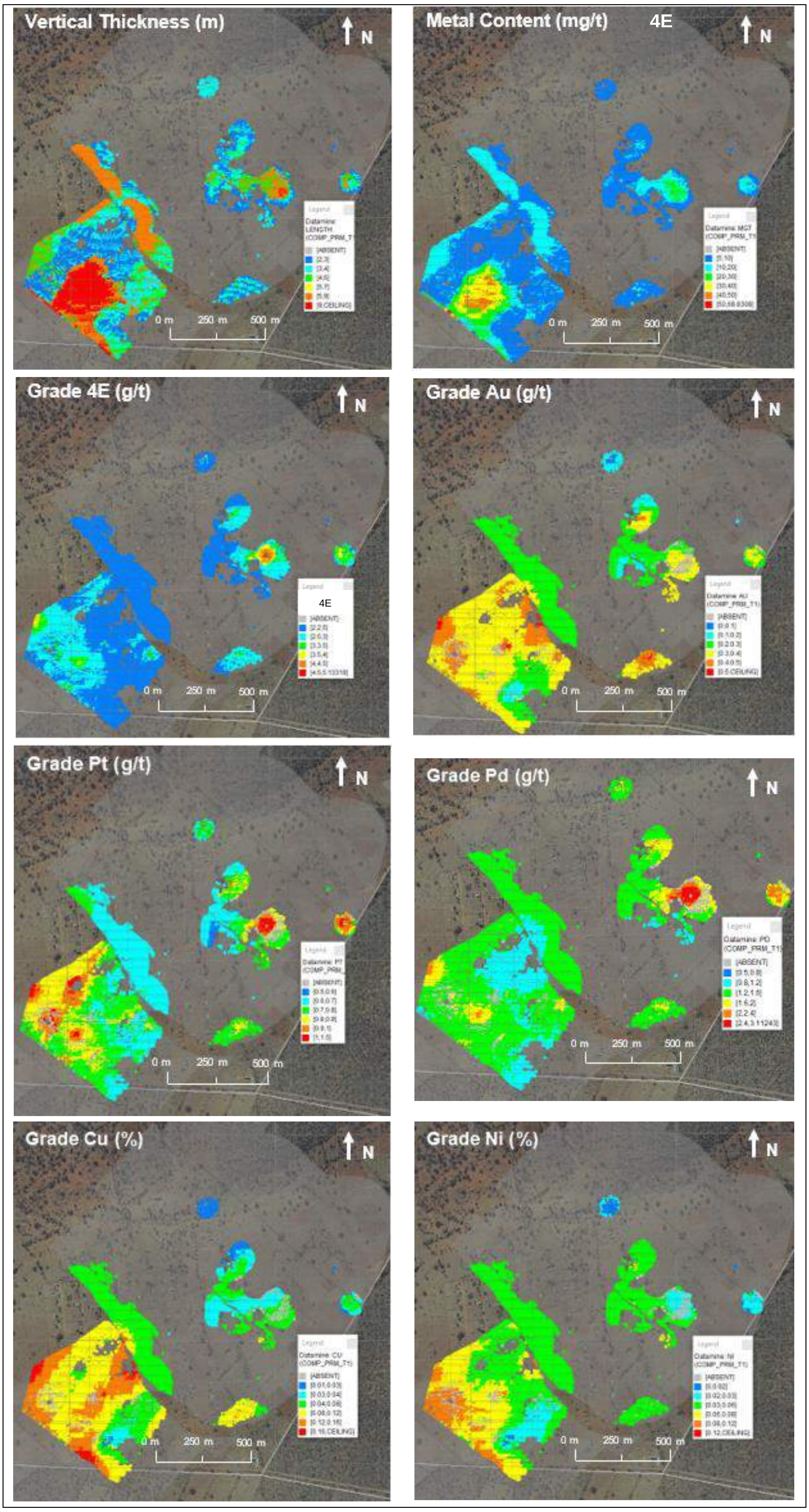


Figure 14-31: Planned Mineral Resource Model Plots (2.0 g/t Cutoff) for T Zone – T0

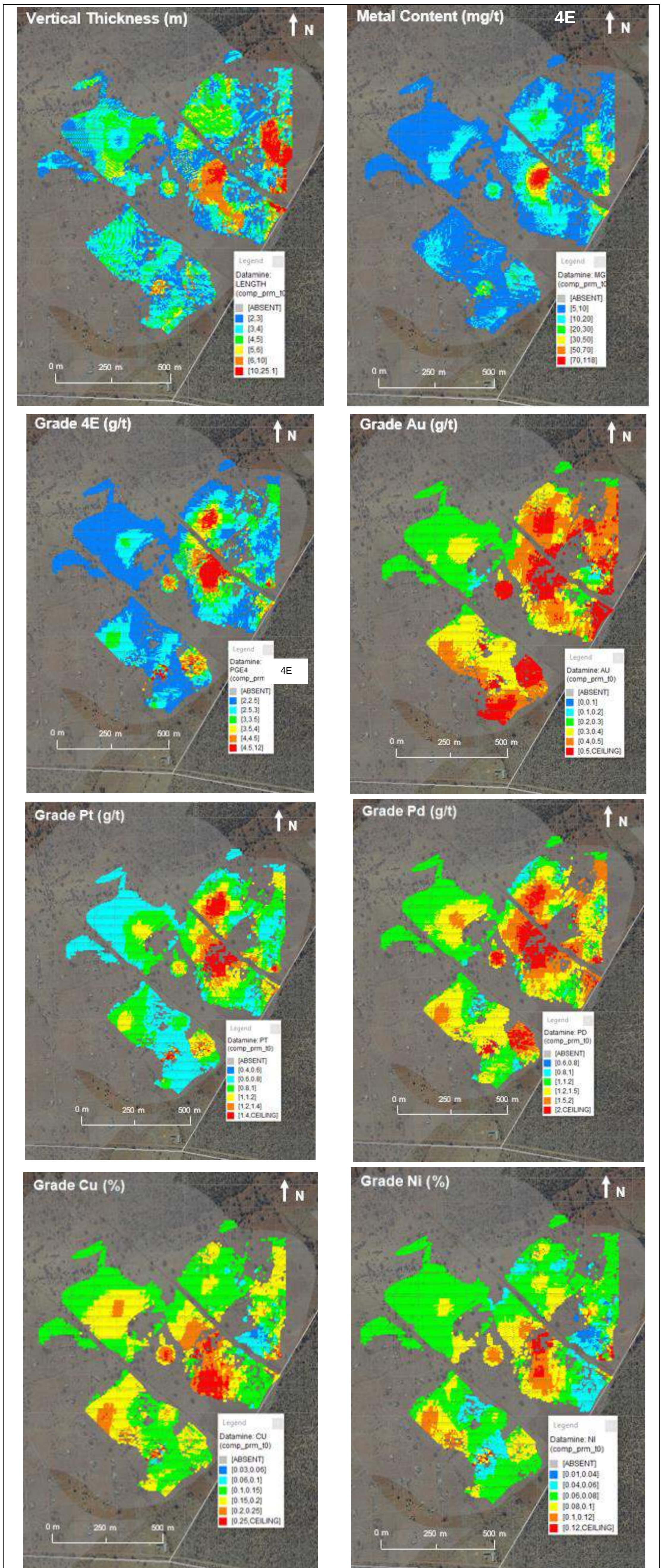
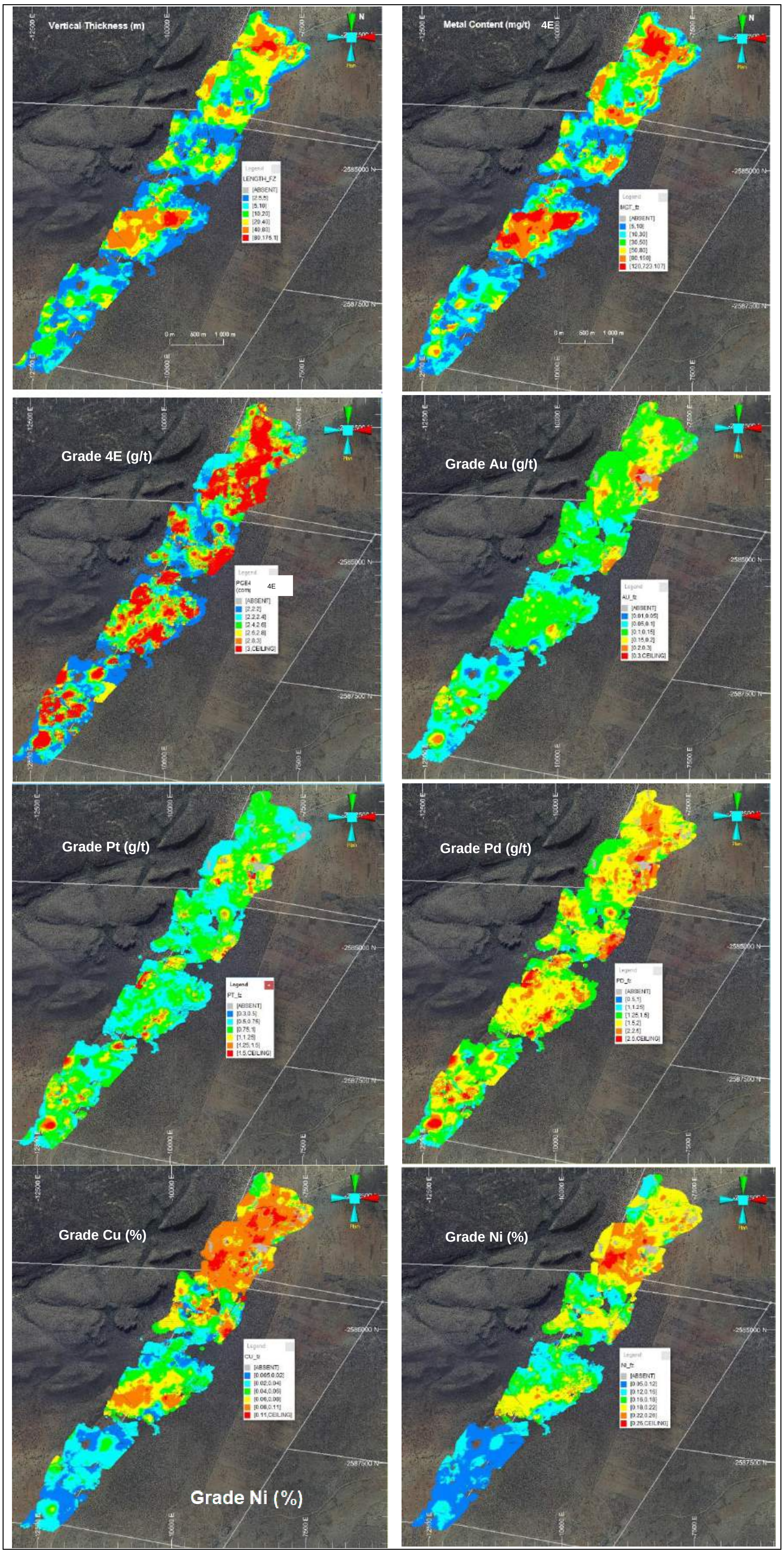


Figure 14-32: Planned Mineral Resource Model Plots (2.0 g/t Cutoff) for F Zone



14.1.20 Metal Groupings and Proportions

4E estimates of Pt, Pd, Au, and Rh are commonly used in Mineral Resource Estimates. The weighted average metal split for the T Zone is Pt:Pd:Rh:Au 29:50:1:20 and the F Zone Pt:Pd:Rh:Au 29:65:1:5.

14.1.21 Effect of Modifying Factors

Modifying factors such as taxation, socio-economic, marketing or political factors were considered 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. 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 Mineral Reserve consideration level. The Mineral Resources may never be classified as Mineral Reserves or be upgraded. These Mineral Resources are utilised in this DFS.

14.2 Mineral Resource Classification Criteria

CJM considers that within the T and F Zones there are areas that can be classified as inferred, indicated, and measured Mineral Resources. The primary criteria differentiating these areas is the spacing of drill hole data, confidence in the kriging estimate (derived from the kriging efficiencies), and regression slope values. Infill drilling increased the confidence in the structure and the perceived continuity of the layering of mineralisation 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 Mineral Resource was classified according to the criteria below.

- Sampling – QA/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 enough data.
  - Indicated: good understanding of geological relationships.
  - Inferred: geological continuity not established.
- Number of samples used to estimate a specific block.
  - Measured: at least 8 drill holes within semi-variogram range and minimum of 27 one m composited samples.

  - Indicated: at least four drill holes within semi-variogram range and a minimum of 12 one m 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 14-33 and Figure 14-34 show the indicated, inferred, and measured Mineral Resource categories for the F and T Zones, respectively.

The classification of the Mineral Resource Estimate was underlain in accordance with requirements and guidelines of the CIM 2014 standard. The Mineral Resource reported here meets the requirements of the current 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.

Figure 14-33: Mineral Resource Categories for the F Zone

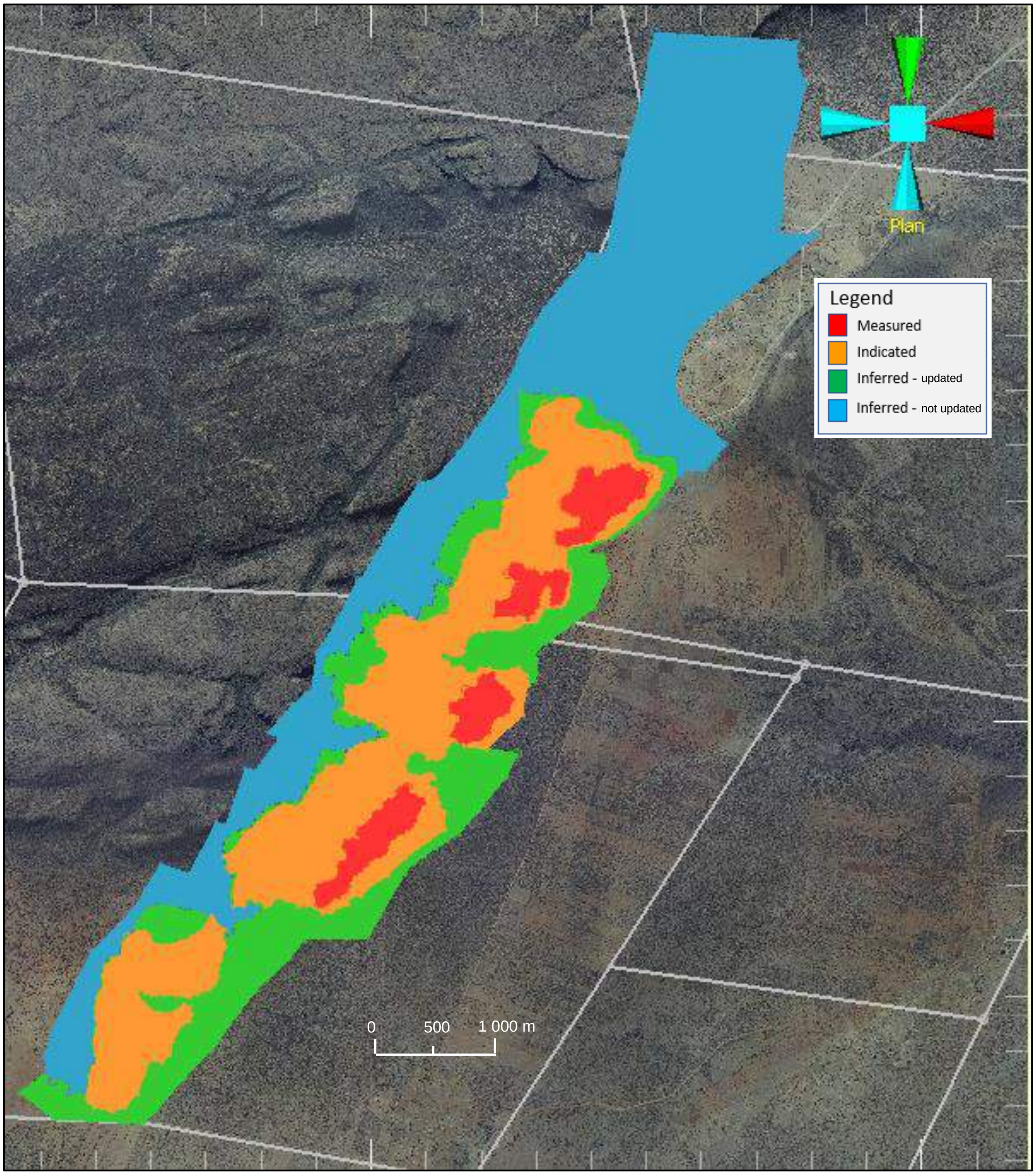
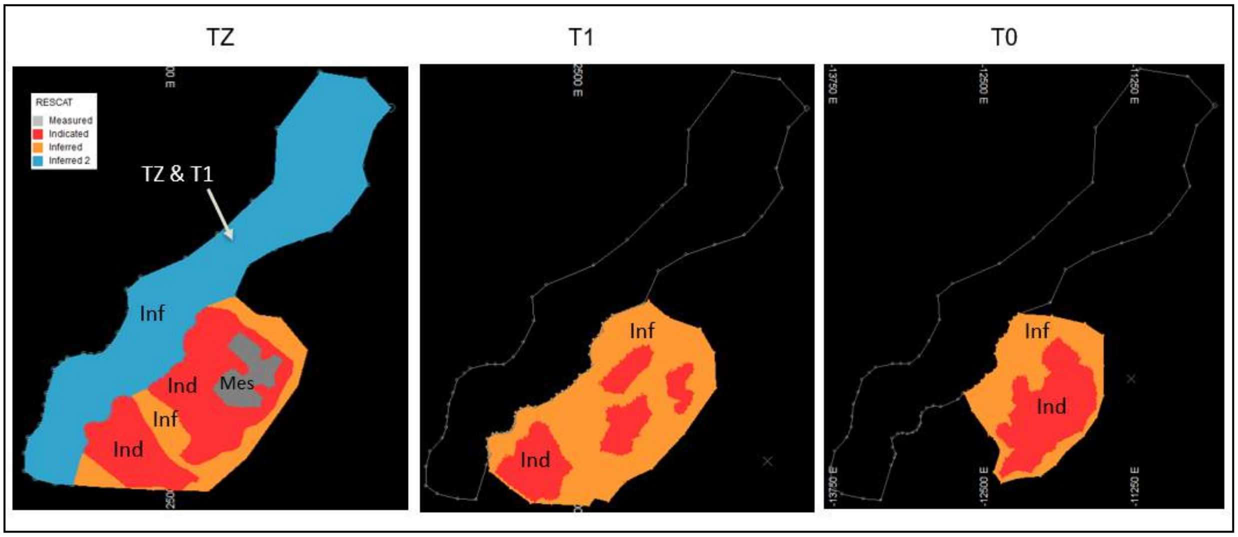


Figure 14-34: Mineral Resource Categories for the TZ, T1, and T0 Zones



14.3 Reasonable Prospects for Eventual Economic Extraction

All the JV partners were involved in developing the latest Mineral Resource Model, appropriate cutoff grades, economic parameters, and Mineral Resource Model criteria. It was determined in relation to basic working costs and in consideration of the overall resource envelope for the deposit, that at a 2.0 g/t cutoff grade, the deposit has a reasonable prospect of economic extraction.

Metal contents and block tonnages were accumulated and formed the basis for reporting the Mineral Resource Estimate. The results are presented in Table 16-11.

Mineral Resources that 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.

No guarantee exists that all or any part of the Mineral Resource will be converted to a Mineral Reserve.

All Mineral Resources were classified as indicated, inferred and measured Mineral Resources, according to the definitions of the CIM Standards.

Inferred Mineral Resources were classified; however, no addition of the inferred Mineral Resources to other Mineral Resource categories took place.

14.4 Mineral Resource Statement

Updated Mineral Resource Estimates were completed for both the F Zone and the T Zone in the project area, incorporating additional and infill drilling since the updates in April 2016 and September 2018. Table 14-7 summarises the updated Mineral Resources for the T Zone and the F Zone at a 2.0 g/t (4E) and 2.5 g/t cutoff.

All the JV partners were involved in developing the latest Mineral Resource Model, appropriate cutoff grades, economic parameters, and Mineral Resource Model criteria. It was determined in relation to basic working costs and in consideration of the overall resource envelope for the deposit, that at a 2.0 g/t cutoff grade, the deposit has a reasonable prospect of economic extraction. For purposes of the DFS, sensitivity analysis and comparison to the 2016 PFS, which utilised a 2.5 g/t 4E cutoff grade, a Mineral Resource Estimate at a 2.5 g/t cutoff grade is the preferred scenario. The Mineral Resource Statement is summarised in Table 14-7.

The data that formed the basis for the Mineral Resource Estimate was an exploration database containing the details of geological logging and assay values derived from a surface drilling programme.

Based on the available data, a Mineral Resource Estimate was completed. 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.

- Mineral Resources are classified in accordance with the SAMREC (2016) standards. There are certain differences with the "CIM Standards on Mineral Resources and Mineral Reserves;" however, in this case, the company and the QP believe the differences are not material and the standards may be considered the same. Inferred Mineral Resources have a high degree of uncertainty. Mineral Resources might never be upgraded or converted to Mineral Reserves.
- Mineral Resources are provided on a 100% project basis. Inferred and Indicated categories are separate. The estimates have an effective date of 04 September 2019. Tables may not add perfectly due to rounding.

Table 14-7: Summary of Mineral Resources Effective 04 September 2019 on a 100% Project Basis

Total T Zone at 2.0 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	4 892 193	1.12	2.01	0.04	0.85	4.02	0.16	0.08	19 667	0.632
Indicated	2.0	21 479 925	1.23	2.09	0.03	0.78	4.13	0.19	0.09	88 712	2.852
M+I	**2.0**	**26 372 118**	**1.21**	**2.08**	**0.03**	**0.79**	**4.11**	**0.18**	**0.09**	**108 379**	**3.484**
Inferred	2.0	25 029 695	1.17	1.84	0.03	0.60	3.64	0.14	0.07	91 108	2.929

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.9	50.0	1.0	21.1
Indicated	29.8	50.6	0.7	18.9
M+I	**29.5**	**50.6**	**0.7**	**19.2**
Inferred	32.1	50.5	0.8	16.6

F Zone at 2.0 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	75 332 513	0.82	2.00	0.05	0.14	3.01	0.08	0.19	226 833	7.293
Indicated	2.0	273 272 480	0.80	1.85	0.04	0.14	2.83	0.07	0.18	772 103	24.824
M+I	**2.0**	**348 604 993**	**0.80**	**1.88**	**0.04**	**0.14**	**2.87**	**0.08**	**0.18**	**998 936**	**32.117**
Inferred	2.0	121 535 227	0.70	1.62	0.04	0.13	2.50	0.07	0.16	303 722	9.765

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.2	66.4	1.7	4.7
Indicated	28.3	65.4	1.4	4.9
M+I	**28.0**	**65.7**	**1.4**	**4.9**
Inferred	28.1	65.1	1.6	5.2

Waterberg Aggregate Total 2.0 g/t Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.0	80 224 706	0.84	2.00	0.05	0.18	3.07	0.08	0.18	246 500	7.925
Indicated	2.0	294 752 405	0.83	1.87	0.04	0.19	2.92	0.08	0.17	860 815	27.676
M+I	**2.0**	**374 977 111**	**0.83**	**1.90**	**0.04**	**0.19**	**2.96**	**0.08**	**0.18**	**1 107 315**	**35.601**
Inferred	2.0	146 564 922	0.78	1.66	0.04	0.21	2.69	0.08	0.15	394 830	12.694

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.3	65.1	1.6	6.0
Indicated	28.4	63.9	1.3	6.4
M+I	**28.1**	**64.3**	**1.3**	**6.3**
Inferred	29.0	61.7	1.5	7.8

Notes:

- 4E = PGE (Pt + Pd + Rh) and Au.
- The cutoffs for Mineral Resources were established by a QP 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 entity.
- Conversion Factor used – kg to oz = 32.15076.
- Numbers may not add due to rounding.
- A 5% and 7% geological loss were applied to the measured / indicated and inferred Mineral Resource categories, respectively.

T Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	4 443 483	1.17	2.12	0.05	0.87	4.20	0.15	0.08	18 663	0.600
Indicated	2.5	17 026 142	1.37	2.34	0.03	0.88	4.61	0.20	0.09	78 491	2.524
M+I	**2.5**	**21 469 625**	**1.34**	**2.29**	**0.03**	**0.88**	**4.53**	**0.19**	**0.09**	**97 154**	**3.124**
Inferred	2.5	21 829 698	1.15	1.92	0.03	0.76	3.86	0.20	0.10	84 263	2.709

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	27.8	50.4	1.2	20.6
Indicated	29.7	50.7	0.6	19.0
M+I	**29.5**	**50.4**	**0.7**	**19.4**
Inferred	29.8	49.7	0.8	19.7

F Zone at 2.5 g/t (4E) Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	54 072 600	0.95	2.20	0.05	0.16	3.36	0.09	0.20	181 704	5.842
Indicated	2.5	166 895 635	0.95	2.09	0.05	0.15	3.24	0.09	0.19	540 691	17.384
M+I	**2.5**	**220 968 235**	**0.95**	**2.12**	**0.05**	**0.15**	**3.27**	**0.09**	**0.19**	**722 395**	**23.226**
Inferred	2.5	44 836 851	0.87	1.92	0.05	0.14	2.98	0.06	0.17	133 705	4.299

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.3	65.4	1.5	4.8
Indicated	29.3	64.4	1.6	4.7
M+I	**29.1**	**64.8**	**1.5**	**4.6**
Inferred	29.2	64.4	1.7	4.7

Waterberg Aggregate Total 2.5 g/t Cutoff											
Mineral Resource Category	Cutoff	Tonnage	Grade							Metal	
	4E		Pt	Pd	Rh	Au	4E	Cu	Ni	4E	
	g/t	t	g/t	g/t	g/t	g/t	g/t	%	%	kg	Moz
Measured	2.5	58 516 083	0.97	2.19	0.05	0.21	3.42	0.09	0.19	200 367	6.442
Indicated	2.5	183 921 777	0.99	2.11	0.05	0.22	3.37	0.10	0.18	619 182	19.908
M+I	**2.5**	**242 437 860**	**0.98**	**2.13**	**0.05**	**0.22**	**3.38**	**0.10**	**0.18**	**819 549**	**26.350**
Inferred	2.5	66 666 549	0.96	1.92	0.04	0.34	3.27	0.11	0.15	217 968	7.008

Mineral Resource Category	Prill Split			
	Pt	Pd	Rh	Au
	%	%	%	%
Measured	28.2	64.0	1.5	6.3
Indicated	29.4	62.6	1.5	6.5
M+I	**29.1**	**63.0**	**1.5**	**6.4**
Inferred	29.5	58.9	1.2	10.4

Notes:

- 4E = PGE (Pt + Pd + Rh) and Au.
- The cutoffs for Mineral Resources were established by a QP 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 entity.
- Conversion Factor used – kg to oz = 32.15076.
- Numbers may not add due to rounding.
- A 5% and 7% geological loss were applied to the measured/indicated and inferred Mineral Resource categories, respectively.

- A cutoff grade of 2.0 g/t and 2.5 g/t 4E (Pt, Pd, Rh, and Au) is applied to the selected Base Case Mineral Resources.
- Cutoff grade for the T Zone and the F Zone considered costs, smelter discounts, and 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 were considered for the cutoff considerations. The upper and lower bound metal prices used in the determination of cutoff grade for resources estimated are as follows: US$983/oz-US$953/oz Pt, US$993/oz-US$750/oz Pd, US$1 325/oz-US$1 231/oz Au, US$1 923US/oz-US$972/oz Rh, US$6.08/lb-US$4.77/lb Ni, US$3.08/lb-US$2.54/lb Cu, US$/ZAR15-US$/ZAR12. These metal prices were based on the estimated 3-year trailing average prices and the spot prices at the time of commencement of the Mineral Resource Estimate modelling. The lower cutoff was tested against the higher metal price in the range and the higher cutoff was tested against the lower price in the range.

The objective of the cutoff grade estimation was to establish a minimum grade for working break even. From the PFS, the following factors were used for the calculation of cutoff at 2.0 g/t (4E) at higher potential prices and 2.5 g/t 4E at more conservative lower prices listed above.

- Working cost mining of US$25.00, ZAR 379 per tonne, LOM average. Total OpEx US$38, ZAR 574 average.
- 80 g/t concentrate 82% recoveries of the PGMs, 88% of the Cu and 49% of the Ni.
- 85% payability of the PGMs from a third-party smelter, 73% for Cu and 68% for Ni.

These costs recoveries and payabilities were updated in the DFS for the consideration of Mineral Reserves (see Section 15 for the Mineral Reserve estimate). Metallurgical work indicates that an economically attractive concentrate can be produced from standard flotation methods.

- Mineral Resources were completed by Charles Muller of CJM and a NI 43-101 technical report for the Mineral Resources reported herein, effective 04 September 2019.
- Mineral Resources were estimated using OK and SK methods in Datamine Studio3 from 4 441 mother holes and 585 deflections in mineralisation. A process of geological modelling and creation of grade shells using indicating kriging was completed in the estimation process.

- The estimation of Mineral Resources has considered environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors. The Mineral Resources may be materially affected by metals prices, exchange rates, labour costs, electricity supply issues, or many other factors detailed in the Company's Annual Information Form.
- Estimated grades and quantities for byproducts are included in recoverable metals and estimates in this DFS work. Cu and Ni are the main value by-products recoverable by flotation and for M&I Mineral Resources are estimated at 0.18% Cu and 0.09% Ni in the T Zone and 0.08% Cu and 0.18% Ni in the F Zone.

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

The independent QP responsible for the Mineral Resource Estimate in this report is Charles Muller. Mr. Muller is a geologist with over 30 years' experience in mine and exploration geology, Mineral Resource and Mineral Reserve estimation, and project management in the minerals industry (especially Pt and Au). He is a practicing geologist registered with the South African Council for Natural Scientific Professions and is independent of PTM and Waterberg JV Resources as that term is defined in Section 1.5 of the Instrument.

14.5 Mineral Resource Reconciliation

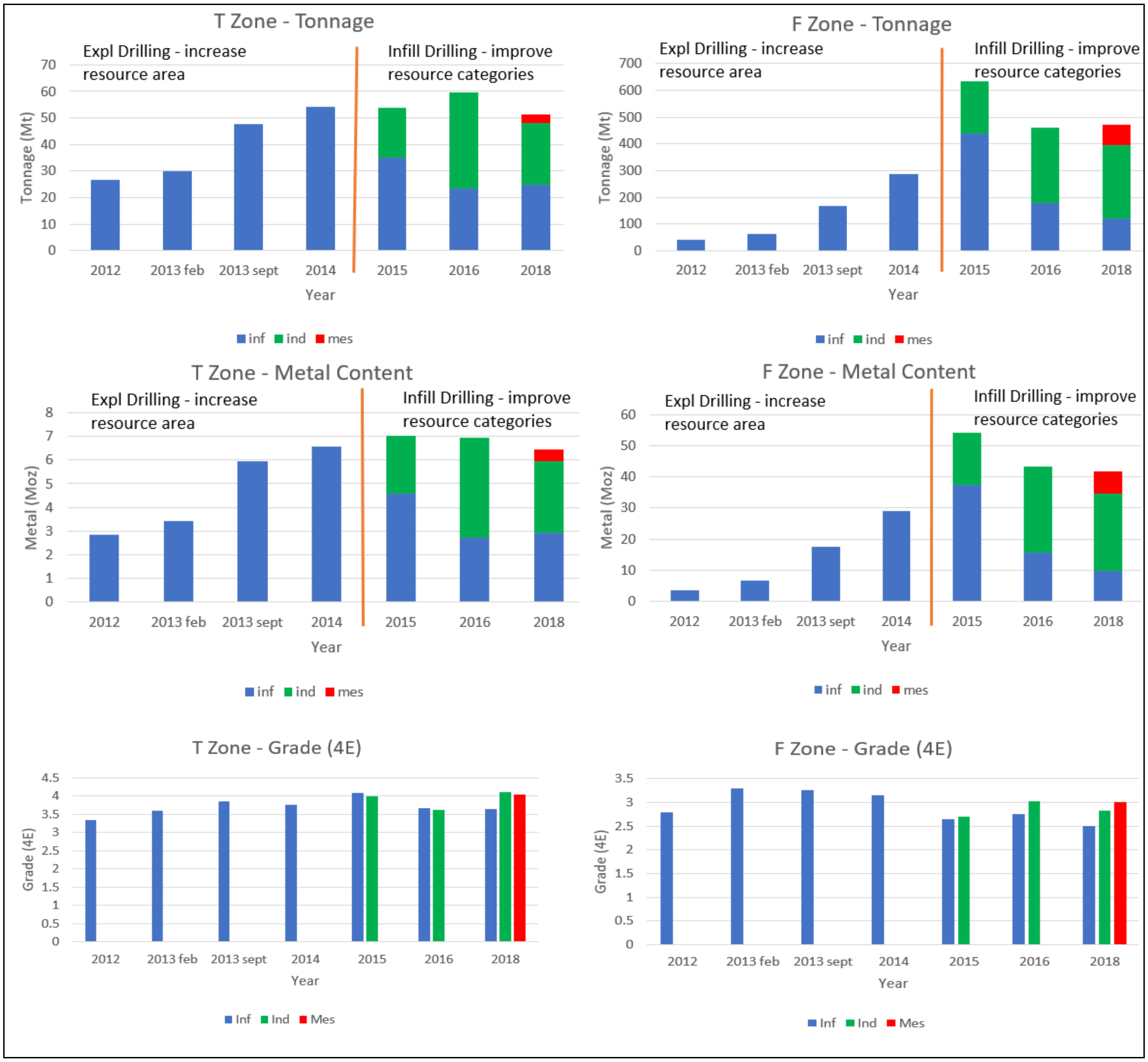
The initial inferred Mineral Resource was declared in September 2012 for the T and F Zone mineralisation.

The period up to 2014 was mainly aimed at increasing the Mineral Resource area. From 2015, the aim was to improve on the Mineral Resource categories or confidence by infill drilling as shown in Figure 14-35.

The 2018 T Zone tonnage decreased by 14% compared to 2016 as shown in Figure 14-35. This is mainly due to the introduction of mining modifying factors for the Mineral Resource categories (i.e., minimum width, elimination of scattered mineralisation, and continuous zones at specific cutoffs). The F Zone showed an overall 2% increase in tonnage from 2016 to 2018. The large decrease in tonnes for the F Zone from 2015 to 2016 is due to a stricter delineation of the inferred category. The indicated category for that period increased significantly, showing greater confidence in the 2016 model.

The metal content (4E) decreased slightly, less than 5% for the project from 2016 to 2018 period as shown in Figure 14-35. The grade (4E) shows higher values from 2016 to 2018 period, especially in the more confident indicated and measured categories.

Figure 14-35: Mineral Resource Statements for the Period 2012 to 2018



The recent updated Mineral Resource Estimate as at 04 September 2019 effective only impacted on the proportion of Measured Mineral Resources for the T Zone.

15 MINERAL RESERVE ESTIMATES

15.1 Resource to Reserve Calculation

The Waterberg Project Mineral Reserve Estimate was based on the M&I Mineral Resource material contained in the resource block models prepared by CJM. The M&I Mineral Resources targeted in the mine design are contained in the T Zone and Super F Zone (F Zone). The F Zone is comprised of the five sub-zones listed below.

- Super F-South Zone (F-South)
- Super F-Central Zone (F-Central)
- Super F-North Zone (F-North)
- Super F-Boundary North Zone (F-Boundary North)
- Super F-Boundary South Zone (F-Boundary South)

15.1.1 Cutoff Grade

The stoping pay limit calculation was based on April 2018 metals spot prices and costs, metal recovery, smelter recovery, and dilution estimates from previous engineering work completed on the property. The inputs to the cutoff estimate are summarised in Table 15-1.

Table 15-1: Mine Planning 4E Cutoff Grade Inputs

Input	Central Zone	T Zone
Exchange Rate (ZAR / US$)	13.00	13.00
4E Basket Price (US$ / oz)	1 009.00	1 062.00
Cu & Ni Revenue (US$ / oz)	7.90	7.70
Total Production Costs (US$ / t)	56.00	60.00
Metal Recovery (%)	82.00	81.00
Smelter Recovery (%)	85.00	85.00
Dilution (0.0 g/t) (%)	2.50	4.70
4E Stoping Pay Limit (g/t)	2.19	2.33

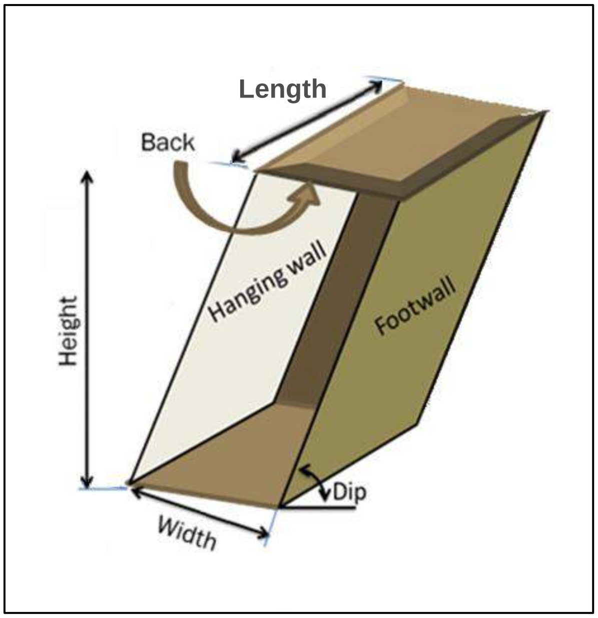
Based on these estimates, a 2.5 g/t 4E stope cutoff grade was used for mine planning to estimate the Mineral Reserves.

15.1.2 Stope Shape Design

The mine design is based on using the sublevel longhole stoping mining method (longhole) to extract the reserves. Details of the mine design are included in Section 16.

Figure 15-1 shows the terminology associated with longhole stoping.

Figure 15-1: Longhole Stoping Terminology



Mining stope shapes were created using specialty mine design software MSO. Numerous iterations of MSO were run to determine the optimal orientation of the stopes to maximize resource extraction. The MSO parameters used to create the stope shapes are shown in Table 15-2.

Table 15-2: Mineable Shape Optimiser Parameters

Parameter	Value
Stope Cutoff Grade	2.5 g/t 4E
Orientation of MSO	Northwest
Stope Length along Strike	20 m
Stope Height	20 or 40 m
Minimum Stope Width Horizontal	3.8 m
Minimum Stope Middling Horizontal	20 m
Minimum Stope Dip Angle	38^{o}

15.1.3 Modifying Factors

Modifying factors include geological losses, planned dilution, external dilution, and mining losses. The following subsections describe the modifying factors and the application of the factors to the mine design.

Geological Losses

Geological losses are anticipated to occur and have been accounted for in the reserves. The *in situ* stope tonnes and metals queried from the block models were discounted by 5% to account for geological losses.

Planned Dilution

Bulk mining methods such as longhole typically capture material below the cutoff grade in the stopes. Planned dilution is material below the 2.5 g/t 4E cutoff grade that is contained within the stope shapes and mined along with material above cutoff. This planned dilution is included in the Mineral Reserve Estimates.

External Overbreak Dilution

External overbreak dilution is material that is outside the stope shape but will overbreak into the stope and mined with the stope. This external dilution is included in the Mineral Reserve Estimates. To calculate external dilution tonnage, the following parameters for overbreak of the footwall and hanging wall and where applicable the paste backfill overbreak of the side / end walls and back were used. There are different overbreak rules applied to the following stope types.

- Type 1 – 40 m high (H) x 20 m length primary transverse stope.
- Type 2 – 20 m H x 20 m length primary transverse stope.
- Type 3 – 40 m H x 20 m length x <40 m wide (W) secondary transverse stope.
- Type 4 – 20 m H x 20 m length x <40 m W secondary transverse stope.
- Type 5 – 40 m H x 20 m length x >40 m W secondary transverse stope less than 1 000 m below surface.
- Type 6 – 20 m H x 20 m length x >40 m W secondary transverse stope less than 1 000 m below surface.
- Type 7 – 40 m H x 20 m length x >40 m W secondary transverse stope 1 000 m or greater below surface.
- Type 8 – 20 m H x 20 m length x >40 m W secondary transverse stope 1 000 m or greater below surface.
- Type 9 – 40 m H x 20 m length longitudinal stope.
- Type 10 – 20 m H x 20 m length longitudinal stope.

External dilution was estimated based on average overbreak depths. All stope types will have a combined footwall and hanging wall dilution of 0.9 m of overbreak. Type 1 and Type 2 primary stopes side wall overbreak will typically be ore and is not calculated as part of the dilution. This overbreak ore is assumed to be included in either the primary or secondary stopes. Type 2 transverse primary stopes will be mined below paste backfilled stopes from the mining block above and include 0.3 m of paste backfill dilution from the back.

Transverse secondary stopes will have side wall dilution as they will be mined adjacent to paste backfill side walls from the primary stopes. Type 3 transverse secondary stopes will have 0.6 m of overbreak in the side walls (i.e. 0.3 m for each side wall). Type 4 transverse secondary stopes will have 0.15 m of overbreak in each side wall and 0.3 m of overbreak in the back.

Transverse secondary stopes greater than 40 m in width from hanging wall to footwall have additional side wall paste backfill dilution. Type 5 and Type 6 secondary stopes are less than 1 000 m below surface. Type 5 secondary stopes have 0.8 m of paste backfill overbreak in each side wall and Type 6 secondary stopes have 0.4 m of paste backfill overbreak in each side wall and 0.8 m of overbreak in the back. Type 7 and Type 8 secondary stopes are greater than 1 000 m below surface. Type 7 secondary stopes have 1.0 m of paste backfill overbreak in each side wall and Type 8 secondary stopes have 0.5 m of paste backfill overbreak in each side wall and 1.0 m of overbreak in the back. Figure 15-2 shows an isometric view of a typical transverse primary and secondary stoping area.

Longitudinal stopes will be mined adjacent to paste backfill end wall from the previous stope. Type 9 longitudinal stopes will have a dilution of 0.3 m of paste backfill overbreak on one end wall. The second end wall overbreak is assumed to be ore and will not be calculated as part of the dilution. Type 10 longitudinal stopes will have a dilution of 0.15 m of overbreak on one side wall and 0.15 m in the back of the stope. Table 15-3 summarises the overbreak depths by stope type. Table 15-4 summarises the overbreak percentages by zone.

To generate an appropriate grade for rock dilution outside of the stope shapes, a 1.0 m thick tabular shape was created on the footwall and hanging wall of the stopes. This 1.0 m thick shape was used to query metal grades from the resource block models and additional hanging wall and footwall block models prepared by CJM. These evaluations were used to estimate an external dilution grade for hanging wall and footwall overbreak for each of the six zones as summarised in Table 15-5.

Zero grade was assigned to the paste backfill dilution.

Figure 15-2: Transverse Stoping Isometric View

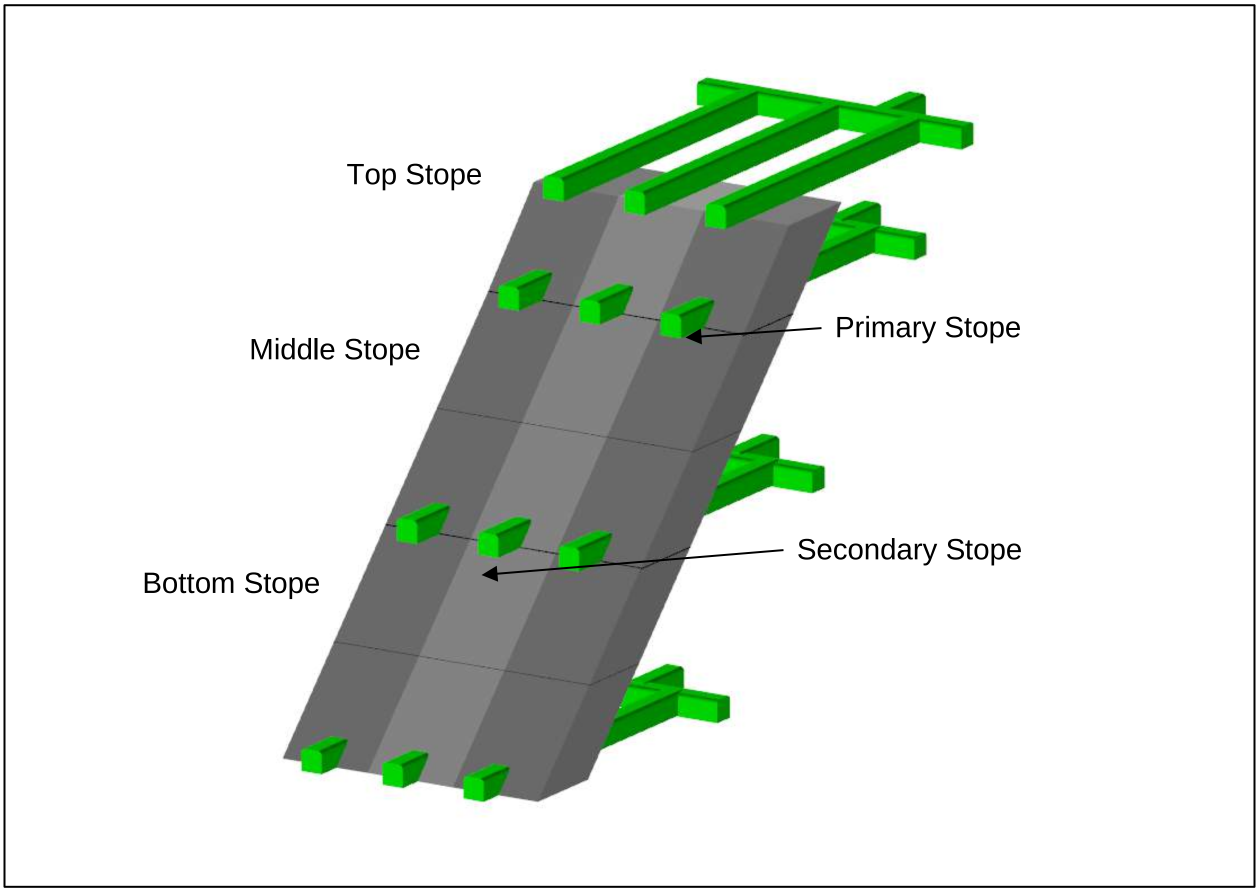


Table 15-3: Longhole Stope Overbreak Dilution Depths in Metres

Overbreak Source	Type 1	Type 2	Type 3	Type 4	Type 5	Type 6	Type 7	Type 8	Type 9	Type 10
Hanging wall and Footwall combined	0.9 m	0.9 m	0.9 m	0.9 m	0.9 m	0.9 m	0.9 m	0.9 m	0.9 m	0.90 m
Side or End Wall	0.0 m	0.0 m	0.6 m	0.3 m	1.6 m	0.8 m	2.0 m	1.0 m	0.3 m	0.15 m
Back	0.0 m	0.3 m	0.0 m	0.3 m	0.0 m	0.8 m	0.0 m	1.0 m	0.0 m	0.15 m

Table 15-4: Longhole Stope Overbreak Dilution Percentage

Zone	Overbreak
T Zone	16.9%
F-Central	8.3%
F-South	14.1%
F-North	7.2%
F-Boundary North	9.9%
F-Boundary South	12.7%
F Zone Total	**9.0%**

Table 15-5: Dilution Grades

Zone	Pd (g/t)	Pt (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)	Cu (%)	Ni (%)
T Zone	0.86	0.50	0.35	0.02	1.73	0.07	0.05
F-Central	1.45	0.64	0.10	0.04	2.23	0.05	0.16
F-South	1.48	0.79	0.11	0.04	2.42	0.03	0.11
F-North	1.31	0.58	0.11	0.03	2.03	0.06	0.15
F-Boundary North	1.44	0.82	0.12	0.04	2.42	0.06	0.19
F-Boundary South	1.61	0.78	0.12	0.04	2.55	0.06	0.16

Mining Losses

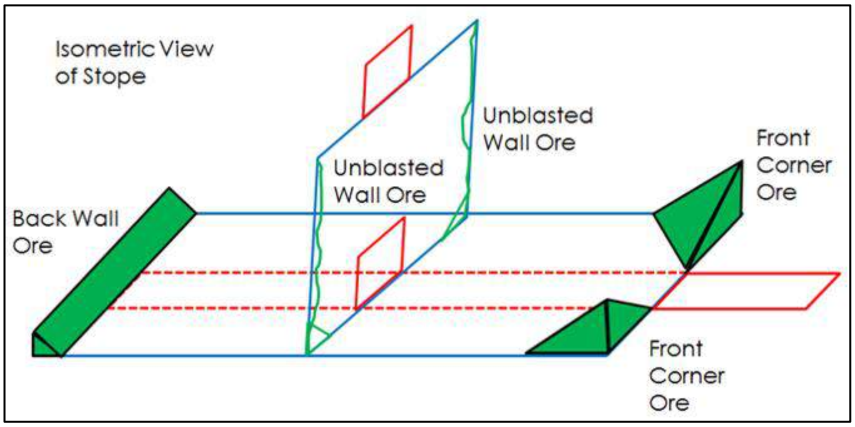
Mining losses account for Mineral Resource that is planned to be mined but will not be recovered due to losses that occur throughout the mining process.

Mining losses in development drifts in ore for longhole stope sills and crosscuts is assumed be zero as any unrecovered development ore will be extracted and included as part of the longhole stope.

Mining losses from longhole stopes was estimated based on an average stope size. Several factors influence mining losses such as mucking line of sight, depth of sight, possible hang ups on the footwall, and blast complications.

It is expected that some ore that is blasted will not be recovered. Line of sight and maneuverability will prevent the load haul dumps (LHDs) from accessing muck from the front corners of the stope. It is assumed that the maximum angle the LHD will be able to operate from the drawpoint will be approximately 45°. Also, cleanup at the back of the stope will be difficult to gauge and result in additional lost ore recovery. Some of the unblasted ore in the side walls may be recoverable with the adjacent stope. Figure 15-3 shows some of the mining losses in a stope.

Figure 15-3: Mining Losses in a Stope

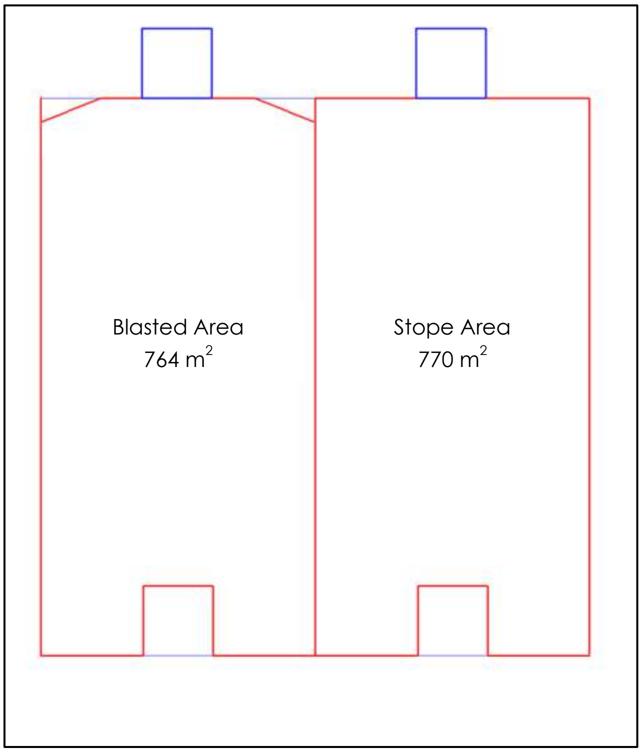


Production blasting in large excavations presents issues that affect ore recovery such as blasted ore left on the footwall, oversized blocks, and unblasted ore left in the walls. This unblasted wall ore could be in the footwall, hanging wall, side wall of stopes with adjacent stopes already mined, or in the side wall of stopes that have no adjacent stopes. Side wall unblasted ore may be recoverable if the adjacent stope has not yet been mined.

Design is another factor in determining how much ore is recovered from a stope. The designed blasted shape does not necessarily recover all the ore. Restrictions on the design drill and blast may have a slight difference in shape when compared with the planned stope to ensure the stope shoulders stay in place for drift re-entry – refer to Figure 15-4.

Mucking complications, blasting limitations, and other unplanned ore losses result in an overall mining loss from longhole stopes of 10%.

Figure 15-4: Blasted Stope Outline



Summary of Modifying Factors

Following is the final Mineral Reserve equation.

> *stope ore tonnes = <in situ stope tonnage> - <geological losses tonnage> + <overbreak dilution tonnage> - <mining loss tonnage>*

The *in situ* stope tonnage is the total tonnage, including planned dilution, in the stope shape and is determined directly from the resource block model evaluation. The geological losses tonnage is 5% of the *in situ* stope tonnage. The overbreak dilution tonnage is calculated for each individual stope according to the criteria discussed in Section 15.1.3. The mining loss is 10%.

15.2 Mineral Resource Conversion

The Mineral Resource is converted into a Mineral Reserve according to a basic mining equation. The Mineral Reserve is made up of M&I material and excludes Mineral Resource material above the Mineral Resource cutoff grade that could not be included in a stope shape above cutoff and is outside the MSO design. In addition, there is some M&I material above cutoff contained in the resource block models that that is outside the resource envelope but was included in the stope shape MSO design. The Mineral Resource to Mineral Reserve conversion is shown in Table 15-6 to Table 15-12 and depicted in waterfall charts in Figure 15-5 to Figure 15-11.

Table 15-6: T Zone Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (oz)
Mineral Resource	21 469 625	4.53	3 124 000
Outside MSO Design	-8 046 762	3.60	-930 347
M&I in MSO Design but Outside Resource	457 380	4.49	66 010
Low Grade Planned Dilution	2 838 632	1.44	131 782
Geological Losses	-771 426	4.26	-105 621
Overbreak Dilution	2 687 581	1.60	138 663
Mining Losses	-1 734 467	4.05	-225 661
Mineral Reserve	16 900 564	4.05	2 198 826

Figure 15-5: T Zone Resource Conversion Tonnage Waterfall

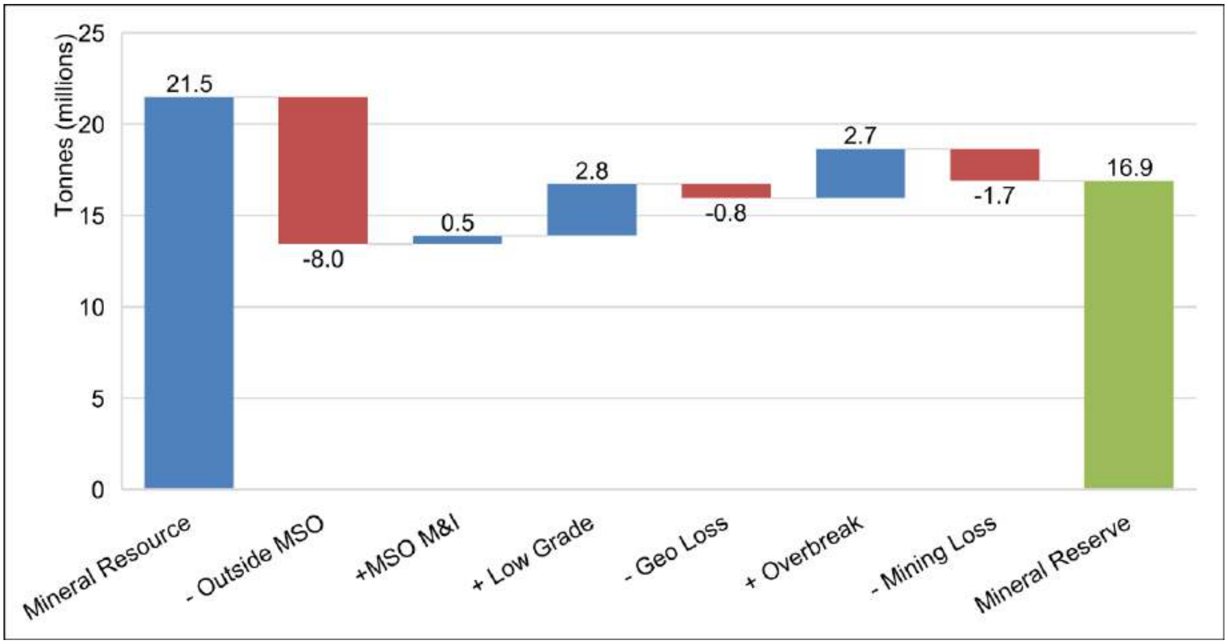


Table 15-7: F Zone Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (ounces)
Mineral Resource	220 968 235	3.27	23 225 527
Outside MSO Design	-65 407 001	2.89	-6 084 823
M&I in MSO Design but Outside Resource	2 963 660	2.96	282 289
Low-grade Planned Dilution	23 690 773	2.17	1 655 727
Geological Losses	-8 525 208	3.17	-869 899
Overbreak Dilution	15 606 846	1.93	965 941
Mining Losses	-18 690 812	3.15	-1 891 817
Mineral Reserve	170 606 492	3.15	17 282 945

Figure 15-6: F Zone Resource Conversion Tonnage Waterfall

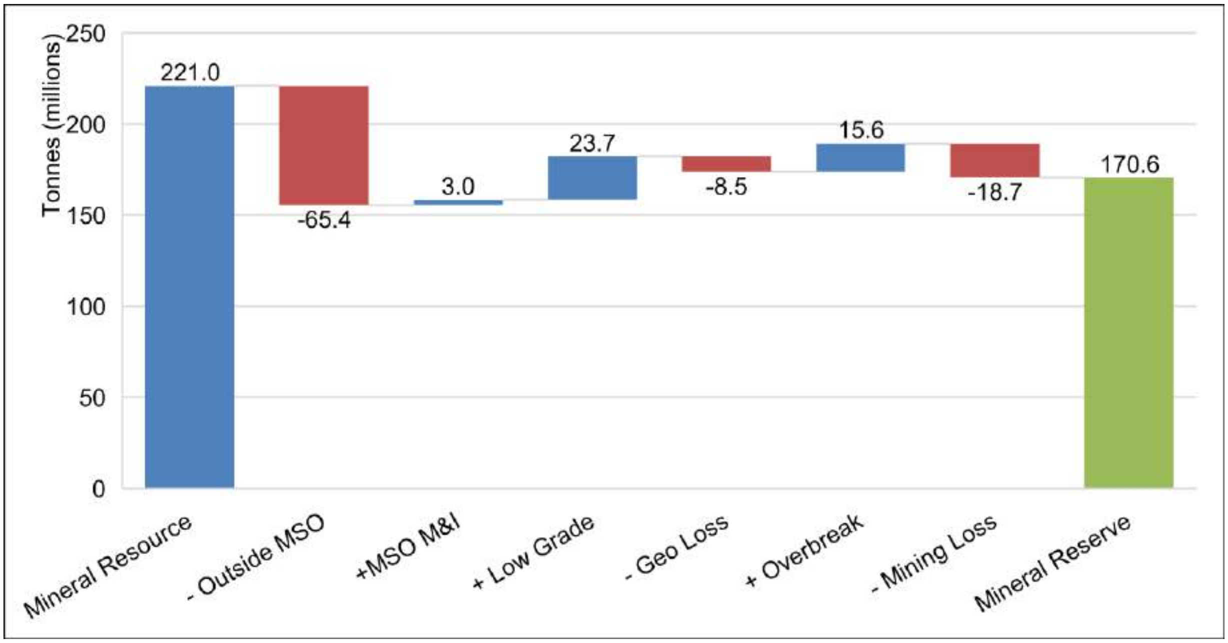


Table 15-8: F-Central Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (ounces)
Mineral Resource	94 575 339	3.22	9 789 447
Outside MSO Design	-30 095 505	2.91	-2 813 203
M&I in MSO Design but Outside Resource	1 172 799	2.88	108 656
Low-grade Planned Dilution	10 105 241	2.12	687 684
Geological Losses	-3 547 673	3.12	-355 923
Overbreak Dilution	6 029 133	1.77	343 281
Mining Losses	-8 107 997	3.08	-804 168
Mineral Reserve	70 131 337	3.08	6 955 773

Figure 15-7: F-Central Resource Conversion Tonnage Waterfall

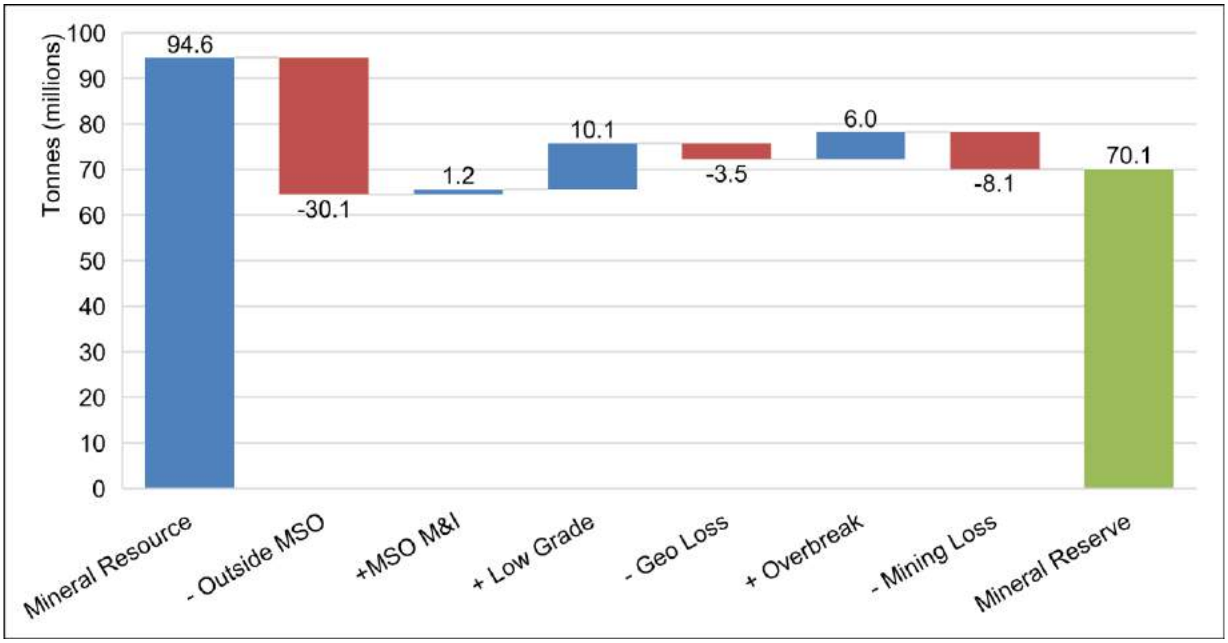


Table 15-9: F-South Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (ounces)
Mineral Resource	20 626 503	3.50	2 320 993
Outside MSO Design	-7 804 436	3.22	-807 880
M&I in MSO Design but Outside Resource	451 887	3.25	47 214
Low-grade Planned Dilution	2 589 516	2.23	185 430
Geological Losses	-734 576	3.27	-77 139
Overbreak Dilution	2 134 181	2.25	154 187
Mining Losses	-1 609 113	3.28	-169 906
Mineral Reserve	15 653 961	3.28	1 652 900

Figure 15-8: F-South Resource Conversion Tonnage Waterfall

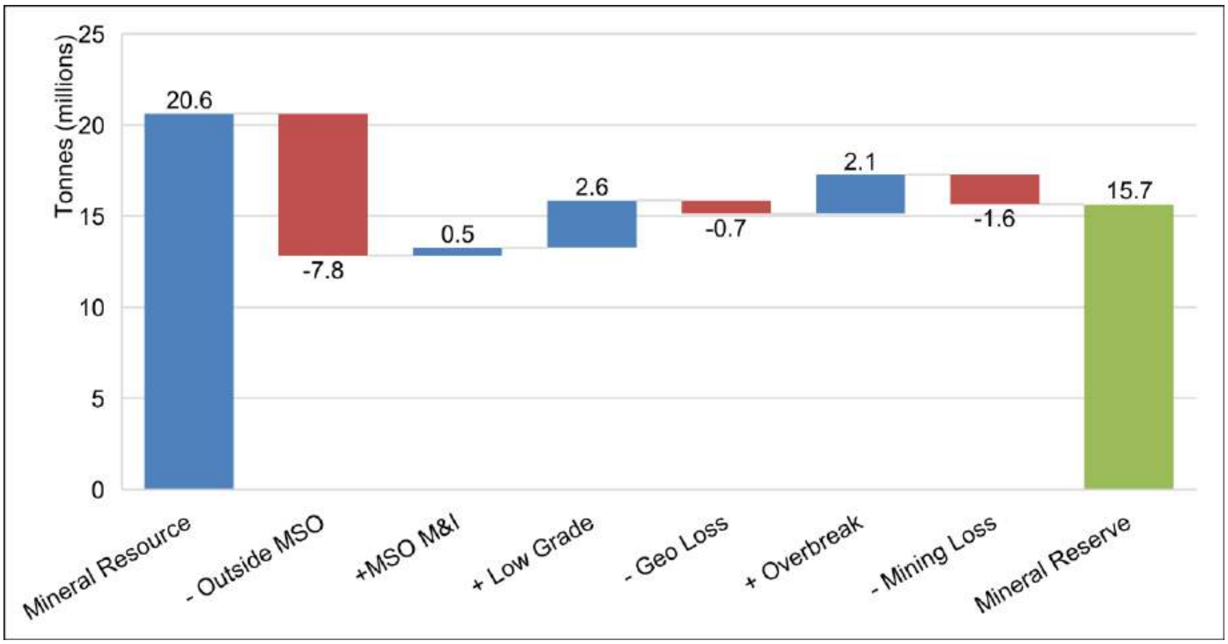


Table 15-10: F-North Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (ounces)
Mineral Resource	62 461 067	3.25	6 532 964
Outside MSO Design	-10 922 743	2.46	-864 851
M&I in MSO Design but Outside Resource	761 940	2.94	72 100
Low-grade Planned Dilution	5 819 786	2.18	408 665
Geological Losses	-2 738 643	3.23	-284 552
Overbreak Dilution	4 012 933	1.70	219 152
Mining Losses	-5 772 441	3.19	-591 244
Mineral Reserve	53 621 900	3.19	5 492 236

Figure 15-9: F-North Resource Conversion Tonnage Waterfall

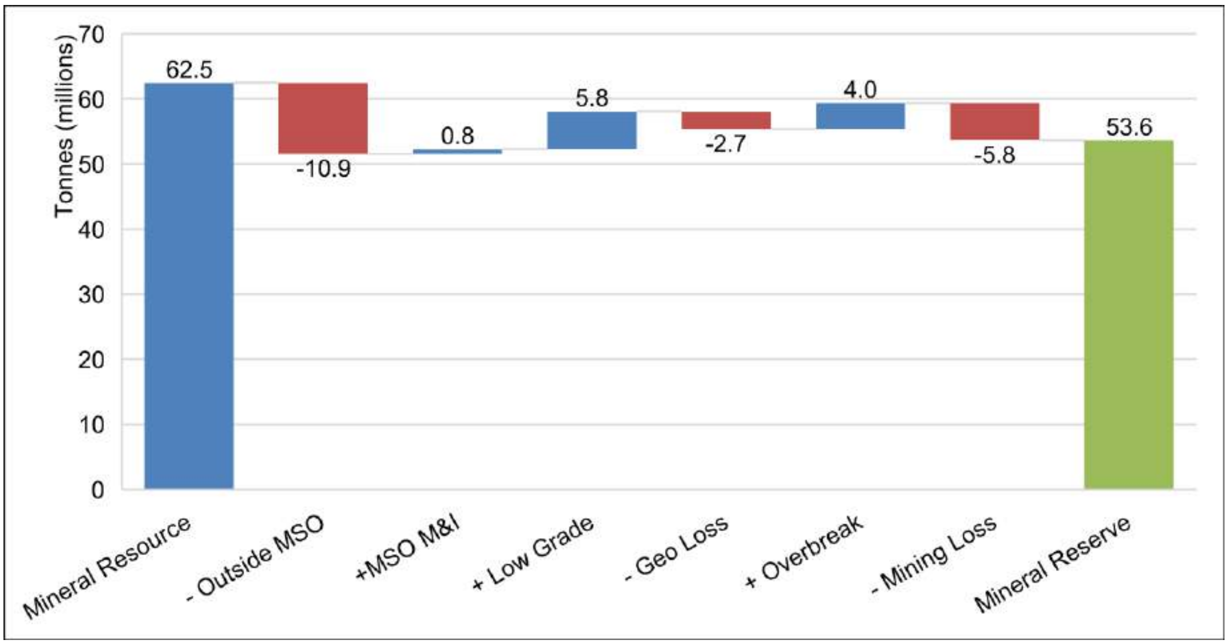


Table 15-11: F-Boundary North Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (ounces)
Mineral Resource	24 160 158	3.31	2 569 004
Outside MSO Design	-8 531 489	3.09	-847 801
M&I in MSO Design but Outside Resource	345 903	2.92	32 462
Low-grade Planned Dilution	3 259 961	2.24	235 106
Geological Losses	-891 734	3.11	-89 162
Overbreak Dilution	1 822 143	2.20	128 680
Mining Losses	-1 876 508	3.13	-188 748
Mineral Reserve	18 288 434	3.13	1 839 540

Figure 15-10: F-Boundary North Resource Conversion Tonnage Waterfall

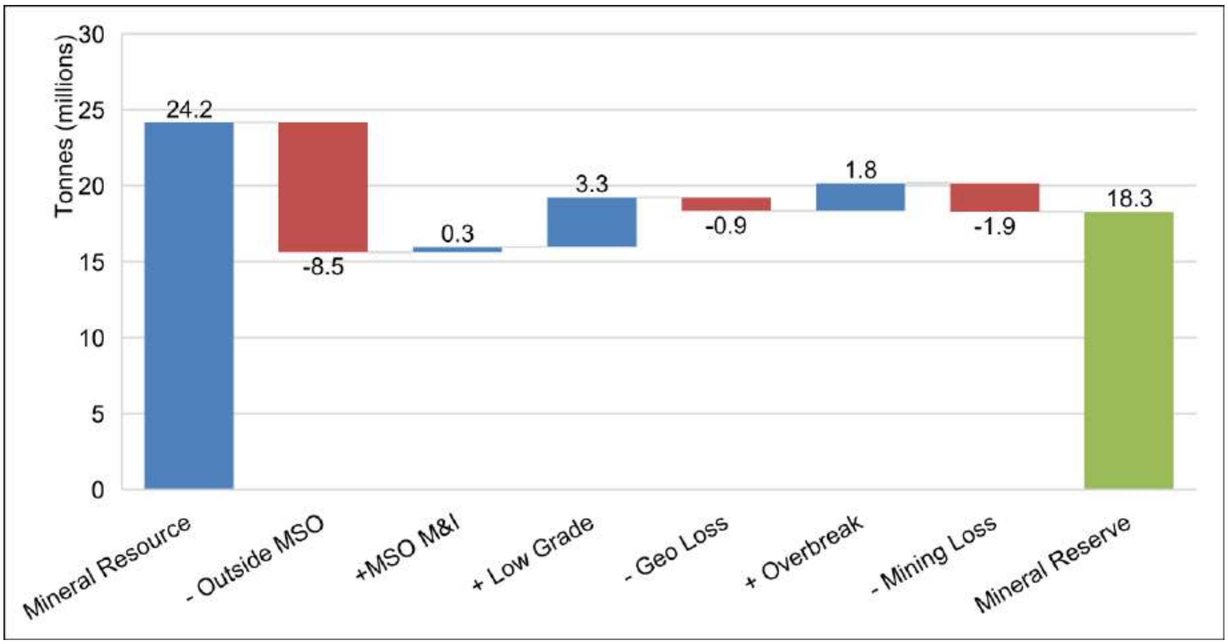
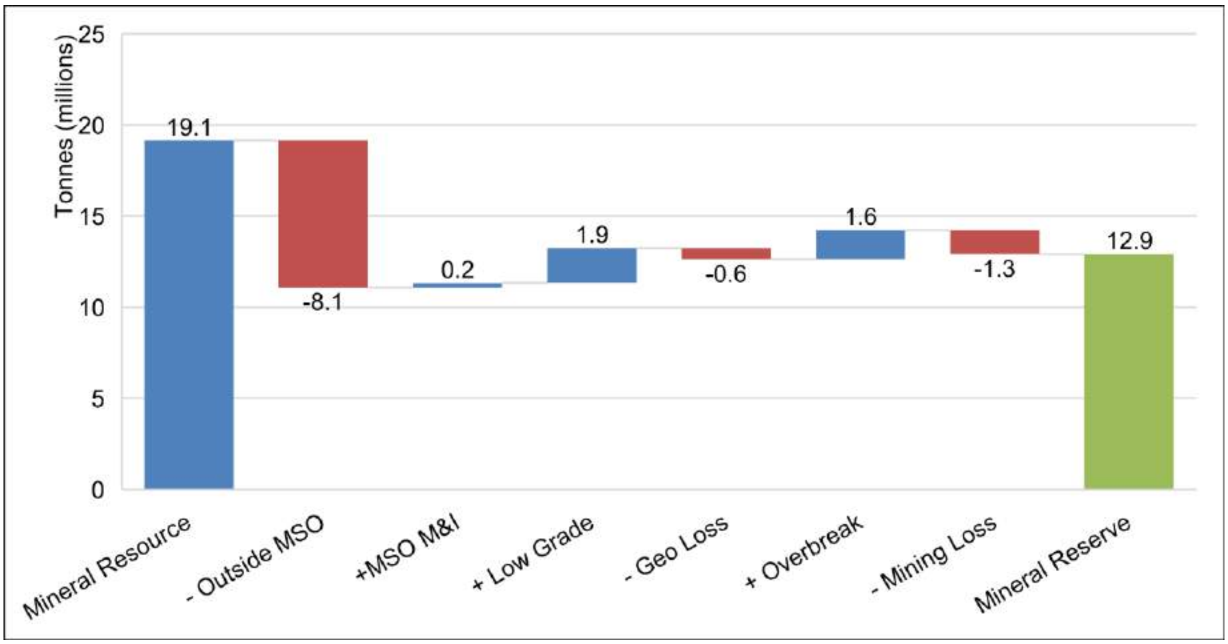


Table 15-12: F-Boundary South Mining Equation Resource Conversion

	Tonnes	4E (g/t)	4E (oz)
Mineral Resource	19 145 168	3.27	2 013 118
Outside MSO Design	-8 052 828	2.90	-751 089
M&I in MSO Design but Outside Resource	231 131	2.94	21 858
Low-grade Planned Dilution	1 916 269	2.25	138 842
Geological Losses	-612 583	3.21	-63 123
Overbreak Dilution	1 608 455	2.33	120 640
Mining Losses	-1 324 753	3.23	-137 750
Mineral Reserve	12 910 859	3.23	1 342 496

Figure 15-11: F-Boundary South Resource Conversion Tonnage Waterfall



15.3 Mineral Reserve Statement

Table 15-13 to Table 15-15 show the estimated proven, probable, and total Mineral Reserves at 2.5 g/t 4E cutoff effective as of 04 September 2019.

The prill splits on Mineral Reserves at a 2.5 g/t 4E cutoff and the additional grade contribution of Cu and Ni are summarised in Table 15-16.

Table 15-13: Proven Mineral Reserve Estimate at 2.5 g/t 4E Cutoff effective 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**3 963 694**	**1.02**	**1.84**	**0.04**	**0.73**	**3.63**	**0.13**	**0.07**	**14 404**	**0.463**
F-Central	17 411 606	0.94	2.18	0.05	0.14	3.31	0.07	0.18	57 738	1.856
F-South	0	0	0	0	0	0	0	0	0	0.000
F-North	16 637 670	0.85	2.03	0.05	0.16	3.09	0.10	0.20	51 378	1.652
F-Boundary North	4 975 853	0.97	2.00	0.05	0.16	3.18	0.10	0.22	15 847	0.509
F-Boundary South	5 294 116	1.04	2.32	0.05	0.18	3.59	0.08	0.19	19 020	0.611
F Zone Total	**44 319 244**	**0.92**	**2.12**	**0.05**	**0.16**	**3.25**	**0.09**	**0.20**	**143 982**	**4.629**
Waterberg Total	**48 282 938**	**0.93**	**2.10**	**0.05**	**0.20**	**3.28**	**0.09**	**0.19**	**158 387**	**5.092**

Table 15-14: Probable Mineral Reserve Estimate at 2.5 g/t 4E Cutoff effective 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**12 936 870**	**1.23**	**2.10**	**0.02**	**0.82**	**4.17**	**0.19**	**0.09**	**53 987**	**1.736**
F-Central	52 719 731	0.86	1.97	0.05	0.14	3.02	0.07	0.18	158 611	5.099
F-South	15 653 961	1.06	2.03	0.05	0.15	3.29	0.04	0.13	51 411	1.653
F-North	36 984 230	0.90	2.12	0.05	0.16	3.23	0.09	0.20	119 450	3.840
F-Boundary North	13 312 581	0.98	1.91	0.05	0.17	3.11	0.10	0.23	41 369	1.330
F-Boundary South	7 616 744	0.92	1.89	0.04	0.13	2.98	0.06	0.18	22 737	0.731
F Zone Total	**126 287 248**	**0.91**	**2.01**	**0.05**	**0.15**	**3.12**	**0.08**	**0.18**	**393 578**	**12.654**
Waterberg Total	**139 224 118**	**0.94**	**2.02**	**0.05**	**0.21**	**3.22**	**0.09**	**0.18**	**447 564**	**14.390**

Table 15-15: Total Estimated Mineral Reserve at 2.5 g/t Cutoff effective as of 04 September 2019

Zone	Tonnes	Pt	Pd	Rh	Au	4E	Cu	Ni	4E Metal	
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(kg)	(Moz)
T Zone	**16 900 564**	**1.18**	**2.04**	**0.03**	**0.80**	**4.05**	**0.18**	**0.09**	**68 391**	**2.199**
F-Central	70 131 337	0.88	2.02	0.05	0.14	3.09	0.07	0.18	216 349	6.956
F-South	15 653 961	1.06	2.03	0.05	0.15	3.29	0.04	0.13	51 411	1.653
F-North	53 621 900	0.88	2.09	0.05	0.16	3.18	0.10	0.20	170 828	5.492
F-Boundary North	18 288 434	0.98	1.93	0.05	0.17	3.13	0.10	0.23	57 216	1.840
F-Boundary South	12 910 859	0.97	2.06	0.05	0.15	3.23	0.07	0.19	41 756	1.342
F Zone Total	**170 606 492**	**0.91**	**2.04**	**0.05**	**0.15**	**3.15**	**0.08**	**0.19**	**537 560**	**17.283**
Waterberg Total	**187 507 056**	**0.94**	**2.04**	**0.05**	**0.21**	**3.24**	**0.09**	**0.18**	**605 951**	**19.482**

Notes:

- A stope cutoff grade of 2.5 g/t 4E was used for mining planning for the mineral reserves estimate.
- Tonnage and grade estimates include geological losses, dilution, and mining losses.
- 4E = PGE (Pt + Pd + Rh) and Au.
- Numbers may not add due to rounding.

Table 15-16: Prill Splits

Zone	4E Grade Prill Split				Grade	
	Pd (%)	Pt (%)	Au (%)	Rh (%)	Cu (%)	Ni (%)
T Zone	50.4	29.2	19.7	0.7	0.18	0.09
F Zone	64.7	29.0	4.8	1.5	0.08	0.19
Total Waterberg	**63.1**	**29.0**	**6.5**	**1.5**	**0.09**	**0.18**

16 MINING METHODS

16.1 Introduction

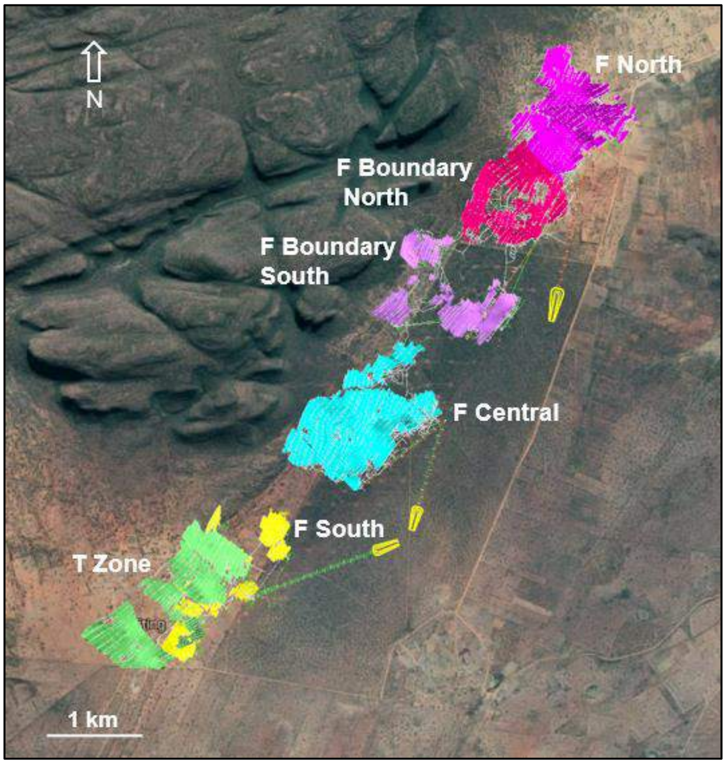
The Waterberg Project will be an underground mining operation accessed via declines from surface. The mine design is based on using the sublevel longhole stoping mining method (Longhole) to extract M&I Mineral Resources contained in the T Zone and F Zone and backfilling the mined voids with paste backfill. Longhole is a highly mechanised, high productivity, and low-cost bulk mining method that uses equipment and processes widely used in the global mining industry.

The Waterberg Project mineralised zones have an overall strike length of approximately 8.8 km extending from the T Zone in the southwest to the F-North Zone in the northeast. Considering the extensive strike length and relative proximity and separation of the zones, the operation was divided into the following three mining complexes.

- The South Complex that includes T Zone and F-South
- The Central Complex that includes F-Central
- The North Complex that includes F-North, F-Boundary North, and F-Boundary South

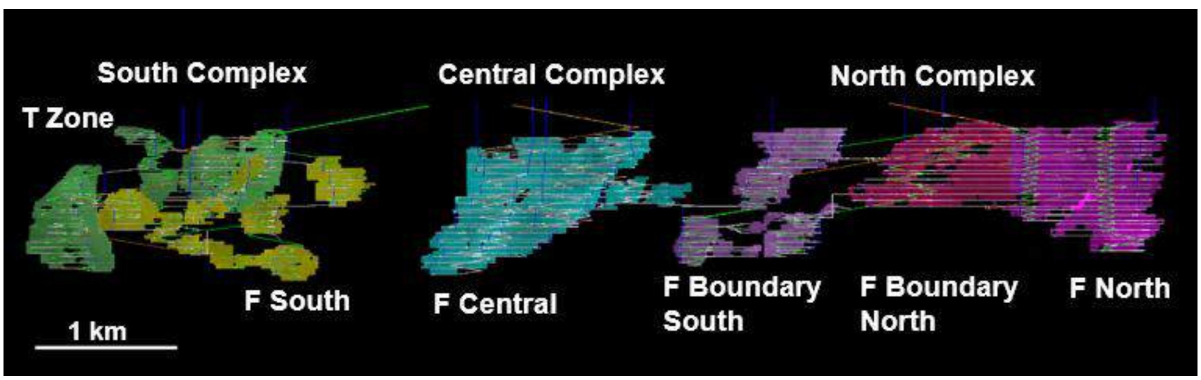
A plan view with the production areas projected to surface is shown in Figure 16-1 and a longitudinal view of the complexes, looking approximately northwest (looking from the footwall), is shown in Figure 16-2.

Figure 16-1: Surface Plan View Showing Production Area Extents



Source: Background – Google Maps

Figure 16-2: Longitudinal View of Waterberg Complexes (Looking Northwest)



16.2 Rock Mechanics

16.2.1 Structural Geology

For the structural geology, numerous dolerite and granodiorite sills and dykes intrude the Waterberg sediments and range in thickness from less than 1 m to more than 90 m.

Shear zones were identified through mapping and geological logging during the PFS with most of the shears indicating a northwest-southeast strike orientation. This aligns with the direction of tectonic forces thought to be associated with the formation of the Limpopo Shear Zone (LSZ). The Waterberg Project is located within the southern margin of the LSZ. Most of these large-scale thrust faults such as the Hout River Fault zone, could have been reactivated after the placement of the Bushveld Complex. This fault zone has an estimated throw of 300 m and a fault splay was interpreted on the southeastern part of the project area.

16.2.2 Geomechanical Model

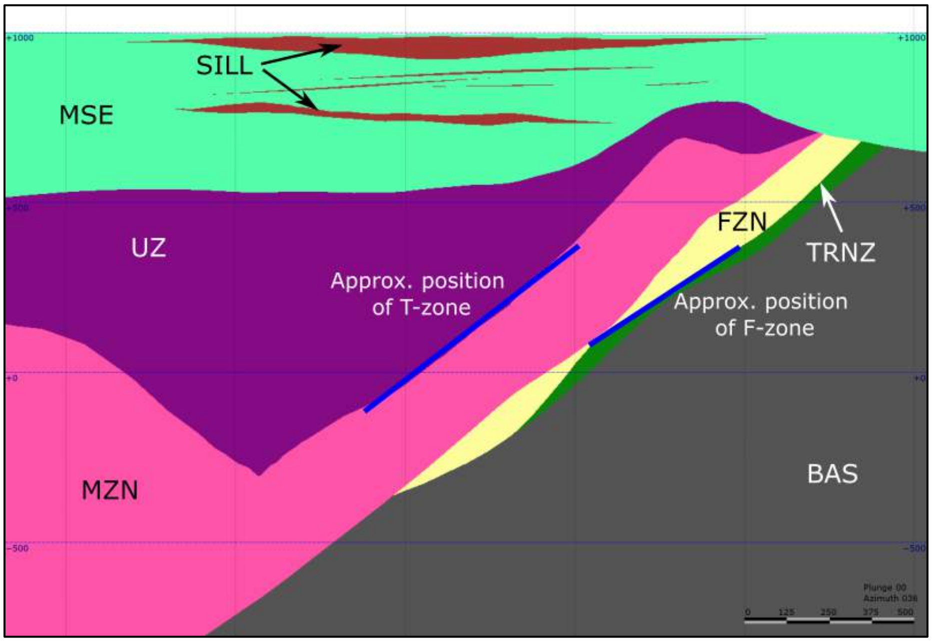
The stratigraphic profile for the Waterberg Project was the key basis for geomechanical domain definition. However, due to the relative paucity of geomechanical data for all lithological units in comparison to geological data, the stratigraphic profile was simplified to develop the principal geomechanical domains summarised in Table 16-1.

Table 16-1: Principal Geomechanical Domains

Geotechnical Domain	Description
MSE	Waterberg Group Sediments
SILL	Sill Intrusions, Dolerite, and Granodiorite
UZ	Upper Zone
TZ_IHW	T Zone Immediate Hanging Wall (0-5 m)
TZ_MIN	T Zone Mineralised Zone
TZ_IFW	T Zone Immediate Footwall (0-5 m)
MZN	Main Zone, Host Rock Mass for T Zone
FZN	F Zone (Lower Main Zone) Host Rock Mass
FZ_IHW	F Zone Immediate Hanging Wall (0-5 m)
FZ_MIN	F Zone Mineralised Zone
FZ_IFW	F Zone Immediate Footwall (0-5 m)
TRNZ	Transition Zone (Lower Main Zone)
BAS	Basement – Hout River Gneiss

The generalised geomechanical model identifies the geomechanical domains recognised for the underground mine design, refer to Figure 16-3. The approximate T Zone and F Zone reef positions within the generalised geotechnical model are also shown. The T Zone and F Zone were further sub-divided into immediate hanging wall (5 m into the hanging wall from the mineralised zone contact), mineralised zone (identified mining zone), and immediate footwall (5 m into the footwall from the mineralised zone contact).

Figure 16-3: Generalised Geomechanical Model



16.2.3 *In Situ* Stress

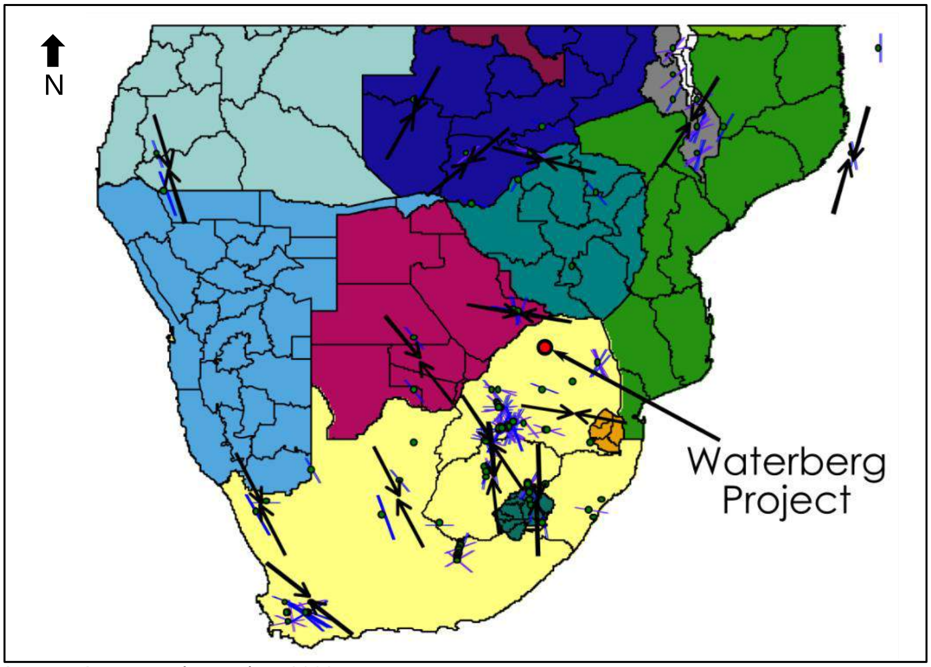
In situ stress measurements have not been undertaken. These stresses will need to be defined as the project moves into execution as the current stress assumptions introduce some uncertainty to the stope design. An approximate range of the likely *in situ* stress regime was estimated from regional measurements and used in the stope design. Maximum principal stress directions for the project region have been estimated from the sources listed below.

- World Stress Measurement Database (Heidbach, Rajabi, Reiter, & Ziegler, 2016).
- (Stacey & Wesseloo, 2002).

Figure 16-4 shows stress directions for South Africa, together with the project location. The general trend for the maximum principal stress in the region of the project location ranges from NNW-SSE to WNW-ESE, with a mean around NW-SE. Data sites taken from the World Stress Measurement Database for the Bushveld Igneous Complex, show trends ranging from 120° to 158°, with a mean trend of around 142°.

To estimate principal stress magnitudes, it is assumed that the minor principal stress is vertical, and that the vertical stress is calculated based on depth below surface.

Figure 16-4: Orientations of Horizontal Principal Stress from *In Situ* Stress Measurements



Source: Stacey and Weseloo, 2002

A summary of the likely *in situ* stress regime for the Waterberg Project is shown in Table 16-2.

Table 16-2: Estimated *In Situ* Stress Regime

Parameter	Upper	Mean	Lower
Maximum Principal Stress Orientation	158°	142°	120°
Major Horizontal Stress (σ_H) versus σ_v Ratio	2.0	1.5	1.0
Minor Horizontal Stress (σ_h) versus σ_v Ratio	1.3	1.0	0.6

16.2.4 Geomechanics Data

The majority of geomechanics data for the DFS was collected by Open House Management Solutions (OHMS). The data consists of geomechanical interval logging, point structure logging (un-oriented), point load tests, and geomechanical laboratory test results.

The following geomechanical data was utilised for the DFS.

- 13 264 m of Geomechanical Core Logging
- 123 UCS Tests
- 177 Indirect UTS Tests
- 233 Peak Load Triaxial Tests
- 12 Base Friction Angle Tests

- 504 m of PFS Televiewer Data (2 715 data points)
- 9 383 m of DFS Televiewer data (50 006 Data Points)

16.2.4.1 Geomechanics Logging

Geomechanics logging data from both the PFS and DFS programmes were incorporated into a drill hole database and used to develop the geomechanical model. The geomechanical logging contained parameters for use in geomechanics rock mass classification systems, including the following.

- RQD (Deere 1964)
- Norwegian Geotechnical Institute (NGI) Q-System (Barton, Lien, & Lunde, 1974).
- Bieniawski´s 1989 RMR’89 System (Bieniawski, 1989).
- Laubscher’s 1990 RMR’90 System (Laubscher, 1990).

16.2.4.2 Rock Quality Designation

RQD is a rock mass classification index that describes the degree of fracturing (Table 16-3). It also forms the basis of other rock mass classification systems which include other characteristics of the rock mass.

Table 16-3: Rock Quality Designation Classification

RQD	Rock Mass Quality
<25%	Very Poor
25% to 50%	Poor
50% to 75%	Fair
75% to 90%	Good
90% to 100%	Excellent

All RQD values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-4.

This analysis indicates that the majority of geomechanical domains have on average a “good” rock mass quality, based on RQD. The exceptions being MSE (sediments) and Sill domains, which display “fair” rock mass quality and higher variability.

NGI Q-System Joint Set Number

The joint set number (Jn) parameter describes and rates the number of identified joint sets (Table 16-5) within the drilling run. All Jn values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-6.

Table 16-4: Rock Quality Designation (%) Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	694	300	309	2 030	6 496	6 737	8 011	1 097	1 851	60	60	85	654
Mean	78.60	79.40	73.60	77.40	78.70	60.30	79.60	63.10	77.70	77.30	76.10	80.40	72.80
Standard Deviation (SD)	18.7	18.3	24.1	20.2	22.0	29.9	21.0	29.3	20.2	18.4	18.3	13.7	20.9
CV	0.24	0.23	0.33	0.26	0.28	0.50	0.26	0.46	0.26	0.24	0.24	0.17	0.29
Variance	349	334	581	407	485	895	441	857	410	340	336	188	436
Minimum	8	32	10	10	10	3	4	4	10	51	49	52	13
Q1	66	70	65	67	68	38	69	41	66	53	55	67	56
Q2	84	86	78	83	87	65	87	60	84	83	81	85	76
Q3	96	95	92	93	96	88	96	95	94	91	94	91	91
Maximum	100	100	100	100	100	100	100	100	100	100	99	100	100

Table 16-5: NGI Q-System Joint Set Number

Description	Jn
Massive, No or Few Joints	0.5-1.0
One Joint Set	2.0
One Joint Set Plus Random Joints	3.0
Two Joint Sets	4.0
Two Joint Sets Plus Random Joints	6.0
Three Joint Sets	9.0
Three Joint Sets Plus Random Joints	12.0
Four or More Joint Sets, Heavily Jointed, "Sugar-Cube", etc.	15.0
Crushed Rock, Earthlike	20.0

Table 16-6: Joint Set Number Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	509	260	259	1 708	5 433	5 775	7 408	986	1 325	60	60	85	654
Mean	8.5	7.3	7.1	8.1	7.1	8.3	8.1	9.0	7.3	9.1	8.0	8.1	8.3
SD	5.3	3.7	4.1	4.2	4.6	5.9	4.8	6.0	4.2	6.0	6.7	6.4	5.6
CV	0.62	0.50	0.57	0.52	0.65	0.72	0.59	0.67	0.58	0.66	0.84	0.78	0.68
Variance	28	14	16	18	21	35	23	36	18	36	45	41	32
Minimum	0.5	0.5	2	0.5	0.5	0.5	0.5	0.5	0.5	2	0.5	1.64	0.5
Q1	4	4	4	4	3	4	4	4	4	4	3	4	4
Q2	9	6	6	9	6	6	9	6	6	12	4	4	6
Q3	12	9	12	9	9	12	12	12	9	12	12	12	12
Maximum	20	20	20	20	20	20	20	20	20	20	20	20	20

It should be noted that, due to the directional drill hole bias and generally short intervals assessed during core logging, the Jn parameters in the core logging may not adequately capture the actual number of sets present within a domain or lithological unit. The discontinuity analysis presented in Section 0, provides a more representative assessment of the families of discontinuities present within each geomechanical domain.

NGI Q-System Joint Roughness Number

The joint roughness number (Jr) parameter describes and rates the small-scale surface features on open and exposed discontinuities within the drilling run as shown in Table 16-7. All Jr values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-8.

Table 16-7: NGI Q-System Joint Roughness Number

Description	Jr
Discontinuous Joints	4.0
Rough or Irregular, Undulating	3.0
Smooth, Undulating	2.0
Slickensided, Undulating	1.5
Rough or Irregular, Planar	1.5
Smooth, Planar	1.0
Slickensided, Planar	0.5

Table 16-8: Joint Roughness Number Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	509	260	259	1 708	5 433	5 775	7 408	986	1 325	60	60	85	654
Mean	2.6	2.0	1.8	2.1	1.6	2.1	1.5	2.1	2.6	1.7	1.3	1.7	1.5
SD	0.9	0.8	0.7	0.8	0.7	0.8	0.6	0.8	0.7	0.6	0.5	0.6	0.6
CV	0.33	0.42	0.39	0.35	0.44	0.39	0.41	0.36	0.28	0.38	0.35	0.35	0.37
Variance	1	1	0	1	1	1	0	1	1	0	0	0	0
Minimum	0.5	0.5	0.5	0.5	0.5	0.0	0.5	1.0	0.5	1.0	0.5	1.0	0.5
Q1	3	2	2	2	2	2	2	2	3	2	1	2	2
Q2	3	2	2	2	2	2	2	2	3	2	2	2	2
Q3	3	3	2	3	2	3	2	3	3	2	2	2	2
Maximum	3	3	3	4	3	4	4	4	3	3	2	3	3

NGI Q-System Joint Alteration Number

The joint alteration number (Ja) parameter describes and rates the small-scale joint wall characteristics and infill characteristics on open and exposed discontinuities as shown in Table 16-9 within the drilling run. All Ja values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-10.

The analysis suggests that joints within most domains are predominantly slightly altered, with some coatings and thin fillings of non-softening materials. Joints within the MSE and SILL domains have higher dispersion, including joints/features with thicker infills, some with soft cohesive materials.

Table 16-9: NGI Q-System Joint Alteration Number

<table>
<tr><th rowspan="6">Contact Between Joint Walls</th><th colspan="2">Joint Wall Character</th><th>Condition</th><th>Jn
(Wall Contact)</th></tr>
<tr><td rowspan="3">Clean Joints</td><td>Healed or Welded Joints</td><td>Filling of Quartz, Epidote, etc.</td><td>0.75</td></tr>
<tr><td>Fresh Joint Walls</td><td>No Coating or Filling, except from Staining</td><td>1</td></tr>
<tr><td>Slightly Altered Joint Walls</td><td>Non-softening Mineral Coatings, Clay-Free Particles, etc.</td><td>2</td></tr>
<tr><td rowspan="2">Coating or Thin Film</td><td>Friction Materials</td><td>Sand, Silt, Calcite, etc. (non-softening)</td><td>3</td></tr>
<tr><td>Cohesive Materials</td><td>Clay, Chlorite, Talc, etc. (softening)</td><td>4</td></tr>
<tr><th rowspan="6">Some or No Wall Contact</th><th rowspan="2">Filling Materials</th><th rowspan="2">Type</th><th>Jn (Some Wall Contact)</th><th>Jn (No Wall Contact)</th></tr>
<tr><th>Thin Filling (<5 mm)</th><th>Thick Filling</th></tr>
<tr><td>Friction Materials</td><td>Sand, Silt, Calcite, etc. (non-softening)</td><td>4</td><td>8</td></tr>
<tr><td>Hard Cohesive</td><td>Compact Filling of Clay, Chlorite, Talc, etc.</td><td>6</td><td>5-10</td></tr>
<tr><td>Soft Cohesive</td><td>Medium to Over Consolidated Clay, Chlorite, Talc</td><td>8</td><td>12</td></tr>
<tr><td>Swelling Clays</td><td>Filling Materials Exhibits Swelling Properties</td><td>8-12</td><td>13-20</td></tr>
</table>

Table 16-10: Joint Alteration Number Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	509	260	259	1 708	5 433	5 775	7 408	986	1 325	60	60	85	654
Mean	1.7	2.5	2.5	2.3	2.3	2.8	2.6	3.6	1.8	2.6	1.8	2.6	2.9
SD	0.6	1.2	1.0	1.0	1.2	2.3	1.1	3.6	0.9	1.0	0.9	1.0	0.6
CV	0.38	0.48	0.39	0.45	0.50	0.82	0.41	0.99	0.47	0.36	0.49	0.37	0.20
Variance	0	1	1	1	1	5	1	13	1	1	1	1	0
Minimum	0.75	0.75	1.00	0.75	0.75	0.75	0.75	0.75	0.75	1.00	1.00	1.00	1.00
Q1	1	1	2	1	1	1	2	1	1	2	1	2	3
Q2	2	3	3	3	3	3	3	3	2	3	1	3	3
Q3	2	3	3	3	3	3	3	3	2	3	3	3	3
Maximum	3	6	4	6	8	13	8	13	4	4	3	4	4

NGI Q-System Q' Number

The Q-system Q' number (Q') parameter includes the calculation of the logged terms *RQD*, *Jn*, *Jr* and *Ja*. Water (*Jw*) and stress (*SRF*) are not considered. The *Q'* parameter is calculated as shown in Equation 16-1.

$$Q' = \frac{RQD}{Jn} x \frac{Jr}{Ja}$$

Equation 16-1

Table 16-11 can be used to describe rock mass conditions based on the range of *Q'* values (assuming *Q'* is equal to *Q*).

Table 16-11: NGI Q-System Classification

Q	Rock Mass Quality
0.001 – 0.01	Exceptionally Poor
0.01 – 0.1	Extremely Poor
0.1 – 1	Very Poor
1 – 4	Poor
4 - 10	Fair
10 - 40	Good
40 - 100	Very Good
100 – 400	Extremely Good
>400	Exceptionally Good

All *Q'* values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-12.

Table 16-12: Q' Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	694	300	309	2 030	6 496	6 737	8 011	1 097	1 851	60	60	85	654
Mean	27.9	23.8	13.4	20.9	29.0	33.7	22.3	40.0	46.7	16.2	52.7	16.6	26.1
SD	39.8	53.3	14.9	40.1	66.8	109.3	66.1	128.1	111.8	29.8	96.5	19.3	65.7
CV	1.43	2.24	1.11	1.92	2.31	3.24	2.96	3.20	2.39	1.85	1.83	1.16	2.52
Variance	1 585	2 841	223	1 611	4 460	11 950	4 365	16 399	12 490	891	9 308	373	4 321
Minimum	0.56	0.73	0.34	0.17	0.00	0.00	0.00	0.16	0.56	1.55	1.38	1.55	0.33
Q1	10	3	4	3	3	2	3	2	8	3	4	6	2
Q2	14	10	8	8	6	6	5	3	16	6	9	8	5
Q3	29	16	16	23	25	30	15	33	33	10	21	15	8
Maximum	600	296	65	506	576	1 067	597	1 067	597	204	297	77	299

Bieniawski's 1989 Rock Mass Rating

Bieniawski's RMR'89 system combines the most "significant" geologic parameters of influence and presents one overall comprehensive index of rock mass quality, see Table 16-13, which is used for the design and construction of excavations in rock, such as tunnels, mines, slopes, and foundations.

Table 16-13: Rock Mass Rating'89 Classification

RMR'89	Rock Mass Quality
0 – 20	Very Poor
21 – 40	Poor
41 – 60	Fair
61 – 80	Good
81 – 100	Very Good

The RMR'89 values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-14.

The RMR'89 statistics generally indicate mean values between 63 and 67 for each domain, with MSE and Sill domains having slightly lower means (around 56 and 58, respectively). This indicates that rock mass conditions are, in general, represented by "good" rock mass conditions, with MSE and SILL domains classified as "fair" rock mass conditions.

Table 16-14: RMR'89 Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	694	300	309	2 030	6 496	6 737	8 011	1 097	1 851	60	60	85	654
Mean	67	65	63	62	66	56	63	58	66	66	56	67	61
SD	9.6	9.5	11.5	8.1	15.7	12.0	15.1	7.9	9.5	10.6	26.1	8.5	14.1
CV	0.14	0.15	0.18	0.13	0.24	0.21	0.24	0.14	0.14	0.16	0.47	0.13	0.23
Variance	93	91	131	66	247	143	229	63	89	113	679	72	200
Minimum	36	45	35	27	0	26	0	32	36	43	0	43	27
Q1	63	58	56	57	59	50	56	52	60	64	51	60	49
Q2	69	64	63	63	67	58	66	59	66	69	66	69	59
Q3	74	72	74	65	77	63	74	63	73	70	74	72	74
Maximum	80	88	88	92	91	84	89	75	92	79	74	79	84

Laubscher's 1990 Rock Mass Rating'90

The mean Laubscher RMR'90 values for each domain show similar mean values to Bieniawski´s RMR'89 values; however, the differences, or variance, between domains is more discernable with the Laubscher values.

Laubscher's RMR'90 values were composited to 1 m and statistically analysed by geomechanical domain, with the results shown in Table 16-15.

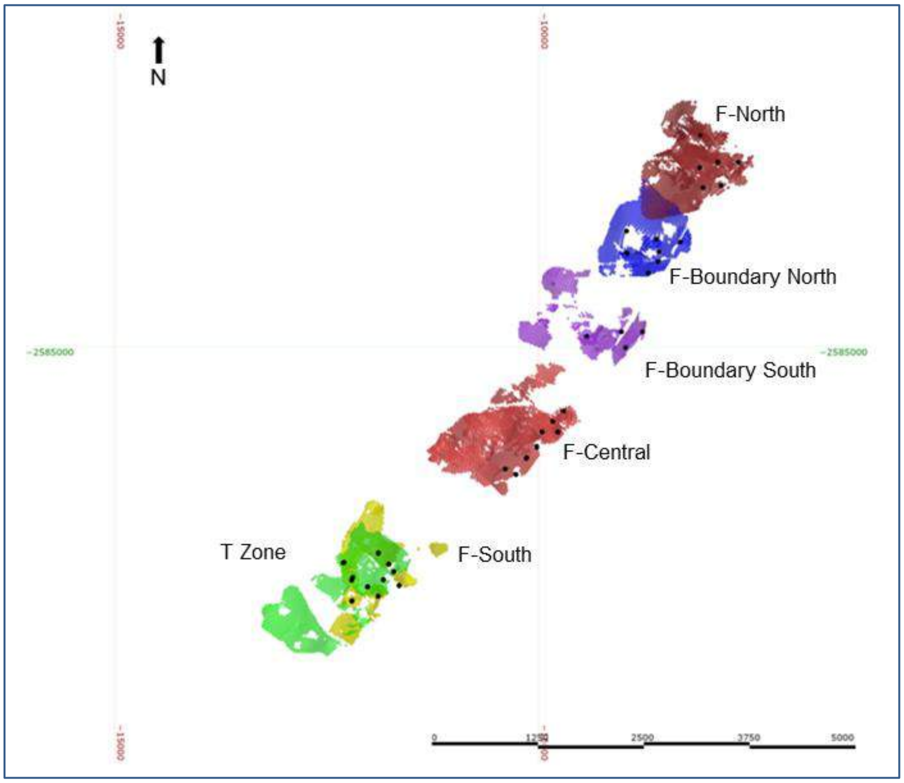
Table 16-15: Rock Mass Rating'90 Summary Statistics by Geomechanical Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	694	300	309	2 030	6 496	6 737	8 011	1 097	1 851	60	60	85	654
Mean	64	55	54	50	59	46	57	51	61	59	63	59	56
SD	9.8	12.6	13.3	9.7	17.4	12.9	13.7	8.1	13.3	8.8	12.3	8.3	11.9
CV	0.15	0.23	0.25	0.19	0.29	0.28	0.24	0.16	0.22	0.15	0.20	0.14	0.21
Variance	96	159	176	95	302	166	187	66	177	77	151	69	141
Minimum	40	35	21	25	0	11	0	33	26	48	45	49	37
Q1	59	46	45	45	50	40	52	46	51	53	54	54	48
Q2	64	55	52	50	59	46	57	51	61	57	58	56	55
Q3	69	62	62	55	69	52	63	58	67	61	74	60	59
Maximum	94	89	85	89	98	83	98	77	94	75	84	75	83

16.2.4.3 Acoustic Televiewer Data

The principal data source for discontinuity orientations was from Acoustic Televiewer (ATV) geophysical logging data. The location of the 38 holes with ATV logs are shown in Figure 16-5. There were 52 721 data points.

Figure 16-5: Plan Showing Distribution of Televiewer Holes (Black Markers)

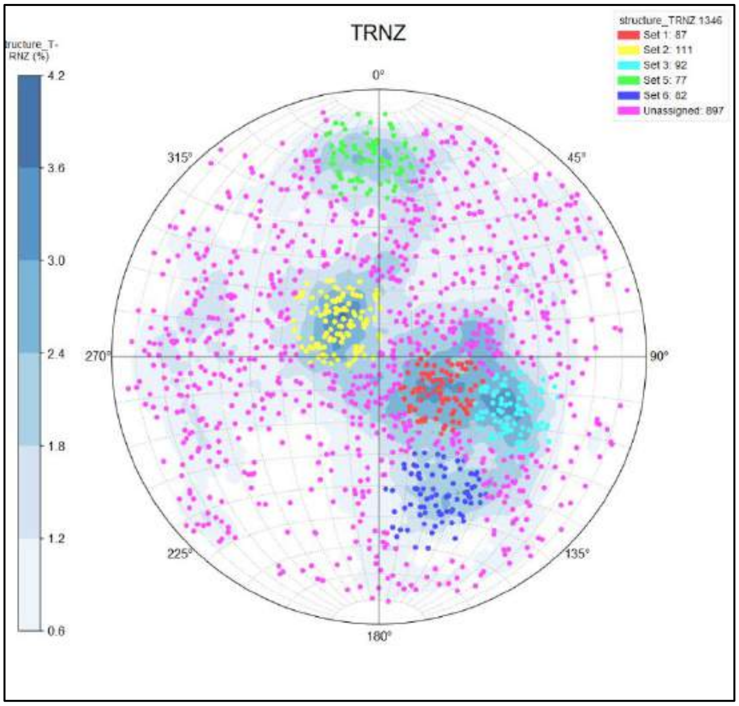


Generalised Discontinuity Orientations

Oriented discontinuity data was restricted to processed ATV survey data. The ATV survey data consisted of corrected (true north referenced) orientation, estimates of aperture (in mm), expression on drill hole wall (59%, 75%), type (planar, non-planar) and openness (open, closed).

Discontinuity orientations per domain were assessed via stereographical analysis. An example stereographic projection is shown in Figure 16-6.

Figure 16-6: Lower Hemisphere Stereographic Projection of ATV Data for TRNZ Domain, Separated into Identified Sets



16.2.4.4 Geomechanics Laboratory Testing

Intact rock properties were developed from geomechanics laboratory testing. The following intact rock property tests were undertaken.

- Density
- UCS with Elastic Properties (Young's modulus and Poisson's ratio)
- Peak Load Triaxial Results (single stage pre-selected confining pressures)
- Indirect Tensile Strength (ITS) (Brazilian)
- Direct Shear Test for Basic Friction Angle Determination (saw cut surfaces)

As all holes are subvertical; no directional bias for intact rock properties could be evaluated.

Uniaxial Compressive Strength

UCS and triaxial results were examined for valid failure modes. During testing, the failure mode was recorded by the laboratory, as either failing through intact rock, along discontinuities, or a combination of both. The angle to the core axis of the discontinuities involved in the failure were also recorded. Results where failure clearly occurred on unfavorably oriented pre-existing discontinuities were removed from the analysis database. In this case, where the angle of the discontinuity to the core axis is between 20° and 60°.

Approximately 88 invalid tests were removed from the entire original database of 702 (approximately 13% were deemed invalid). For only the UCS tests, 35 tests were removed from a total of 169 UCS test results, (approximately 21% deemed invalid).

The results of validated UCS test results for each domain are presented in Table 16-16.

The intact rock strength for most domains is approximately 200 MPa, with the MZN domain slightly lower at 178 MPa, and the UZ domain around 120 MPa (one sample) and the MSE averaging around 146 MPa.

The immediate footwall of the F Zone, as well as the immediate footwall and hanging wall of the T Zone (FZ_IFW, TZ_IHW, and TZ_IFW), contain no UCS samples. This is principally due to the relatively small domain volume, being a 5 m thick skin above and below the mineralised zones.

The intact rock strength for the immediate hanging wall of the T Zone can be estimated from the representative UZ host rock mass, and the immediate footwall of the T Zone from the MZN host rock mass. The intact rock strength for the immediate hanging wall and footwall of the F Zone can be estimated from the representative FZN host rock mass.

It should be noted that the mineralised T Zone only contains three valid UCS samples, which represents uncertainty for geomechanics mine design, especially pillar design. The UCS sample results for T Zone vary between 106 MPa and 234 MPa. Although this results in a mean intact rock strength of 151 MPa, the triaxial data indicates that T Zone UCS should be higher. Based on the triaxial results, the mean T Zone UCS is closer to 200 MPa. For this DFS, the value of 151 MPa was used for analysis but further testing to confirm the intact strength values may provide opportunities as the project progresses.

Indirect Tensile Strength

ITS results for each domain are presented in Table 16-17.

Table 16-16: Results of Validated Uniaxial Compressive Strength (MPa) Tests by Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	5	1	2	20	26	25	34	6	6	-	1	3	5
Mean	247	231	202	225	189	146	178	246	231	-	70	151	120
SD	50.10	Infinity	43.00	33.70	52.90	93.30	48.60	115.70	66.60	-	Infinity	72.29	40.42
CV	0.20	Infinity	0.00	0.15	0.28	0.64	0.27	0.47	0.29	-	Infinity	0.48	0.34
Variance	2 509	Infinity	1 812	1 132	2 798	8 704	2 361	13 380	4 433	-	Infinity	5 226	1 634
Minimum	178	231	172	144	62	1	22	18	141	-	70	106	60
Q1	222	231	172	204	177	100	160	238	188	-	70	106	101
Q2	247	231	172	229	194	162	185	281	201	-	70	112	126
Q3	290	231	232	249	228	209	206	314	302	-	70	234	155
Maximum	300	231	232	272	262	300	248	330	309	-	70	234	157

Table 16-17: Results of Indirect Tensile Strength (MPa) by Domain

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
Count	4	4	3	48	49	13	49	12	20	-	1	1	-
Mean	15	13	13	14	15	12	14	14	14	-	12	6	-
SD	2.2	5.0	0.9	2.7	3.1	6.6	2.9	9.1	3.0	-	Infinity	Infinity	-
CV	0.15	0.37	0.07	0.20	0.21	0.54	0.21	0.66	0.21	-	Infinity	Infinity	-
Variance	5	25	1	8	10	44	8	84	9	-	Infinity	Infinity	-
Minimum	13	6	12	6	9	0	7	2	8	-	12	6	-
Q1	13	6	12	12	13	7	12	4	13	-	12	6	-
Q2	13	14	13	13	14	13	14	13	14	-	12	6	-
Q3	16	15	14	15	17	17	16	22	16	-	12	6	-
Maximum	17	18	14	20	23	24	20	25	20	-	12	6	-

Triaxial Strength

Triaxial strength tests on intact rock were undertaken at certain confining pressures on individual intact samples to obtain peak strength envelope for the intact rock within each domain.

The Hoek-Brown (H-B) failure criterion (Hoek & Brown, 1988) was used to estimate the triaxial strength curve of intact rock for each domain, where sufficient test data were available. In fitting the H-B curve, valid UCS and ITS results were also considered. The curves can also be used to estimate averaged UCS and ITS by the curve intercepts with the vertical and horizontal axes, respectively. A comparison of estimated fitted values against test results is shown in Table 16-18, together with the respective Hoek-Brown m_i value.

Table 16-18: Comparison of Mean Laboratory Uniaxial Compressive Strength Test Results (MPa) with Values Estimated from H-B Fit to Triaxial Test Data

	BAS	FZ_IFW	FZ_IHW	FZ_MIN	FZN	MSE	MZN	SILL	TRNZ	TZ_IFW	TZ_IHW	TZ_MIN	UZ
UCS Lab Test	245	231	202	227	197	146	177	246	244	-	70	151	120
UCS Estimated	258	198	200	216	217	169	205	300	261	-	172	201	128
ITS Lab Test	14.7	13.4	12.9	13.7	14.6	12.4	13.9	13.8	14.3	-	12.0	5.7	-
ITS Estimated	17.1	16.3	15.2	14.1	15.8	13.8	16.0	20.0	14.5	-	12.3	9.5	4.3
m_i Value	15.0	12.1	13.1	15.3	13.7	12.2	12.7	15.0	17.9	-	13.9	21.0	30.0

16.2.5 Geomechanics Parameters for Mine Design

The following section outlines the development of key geomechanical design parameters for the proposed mining methods, principally focusing on verifying stope dimensions and backfill performance.

The vertical distance between mining blocks will be 100 m. Individual longitudinal and transverse stopes will be limited to a maximum vertical height of 40 m. The maximum longitudinal stope strike length will be 20 m, while the stope width for transverse stopes will be 20 m along strike. In thicker parts of the ore, there will be a need to limit the maximum stable length of transverse stopes to 40m.

Backfill pillar and stope span stability and dimensioning were undertaken using empirical methods commonly used in the mining industry. These were subsequently checked using three-dimensional finite element modelling.

For the definition of span design, the Mathews method (Mathews, Hoek, Wyllie, & Stewart, 1981) and the extended Mathews empirical stability graph for open-stope design (Mawdesley, Trueman, & Whiten, 2001) were used.

The method was utilised to confirm the proposed stable stope dimensions, by ensuring that the design hydraulic radius for back and wall spans do not exceed an “allowable” hydraulic radius. For stable stope design, with “acceptable” stability and dilution parameters (based on the empirical case history database), based on current industry practice, the “Stable-Failure” design line was used.

16.2.5.1 Backfill Stability

The mine design relies on stable paste backfill exposures. The required backfill strength is largely a function of the role and requirements of backfill, geometrical aspects of the fill / void, and extent of exposure of backfill with mining. For the proposed mining method, the following locations affect the backfill needs.

- Primary Stope Fill Face Exposure
- Secondary Stope Fill Face (no exposure)
- Underhand Fill Sill Pillar
- Working Platform

Backfill stability was assessed primarily using empirical-analytical methods (Mitchell, Olsen, & Smith, 1982) with developed backfill strength requirements validated by benchmarking and limited 3D finite element modelling.

16.2.5.2 Backfill Design Parameters

The design parameters for backfill calculations used to perform various stability assessments and provide empirical mine design parameters as shown in Table 16-19.

Table 16-19: Backfill Design Parameters

Parameter	Value
Factor of Safety Underhand	2.00
Factor of Safety Walls	1.20
Density of Fill Above (tpm^3)	2.00
Stope Dip	40.00º
Tensile to Compressive Ratio	0.12
Friction Angle	33.00º

16.2.5.3 Stope Stability

The empirical stability chart method was used to assess the stability of the proposed stope dimensions. Two separate stability charts were developed, one for F Zone and another for T Zone. The stability number (*N'*) was calculated at various depths (300 m to 800 m depth) and the "allowable" hydraulic radius calculated for the selected design line. The following two design lines were evaluated.

- Stable Failure Line
- Failure Major Failure Line

The resulting stability charts are shown in Figure 16-7 and Figure 16-8.

For almost all analysis cases, the "allowable" hydraulic radius is much greater than the hydraulic radius for the proposed stope dimensions. Only one case, (hanging wall for greater than 800 m depth) were the proposed stope dimension plots slightly more than the "allowable" hydraulic radius. Some stope minor failure and/or early entry of dilution may be anticipated close to final stope extraction; however, a very low probability of major failure is anticipated.

Figure 16-7: Stope Span Dimensions – F Zone

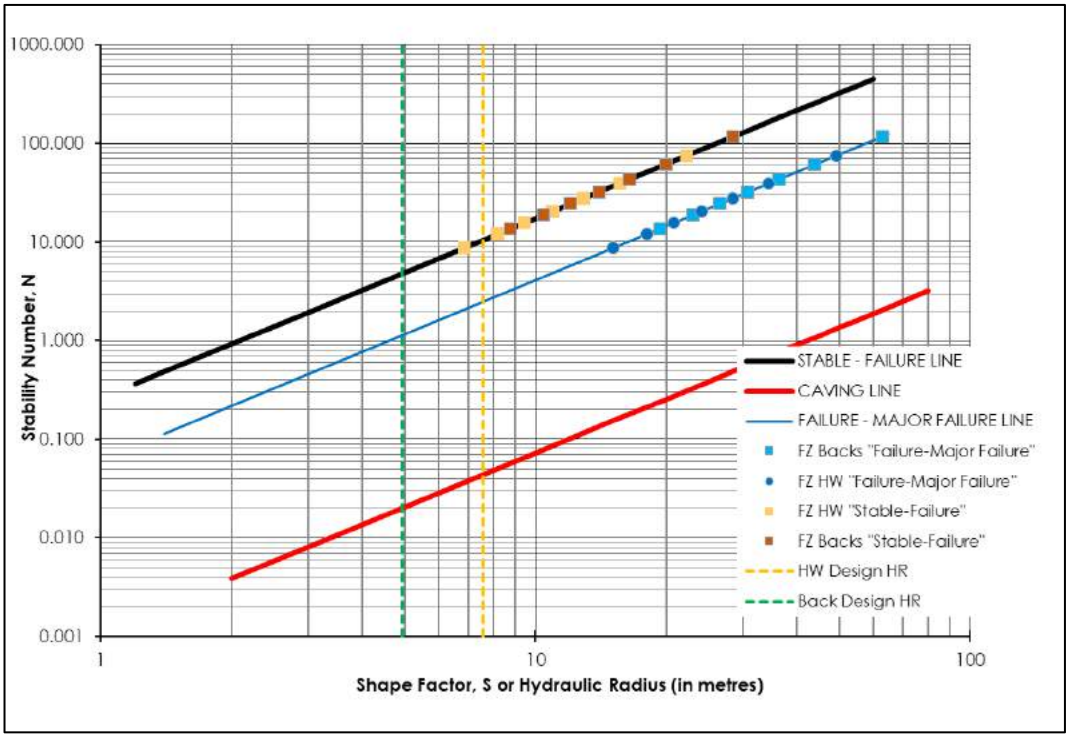
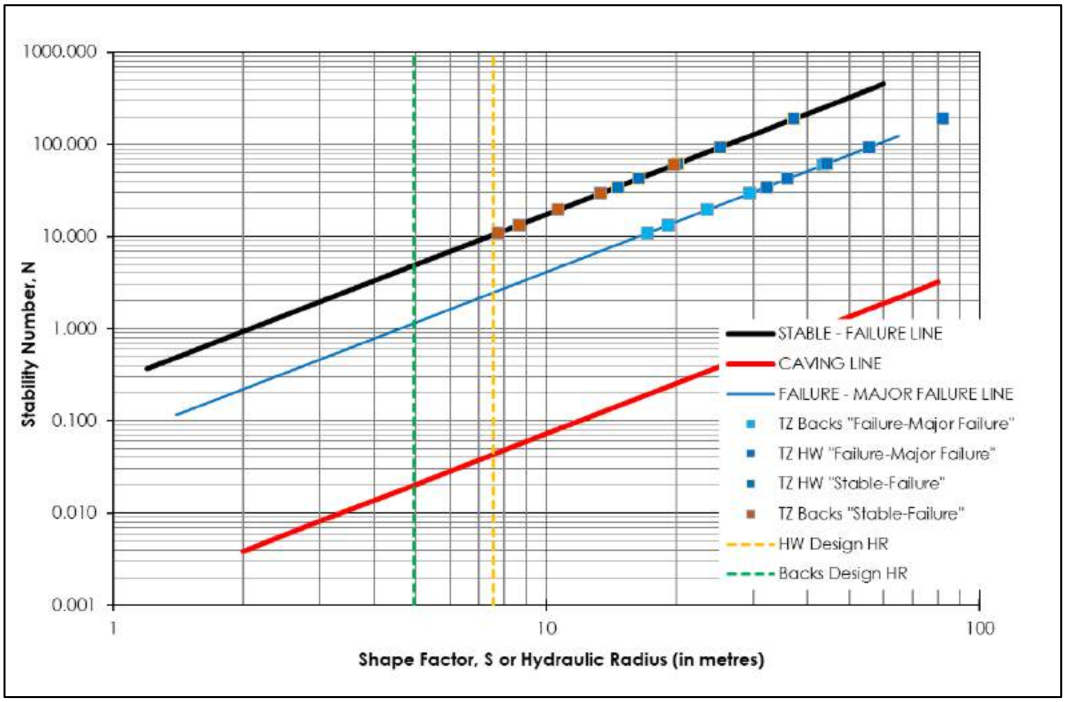


Figure 16-8: Stope Span Dimensions – T Zone



16.2.5.4 Cable Bolt Support

The empirical analysis indicates that for the proposed stope dimensions, stopes are stable without support.

The presence of hanging wall parallel structures will potentially have a large impact on hanging wall stability and dilution during production. Due to the low dip angle of the ore body, and practical limits to production equipment, the potential to undercut these unfavorable structures will generate instability and dilution. To mitigate potential instability, cable bolting of the hanging wall has been incorporated into the design. Following are the principal mechanisms of the cable bolt design.

- Apply compression to improve resistance against shear and tension across stope wall parallel geological structures.
- Create a composite beam of rock between structures. The strength of the beam can be improved with concentrated installation in bands, minimizing slip along strike and dip of adjacent stopes.
- Anchor unstable zones to stable / solid ground while providing retention capability.
- Minimise large stope deformations from relaxation of spans to assist in backfill performance.

Based on this, Table 16-20 shows the recommended cable bolt design guidelines.

Table 16-20: Cable Bolts Required for Longitudinal and Transverse Stopes

Stope	Cable Bolt Ring Spacing	Number of Cables per Ring
Longitudinal – Hanging Wall Support	2.5 m	2 x 10 m Cable Bolts 3 x 15 m Cable Bolts
Transverse – Hanging Wall Support	2.5 m	2 x 10 m Cable Bolts 5 x 15 m Cable Bolts
Transverse – Back Support	3.5 m	5 x 10 m Cable Bolts

16.2.5.5 Paste Backfill Wall Exposures

Using the proposed stope geometries, following is the approximate average required UCS of the paste backfill.

- 0.46 MPa for Primary Transverse Stopes
- 0.35 MPa for Longitudinal Stopes

To mitigate the potential of liquefaction of placed paste backfill, it is recommended that the strength of fill in secondary stopes is a minimum of 0.1 MPa.

16.2.5.6 Underhand Fill Sill Pillar Strength

For each potential failure mode, the limiting equilibrium conditions were established and the estimated fill unconfined compressive strength determined to provide factors of safety of 2.0, which provide more than sufficient degree of safety for non-entry mining under backfill.

The dip of the hanging wall and footwall were fixed at 40º and the sill pillar width to height ratio (pillar thickness to stope width) was fixed at 0.5. For a 20 m wide (W) stope, a sill pillar thickness of 10 m was used. For the sliding mechanism, only cohesion was used, and stabilizing influence of wall closure was not included in the analysis.

The results of the limit equilibrium failure mode analysis are shown in Figure 16-9. The rotational failure mode is the most critical, requiring higher strength backfill to maintain the factor of safety of 2.0.

A parametric analysis was also completed of the rotational failure mode to establish the pillar thickness and strength requirements for various stope widths. The results of this analysis are shown in Figure 16-10. This figure can be used to determine the minimum sill fill pillar strength based on stope width and thickness of pillar. The potential for rotational failure, although controlled by stope dip, is heavily influence by fill pillar thickness (Figure 16-11). To effectively mitigate the risk of rotational failure, d:L ratios of greater than 0.6 are required.

Figure 16-9: Underhand Fill Sill Pillar Limit Equilibrium Results (d:L = 0.5)

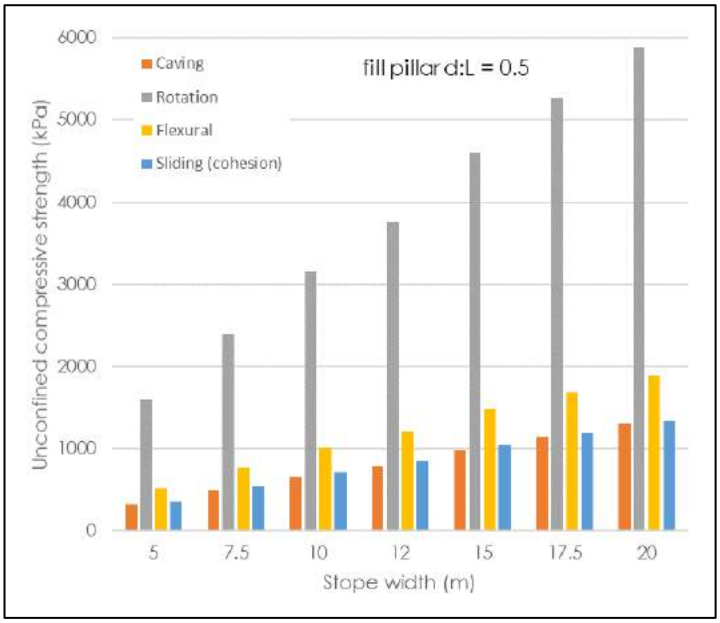


Figure 16-10: Underhand Fill Sill Pillar Rotational Limit Equilibrium Results

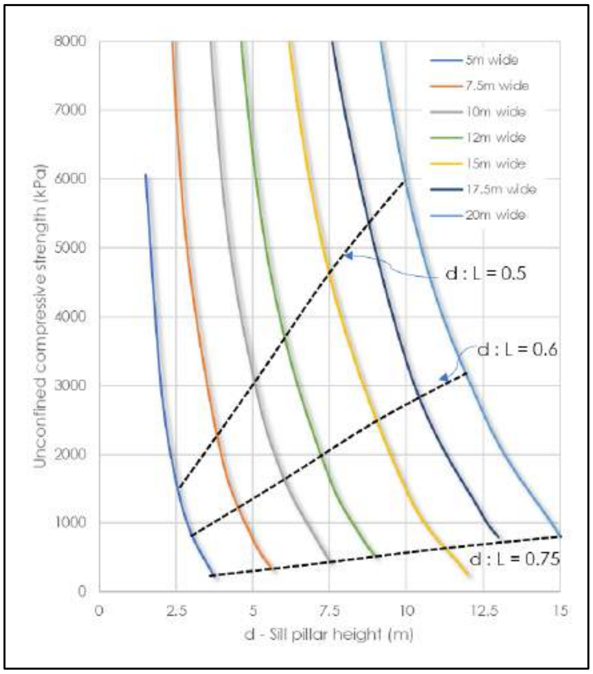
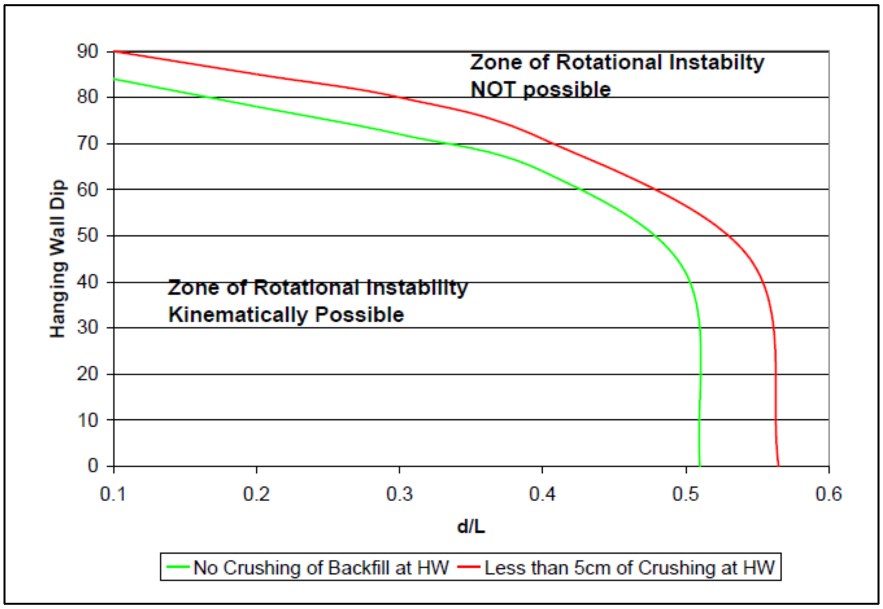


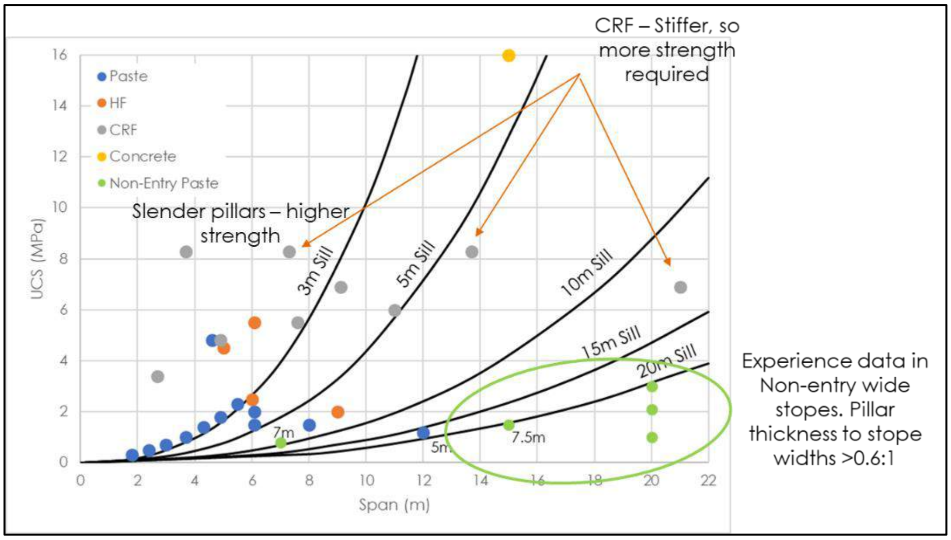
Figure 16-11: Rotational Failure Kinematic Potential



Source: Hughes, 2014)

Experience shows that thicker sill pillars (with d:L ratio greater than 0.6) require lower strength paste backfill (Figure 16-12).

Figure 16-12: Underhand Cut and Fill (Entry) Sill Pillar Benchmark Data



Source: Pakalnis et al., 2005

Fill sill pillars of d:L ratio greater than 0.6:1 will result in stable fill sill pillars with acceptable and achievable paste backfill strengths (less than 3.0 MPa), without the need for aggregates. To ensure an adequate factor of safety and lower required paste backfill strengths, it is recommended that a fill sill pillar to stope width (d:L) ratio of 1:1 is used and a paste backfill strength of 2.0 MPa. This is in line with current industry practice for non-entry underhand stoping methods.

16.2.6 Three-dimensional Finite Element Modelling

To validate the proposed empirical mine design parameters, a 3D numerical modelling exercise was undertaken using GTS NX finite element modelling.

The model considered the following key aspects.

- The principal geotechnical unit geometries and associated material properties.
- The estimated *in situ* stress regime.
- Mine excavations consisting of the optimal empirical mine design parameters (stope and backfill parameters defined above).
- Critical state criteria to evaluate design stability performance.

16.2.6.1 Modelling Approach

Numerical modelling was conducted for a small scale stope model and a large scale mine sector model.

Small-scale Stope Model

A smaller stope scale model was developed to verify the performance of fill sill pillars based on the empirically derived strength parameters. The model consisted of a panel of 4 stopes W and 5 stopes H to simulate the performance of mining under a fill sill pillar

Large-scale Mine Sector Model

The purpose of the large-scale mine model was to accomplish the following activities.

- Evaluate and confirm the proposed mining method.
- Understand performance of backfill on regional deformation.
- Evaluate if the proposed mine sector sequence is viable.
- Assess the evolution of rock mass damage and impact on stoping as mining progresses.

Due to the size and complexity of proposed mining, two large-scale models were constructed using the mining geometry from the Deswik 3D mine design.

- Model 1 – F-Central
- Model 2 – F-South and T Zone

To assess the evolution of rock mass damage and performance of backfill with mining, the modelling was completed in annual excavation steps based on the mining schedule. An annual basis was selected to manage model size and run times.

Modelling steps incorporated stope excavation and then immediate backfill before starting the next excavation step. For simplicity, tight filling is assumed in the model. This resulted in 50 steps for Model 1 (F-Central) and 64 steps for Model 2 (F-South / T Zone).

16.2.6.2 Results

Small-scale Stope Modelling

The main purpose of the small-scale model is to verify the performance of fill sill pillars. The numerical modelling of fill sill pillars was used to model slender / high strength pillars. An example output of the small-scale modelling is shown in Figure 16-13, which shows a vertical cut through fill pillar, and contoured results of safety factor. A results line was taken and plotted in Figure 16-14.

Figure 16-13: Example Output of Small-scale Fill Pillar Model (Safety Factor)

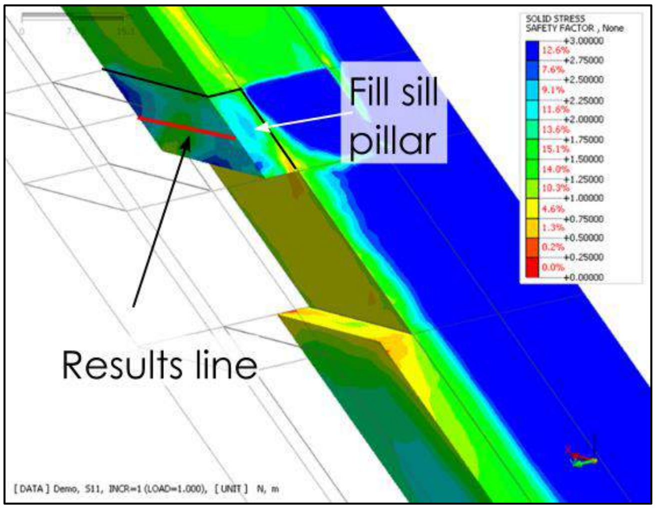
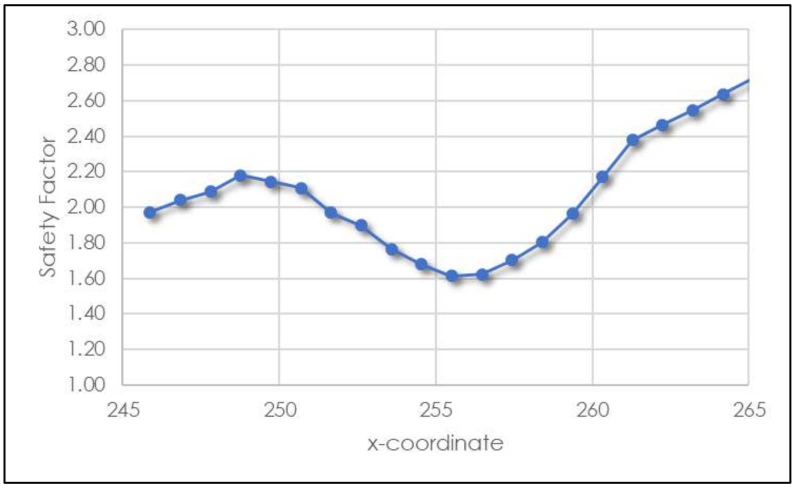


Figure 16-14: Example Output through Pillar Centre



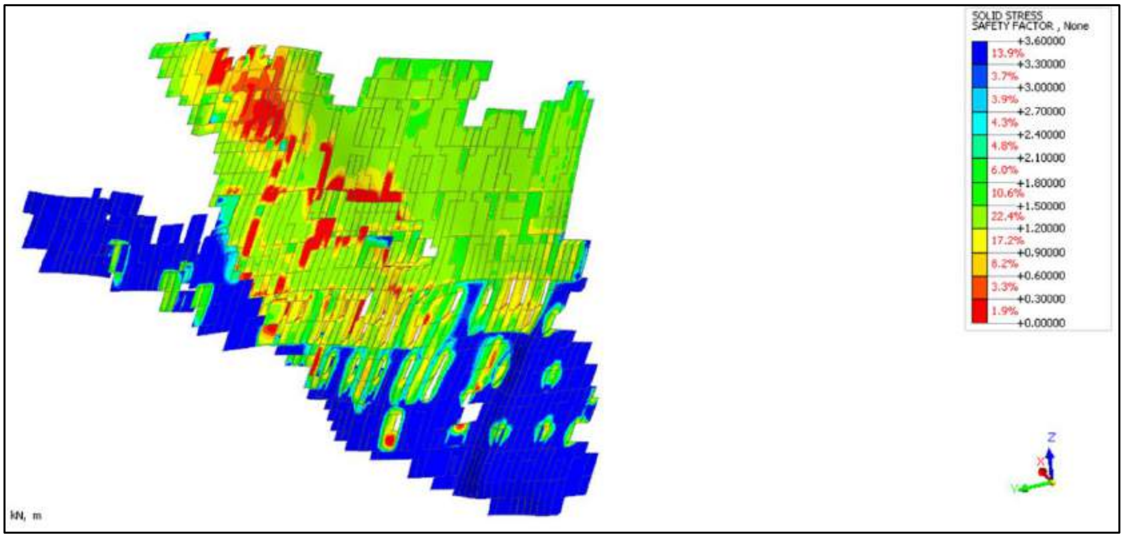
The results indicate that a fill pillar height to stope width (d:L) of 0.5 is likely to be stable yielding an average factor of safety of 2.07. This is based on a 3.8 MPa strength paste backfill. These results are in line with the empirical work and demonstrate that the empirical results are conservative, which was found in other numerical modelling work (Hughes, 2014). During the analysis, it was noted that fill sill pillar performance and stability is influenced by fill stiffness more than strength. It is recommended that elastic properties are collected from laboratory test programmes of future paste backfill investigations.

Large-scale Mine Sector Modelling

Principal findings of the modelling exercise include that complete extraction with paste backfill is achievable and no requirement exists for substantial designed “regional pillars.” Additionally, no major rock mass damage (stopes and rock pillars) was developed above the 300 m Level and moderate to major rock mass damage developed in stope abutments and secondary stope cores towards end of the mine sector sequence, especially below 1 000 m. The risk of rock mass damage and impact to operations can be reduced by optimizing the mining sequence, which should be undertaken during execution. Fill dilution in wider (>40 m) parts of the ore body is expected, principally affecting secondary transverse stopes, which can be mitigated by taking shorter length transverse stopes. The modelling was done in “large” geometrical steps, exacerbating this effect. In general, fill dilution is anticipated to increase with depth and towards completion of the mining level.

An example output of the modelling for F-Central is shown in Figure 16-15. This shows the safety factor at a mining step equivalent to Year 2038 in the mine schedule. The green / blue interface generally represents the mining front, with continuous green mass representing the placed backfill. The red colours inside the upper fill mass indicate over stressing (as this is an elastic model, safety factors can be less than 1.0). Importantly, the modelling generally shows little over stressing of the rock mass at the mining front.

Figure 16-15: Example Output of Safety Factor for Model 1 (F-Central) at Year 2038

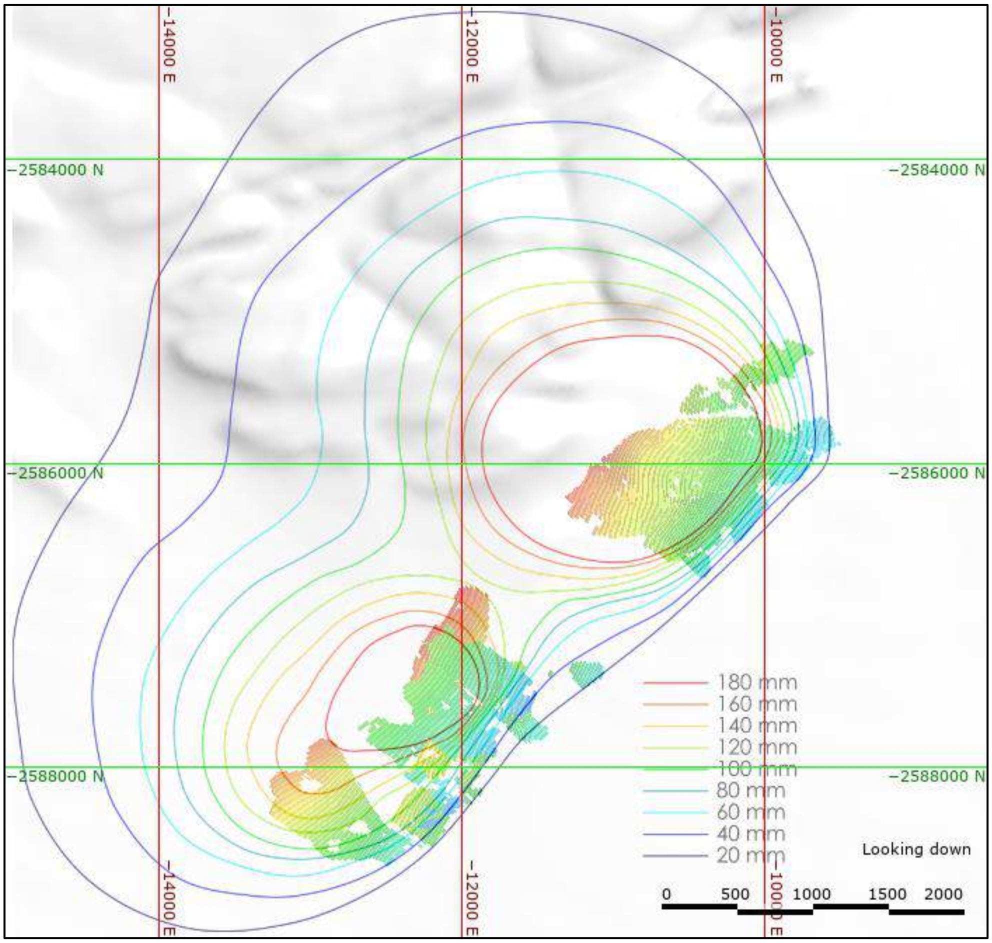


There are some isolated remnant stopes, which, due to the mining sequence, have attracted stress concentrations and, in some cases, overstressing occurred potentially indicating rock mass damage. This could potentially translate to production issues (e.g. delays, higher costs, recovery issues); however, an assessment of the proposed mine sequence indicates that this is isolated.

The results of the numerical modelling exercise were used to develop mine design guidelines on dilution, given stoping method, dimensions and depth below surface.

Based on the elastic analysis, it is estimated that the maximum surface subsidence (at the centre of the fully excavated backfilled mine) will be approximately 35 cm. A combined estimate of surface subsidence for the two models is shown in Figure 16-16.

Figure 16-16: Combined F-Central / F-South Maximum Linear Elastic Surface Displacement Estimate



It must be noted that this maximum subsidence is based on elastic models and, therefore, is relatively conservative as it assumes a complete elastic continuum. There will be some accommodation of deformation and displacement across discontinuities in the rock mass; therefore, total maximum displacements are anticipated to be less. No surface "disturbance" is indicated in the modelling as this maximum displacement is fully recoverable and elastic.

16.2.7 Raisebore Risk Assessment

The mine design includes eleven, 6 m diameter, raisebored ventilation raises to surface. The details of the proposed vent raises are shown in Table 16-21.

Table 16-21: Ventilation Raise Details

Vent Raise	Easting	Northing	Elevation Meters above Sea Level (MASL)	Total Depth (m)	Approximate Depth of Overburden	Minimum Distance to Geotechnical Data (m)	Diameter (m)
NC-1	-7 438	-2 582 911	1054	374	Undetermined	327	6 m
NC-2	-8 361	-2 584 093	1065	245	3.8	188	6 m
NC-3	-8 644	-2 584 185	1055	295	14.2	59	6 m
NC-4	-8 986	-2 585 159	1054	334	11.8	220	6 m
CC-1	-9 530	-2 586 030	1042	270	12.7	130	6 m
CC-2	-10 001	-2 586 395	1040	310	13.0	301	6 m
CC-3	-10 026	-2 586 498	1038	350	20.2	256	6 m
CC-4	-10 312	-2 586 760	1035	350	13.2	156	6 m
SC-1	-11 466	-2 587 503	1016	243	6.5	310	6 m
SC-2	-11 858	-2 587 990	1004	258	5.7	172	6 m
SC-3	-11 934	-2 588 071	1002	440	3.6	72	6 m

Only three of the proposed locations are located reasonably close (within 150 m) to existing geomechanical data. Most raises are further from existing geomechanical data, which reduces the ability to make accurate and reliable assessments of raisebore stability. For these holes, core logs from existing nearby surface diamond drill holes were reviewed and ground conditions were assessed and categorised; however, the information is of insufficient detail to undertake raisebore risk assessments. The review of this core did allow for the depth of over burden to be estimated for each raise. It is recommended that during execution, a geotechnical hole is drilled at each ventilation raise location for further analysis.

The main ventilation raises to surface are all less than 500 m depth, which indicates that the likelihood of stress induced instability will be very low in the more competent / massive rock masses, where UCS values are greater than 125 MPa. This value represents the mean intact rock strength of the near surface MSE domain (sediments), in which the upper sections of all raises will be constructed.

A brief analysis of the potential of stress induced failure was undertaken comparing the estimated maximum tangential stress to the UCS of intact rock to indicate stress induced failure potential (O'Toole & Sidea, 2005) and depth of failure (Martin, Kaiser, & McCreath, 1999). Closed form solutions of stresses around a circular opening (Brady & Brown, 2004) were used to calculate the maximum induced tangential boundary stresses. A summary of the results for a 6 m shaft with an intact UCS of 125 MPa (mean intact strength of the MSE domain) is shown in Table 16-22.

Table 16-22: Shaft Stress Induced Failure Potential Assessment (for UCS of 125 MPa)

Depth (m)	θ_{max}/UCS	Depth of Failure (m)	Damage Class Description
100	0.08	0.00	Little or no Fracturing
200	0.15	0.00	Little or no Fracturing
300	0.22	0.00	Little or no Fracturing
400	0.29	0.00	Little or no Fracturing
500	0.35	0.00	Localised Slabbing
600	0.42	0.10	Localised Slabbing
700	0.49	0.61	Widespread or General Slabbing, Not Very Deep
800	0.56	1.12	Walls Broken into Blocks, Failure of Rock around Excavation
900	0.62	1.62	Walls Broken into Blocks, Failure of Rock around Excavation
1 000	0.69	2.13	Walls Broken into Blocks, Failure of Rock around Excavation
1 100	0.76	2.63	Spalling, Rockburst in Brittle Rock

The results show that, for a UCS of 125 MPa, the potential for stress induced failure of a 6 m diameter raise commences around 500 m depth. More significant damage tends to occur at around 700 m below surface. For intact rock strengths around 200 MPa (the mean intact rock strength of domains below the MSE), the potential for stress induced failure commences at around 800 m depth, with more significant damage at around 1 100 m depth below surface.

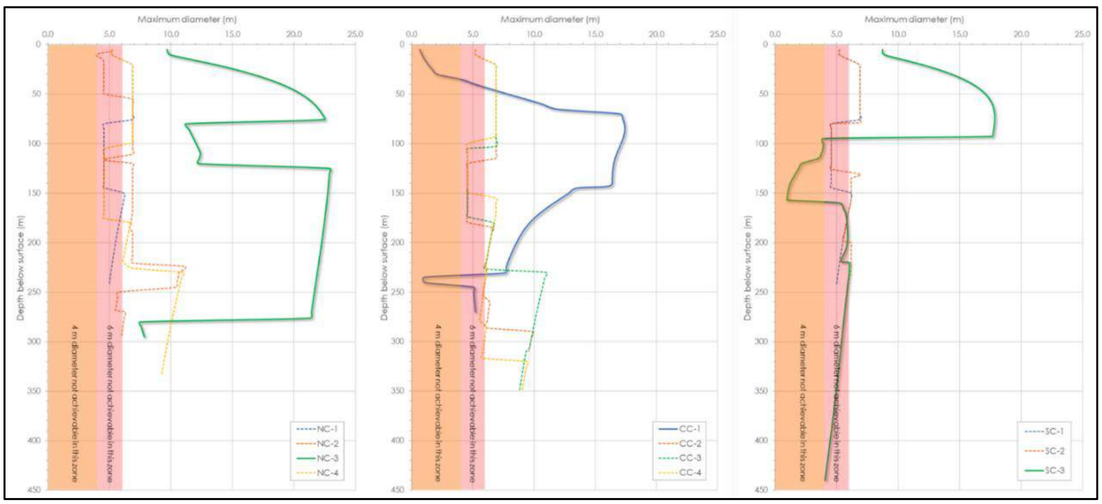
A raisebore assessment was also undertaken using the McCracken and Stacey method (McCracken & Stacey, 1989). It must be noted that the McCracken and Stacey database does not include many large diameter raises. The method, due to its empirical nature, is not a rigorous stability analysis, yet is intended to provide an indication of overall geotechnical feasibility of raisebore diameter given the general geotechnical characteristics.

For the analysis, the location of geomechanical data in relation to the distance to proposed raise locations was evaluated. Where sections of the raise centreline are located within 150 m from existing geomechanical logging data, the logged Q' values were used for the analysis.

Where raises were located more than 150 m from existing geomechanical logging data, a logged value approach cannot be justified and as such the median Q' values for each intersected domain were used in the analysis. It is considered that this approach will lead to less reliable assessments of raisebore risk.

The results for short term instability potential (during raising and prior to installation of support) generally indicate that 4 m raises can be achieved in most proposed shaft locations. However, the analysis indicates that raise instability and complications during raiseboring a 6 m diameter raise will occur in the 20 m below surface for two locations (CC-1 and NC-2). It is considered that special ground improvement pre-support measures will be required during raising for these two shafts, such as grouting and/or contiguous piles to improve near-surface ground conditions. A summary of the results for long-term unsupported instability is shown in Figure 16-17. The thicker lines in the graph indicate where proposed raise locations are reasonably close to geotechnical holes and logged values were used. Apart from raise SC-3 (100-160 m), the analysis undertaken using logged values shows much higher maximum QR values (and larger maximum diameters) values compared to the median QR value domain-based analysis (dashed lines). This highlights the site-specific spatial variability of rock mass conditions and its impact on raisebore risk assessment, and the need to undertake site investigations at each proposed raise location.

Figure 16-17: McCracken and Stacey Maximum Unsupported Diameter Analysis (*RSR*=1.3)



The analysis suggests that unsupported 6 m diameter raises are feasible for most proposed raises; however, there are sections of raise where there is a high risk of instability for unsupported raises.

Apart from CC-1 and SC-3, 6 m diameter raises can be achieved below approximately 200 m below surface for all proposed raise locations. For SC-3, local rock mass fracturing intensity and blockiness in the UZ and MZN domains appears to be driving low maximum unsupported diameter values below 200 m. The results indicate that, to achieve a 6 m diameter, rock reinforcement and ground support will be required to adequately control any potential instability. It is estimated that support in this zone would consist of 2.4 m by 22 mm grouted rebar on a 1.7 m pattern, together with 75 mm fibre-reinforced shotcrete (FRSC). It is also recommended that alternate raise locations be considered, the result of which may avoid the need for rock reinforcement and support for this raise.

A minor problematic zone is identified around 230 m to 250 m in the CC-1 raise and between 100 – 160 m in SC-3 raise, principally related to the UZ domain, which has lower UCS and median Q' values than domains at depth. Although it is considered potentially feasible to develop unsupported diameters of up to 6 m in this zone, it is recommended that these zones are also supported to control any risk of potential instability and rock mass degradation over time. It is estimated that support in this zone would consist of 2.4 m by 22 mm grouted rebar on a 1.8 m pattern, together with 50 mm FRSC. It is also recommended that additional detailed information be obtained at raise locations passing through this zone.

Generally, the upper near surface sections of the MSE domain (sediments) tend to be problematic for long term stability, principally due to the degree of fracturing and bands of lower strength rock. It is considered that potential long-term stability issues in the first 0 m to 40 m can be managed by special ground improvement 'pre-support' measures for all raises, such as grouting, and/or contiguous secant piles to improve near-surface ground conditions.

To mitigate the risk of stress-induced rock mass damage and instability, 4 m diameter twin raises are planned below 800 m.

16.2.8 Rock Reinforcement and Ground Support Recommendations

Rock reinforcement and ground support recommendations were made using empirical based approaches (Barton, Lien, & Lunde, 1974). The support recommendations were developed considering depth, geometry (back spans, wall heights, and intersection widths), purpose, and planned life.

As the NGI Q-System was originally developed for civil engineering purposes, mainly tunnels in Norway, its use in mining may result in over-conservative design recommendations. However, modifications can be made to rationalise the system to provide more appropriate design recommendations for mining (Potvin & Hadjigeorgiou, 2015).

Considering this, the NGI Q-System recommendations were rationalised into the following support categories (Table 16-23).

Based on the excavation group, depth, and domain the rock reinforcement and support recommendations were then developed.

In general, patterned rock bolts and mesh will be required for the majority of excavations to approximately 400-600 m below surface, depending on domain and excavation type. Below 600 m, in some areas, FRSC with fully grouted rebar will be required. Below 800 m, mesh reinforced shotcrete will be required in most excavations. Cable bolting will be required as secondary support in all large excavations and intersections (>7-9 m spans).

Table 16-23: Waterberg Rock Reinforcement and Support Classes

Class	Subclass	Bolt Type	Surface Support	Shotcrete Type	Shotcrete Thickness (mm)
1	1A.1	Split Set (SS) 2.4	Mesh	-	-
	1A.2	Rebar	Mesh	-	-
	1C.2	Rebar	Mesh	Mesh + Shotcrete (SC)	50
3	3B.1	SS 2.4	-	FRSC	50
	3B.2	Rebar	-	FRSC	50
	3C.2	Rebar	Mesh	SC	75
4	4B.1	SS 2.4	-	FRSC	75
	4B.2	Rebar	-	FRSC	75
	4C.2	Rebar	Mesh	SC	100
5	5B.2	Rebar	-	FRSC	100
	5C.2	Rebar	Mesh	SC	150
6	6B.2	Rebar	-	FRSC	150
	6C.2	Rebar	Mesh	SC	200
7	7B.2	Rebar	-	FRSC	150
	7C.2	Rebar	Mesh	SC	200
8	8B.2	Rebar	-	FRSC	150
	8C.2	Rebar	Mesh	SC	150
9	9.C2	Rebar	Mesh	SC	200
10	10.X	Unsupportable	-	-	-
Subclass Legend					
A	Mesh		1		46 mm friction bolts
B	FRSC		2		22 mm rebar
C	SC				

Main Service and Conveyor Declines from Surface

Rock reinforcement and ground support estimates for the main service and conveyor declines from surface have been based on the (Grimstad & Barton, 1993) empirical design method. This empirical approach is a widely accepted as appropriate for mine planning. The estimated sub-surface weathering profile and rock mass conditions have been used to develop the support guidelines.

The principal classes used for the proposed access and conveyor decline systems include 9C.2 for the first 10 m from the portal, 3B.2 in the MSE_M domain, and 1A.2 for the balance of the declines. Due to the permanent nature of the excavations, 2.4 m long, 22 mm diameter grouted (resin, or preferably cement) rebar installed on an approximate 1.5 m pattern are recommended in class 1.A2 and 3B.2.

16.2.9 Conclusions

In Stantec's opinion, an adequate level of geomechanical information was provided to complete a DFS. The analysis completed by Stantec utilised several common empirical models and was validated with numerical modelling in several instances.

The support requirements for the stoping and development headings are in line with both empirical calculation methods and common support types utilised.

A numerical modelling exercise was undertaken to evaluate the evolution of rock mass damage and paste backfill performance of the proposed mining method. The principal findings of the modelling exercise are listed below.

- Continuous extraction with backfill is achievable.
- No requirement exists for substantial designed "regional pillars."
- No major rock mass damage (stopes and rock pillars) was developed above 300 m Level.
- Moderate to major rock mass damage developed in stope abutments and secondary stope cores towards end of the mine sector sequence, especially below 1 000 m.
  - The risk of this and impact to operations can be reduced by optimizing the mining sequence, which should be undertaken during execution.
- Paste backfill dilution in wider (>40 m) parts of the ore body is expected, principally affecting secondary transverse stopes. This may be mitigated by taking shorter length transversal stopes. The modelling was done in "large" geometrical steps, exacerbating this effect.
- In general, paste backfill dilution is anticipated to increase with depth and towards completion of the mining level.

The proposed stope dimensions were evaluated by empirical methods and it was found that in almost all domains and depths the stope dimensions fall on the Stable-Failure line of the Extended Mathews Stability Chart. Proposed hanging walls for stopes within the F Zone, at depths greater than 800 m fall on the Failure-Major Failure line. It is considered that this is acceptable and can be managed during operations with the addition of cable bolt ground support. It will be important to monitor stope reactions and revise the analysis as more detailed geotechnical information is obtained through monitoring programmes to assess design performance during implementation.

16.3 Underground Mining

16.3.1 Introduction

The mining methods and mine design have been modified and optimised from those presented in the PFS. The selection of the longhole mining method and the introduction of paste backfill to the design was based on safety, mitigating geomechanical risk, maximizing Mineral Resource extraction, increasing flexibility and productivity, and low operating costs (with bulk mining). The mining method uses common mechanised equipment and processes widely used in the global mining industry and a comprehensive worker skills training and development programme is included in the operational readiness plan, with ongoing training throughout LOM operations.

16.3.2 Mine Design Parameters

Design criteria and parameters specific to the various aspects of the mining method and mine design are discussed in the appropriate subsections. The following were considered when determining the criteria and parameters during the mine design process.

- Worker health and safety, local communities, and the environment.
- The Mine Health and Safety Act of 1996 (Act No. 29 of 1996).
- Company standards and specifications (industry best practices where company standards and specifications were not available).
- Prevention through design concepts.
- Minimise risk to production.
- Use proven industry technology, equipment, and processes.
- Operational flexibility.
- Operating costs.
- Mineral Resource recovery.

16.3.2.1 Resource Geometry

The Mineral Resources targeted for mining extend from 220 m below surface (North Complex) to approximately 1 280 m below surface (South Complex). The Mineral Resource depth below surface by complex are summarised in Table 16-24. The naming convention for underground sublevels is expressed in approximate metres below surface (i.e. 280 Level is approximately 280 m below surface).

Table 16-24: Mineral Resource Depth Below Surface by Complex

Complex	Top Level	Bottom Level
Central Complex	280 L	1240 L
South Complex	260 L	1280 L
North Complex	220 L	1180 L

The *in situ* and blasted densities for the mineralised zone and waste rock are summarised in Table 16-25.

Table 16-25: Mineralised Zone and Waste Rock Densities

	Mineralised Zone			Waste Rock		
Item	*In Situ* Density tpm³	Swell Factor %	Blasted Density tpm³	*In Situ* Density tpm³	Swell Factor %	Blasted Density tpm³
T Zone	2.90	40	2.07	2.80	40	2.00
F-South	2.93	40	2.09	2.80	40	2.00
F-Central	2.94	40	2.10	2.80	40	2.00
F-North	2.93	40	2.09	2.80	40	2.00
F-Boundary North	2.93	40	2.09	2.80	40	2.00
F-Boundary South	2.93	40	2.09	2.80	40	2.00

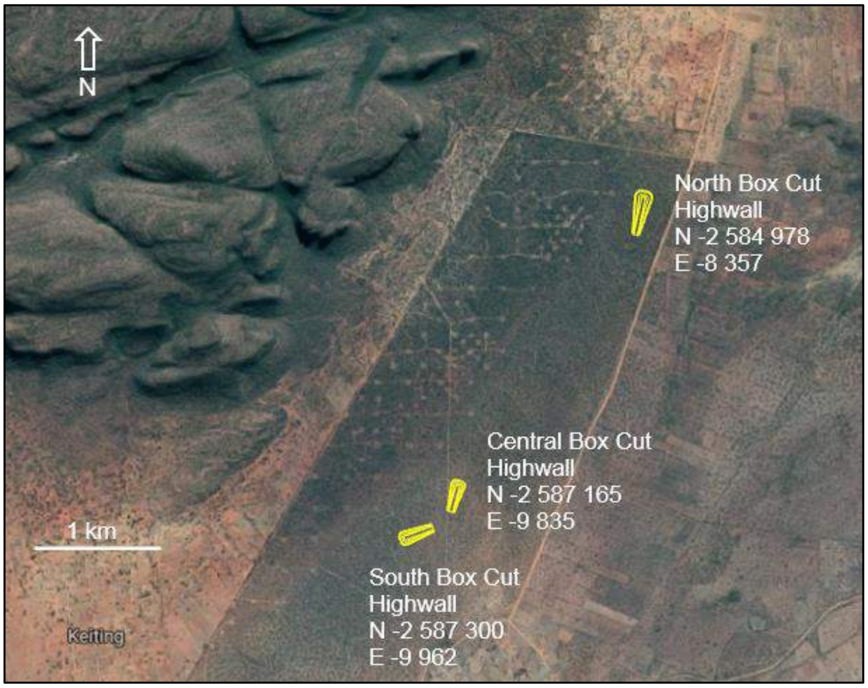
16.3.3 Mine Access

Due to the relatively shallow depth at the top elevations of the Mineral Resource, there will be a box cut and portal constructed at each complex and declines developed to access the Mineral Resource and service the operation for the LOM. Each portal will include a main service decline and a main conveyor decline.

16.3.3.1 Box Cuts and Portals

The portal locations were selected based on surface property agreements, proximity to site infrastructure, proximity to existing settlements, and to minimise the length of decline development required to reach the underground target location at -15.8% (-9^{o}) gradient. The portal locations for each complex are shown on the project site plan view in Figure 16-18.

Figure 16-18: Project Site Plan View Showing Portal Locations



Source: Background – Google Maps

Geotechnical holes were drilled at the box cut and decline locations to investigate the soil and rock characteristics. The programme included geotechnical core logging and laboratory test samples including UCS, Triaxial Compressive Strength, Brazilian Tensile Strength, elastic modulus measurements, and Poisson Ratio measurements.

The following box cut slope angles were used with a factor of safety of 1.5.

- North Box Cut: Highwall height 45 m and slope inclination of 52°
- Central Box Cut: Highwall height 30.8 m and slope inclination of 52°
- South Box Cut: Highwall height 29.4 m and slope inclination of 52°

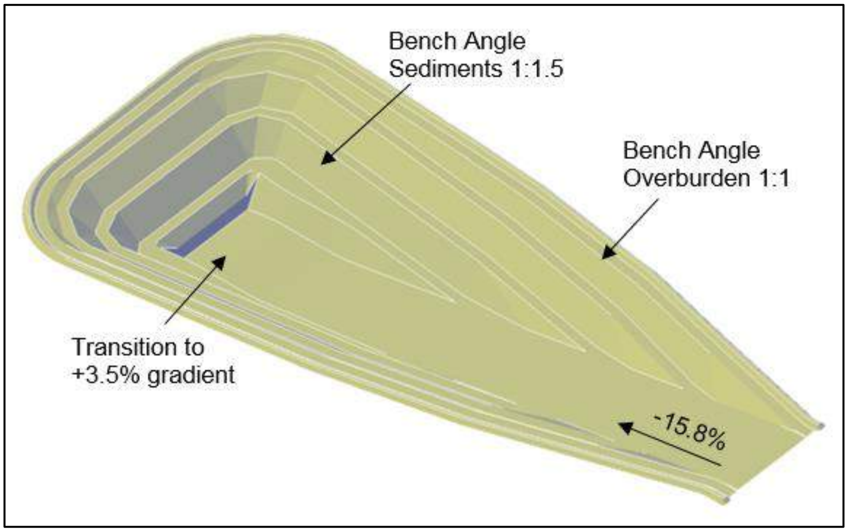
The following design was used for the bench face angles and bench dimensions.

- The bench face angle in loose overburden and cemented overburden is 1:1 (45º from horizontal) with a maximum bench height of 7.5 m and a 3.0 m bench width. The bench width was increased to 3.6 m to allow access for cleaning debris with a small vehicle.
- The bench face angle in Waterberg sediments and granodiorite (intrusive) is 1:1.5 (55º from horizontal) with a maximum bench height of 10 m and a minimum 2.5 m bench width. The bench width was increased to 3.6 m to accommodate a small vehicle for cleaning debris.

Bench face ground support will consist of 6 m long, galvanized, fully threaded 25 mm diameter rock bolts installed on a 1.5 m by 1.5 m staggered pattern. To prevent small pieces of rock from falling from the excavated walls, welded wire mesh will be installed, and a 50 mm thick shotcrete layer will be applied. Geotextile was included for erosion control in the loose sand overburden.

An isometric view of the South Complex box cut model is shown in Figure 16-19. The Central Complex box cut has a similar design.

Figure 16-19: Isometric View of South Complex Portal Box Cut



16.3.3.2 Portal Socket

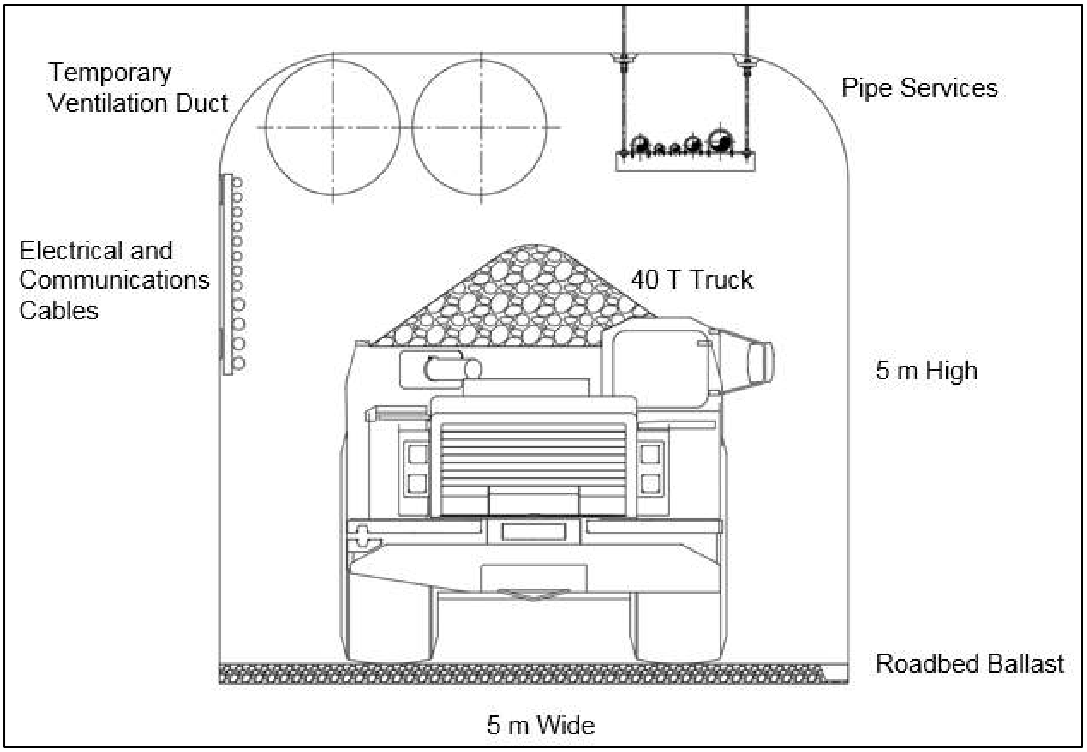
The ground support for the portal socket will include reticulated steel sets installed from the portal face to 10 m into the decline from the face, at 1.0 m spacing. In addition to the steel sets, resin-rebar bolts, welded wire mesh screen, and shotcrete support will be installed in the sockets.

16.3.3.3 Main Service Decline

The main service decline will be the primary access for transferring personnel and material by vehicle between surface and underground and for hauling waste rock to surface. The main service decline profile will be 5.0 m W by 5.0 m H with a 15.8% (9º) gradient. Utility lines installed in this decline will include piping for service water, potable water, mine dewatering, fuel and compressed air, as well as electrical and communications cables. Roadbed ballast material will be provided to maintain a proper driving surface. During the development stage, temporary 1 220 mm diameter ventilation ductwork will be suspended from the back, and the drift profile will accommodate a loaded 40-t class haul truck. When the ventilation ductwork is removed, this drift will accommodate a loaded 50-t class haul truck. The main service declines will be developed parallel and concurrently with the conveyor declines to establish a ventilation loop and synergies with equipment and labour during development. There will be a 15 m pillar (rib to rib) separating the two declines and connections between the declines will be made at 75 m intervals to establish the ventilation loop and to provide access for transfer equipment and personnel between the headings.

The main service decline profile is shown in Figure 16-20.

Figure 16-20: Main Service Decline Profile

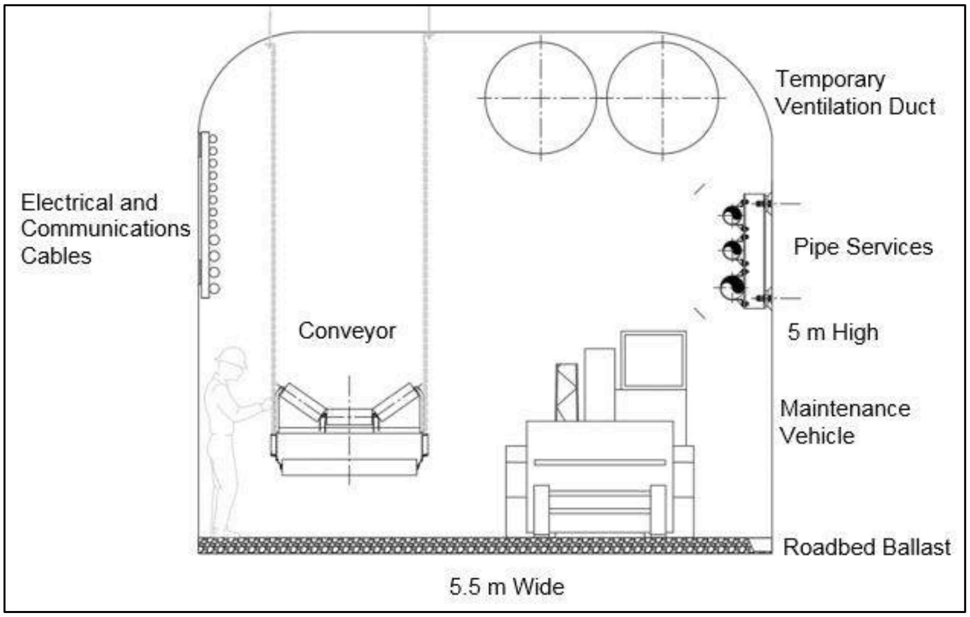


16.3.3.4 Main Conveyor Decline

The main conveyor decline will be equipped with a conveyor to transfer ore to surface. The profile will be 5.5 m W by 5.0 m H with a 15.8% (9°) gradient. The decline cross-section will include space to accommodate mobile equipment required for maintenance, cleaning, and inspection of the conveyor system. During development, temporary services will be installed in the decline, including service water and dewatering piping. Permanent services installed in the decline will include piping for dewatering and fire water and electrical and communications cables. Roadbed ballast material will be provided to maintain a proper driving surface. During the development stage, temporary 1 220 mm diameter ventilation ductwork will be installed from the drift back and the resulting profile will accommodate a loaded 40-t class haul truck.

The conveyor decline profile is shown in Figure 16-21.

Figure 16-21: Conveyor Decline Profile



16.3.4 Development Methods

All decline and lateral excavations will be developed using drill and blast methods and diesel-powered mobile equipment. The mobile equipment required for development activities is listed below.

- Drill – 2-Boom Electric-Hydraulic Jumbo
- Blast – Mobile Explosives Loader
- Muck – 17-t Class LHD
- Haul – 40-t Class Haul Truck
- Ground Support Installation – Mechanical Bolter

There will be four main development heading profiles for the underground workings as summarised in Table 16-26. For larger infrastructure excavations (such as conveyor transfer stations, rock breaker stations, shops, etc.) general arrangement drawings were prepared and the excavation dimensions incorporated into the 3D mine model. For these excavations, initial pilot drifts will be developed, and a combination of wall slashing, floor benching, and back-slashing will be used to achieve the final dimensions.

Table 16-26: Main Development Heading Profiles

Heading Profile	Notes
5.0 m W x 5.0 m H Arched	Service Decline and Lateral Waste Rock Headings
5.5 m W x 5.0 m H Arched	Conveyor Declines
6.0 m W x 5.0 m H Arched	Ore sills / Crosscuts in Stopes Greater than or equal to 9 m W
5.0 m W x 4.0 m H Arched	Ore Sills in Stopes less than 9 m W

16.3.4.1 Development Drilling

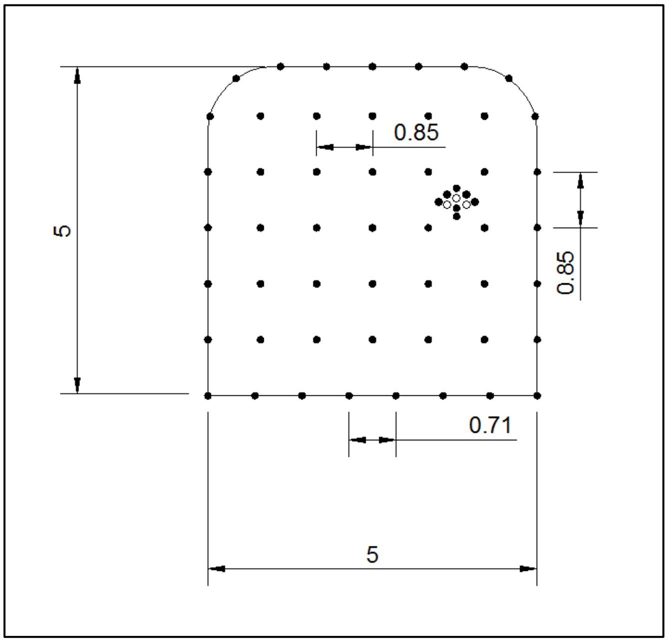
Development rounds will be drilled using a 2-Boom Electric-Hydraulic Jumbo. The development drilling designs are summarised in Table 16-27.

Table 16-27: Development Drilling Design

Item	5 m W x 5 m H	5.5 m W x 5 m H	6 m W x 5 m H	5 m W x 4 m H
Drill Depth	4.4 m	4.4 m	4.4 m	4.4 m
Break per Round	3.8 m	3.8 m	3.8 m	3.8 m
Over-break Allowance	10%	10%	10%	10%
Hole Diameter	45 mm	45 mm	45 mm	45 mm
Hole Burden	0.85 m	0.85 m	0.85 m	0.85 m
Hole Spacing	0.85 m	0.85 m	0.85 m	0.85 m
Hole Spacing – Lifters	0.71 m	0.69 m	0.75 m	0.69 m
Total Holes Drilled	60 holes	66 holes	69 holes	53 holes
Holes Reamed for Cut	3 holes	3 holes	3 holes	3 holes

An example of the drilling pattern for the 5 m W x 5 m H heading type is shown in Figure 16-22.

Figure 16-22: Drilling Pattern for a 5 m x 5 m Heading



16.3.4.2 Blasting

Development rounds will be loaded using a mobile mechanical explosives loader. The development blasting design basis is summarised in Table 16-28.

Table 16-28: Development Blasting Design Basis

Item	Comment
Explosives Type	Bulk Emulsion (1 150 kg/m^3)
Perimeter Control Blasting (Back Holes)	Specialty Packaged Explosive
Detonator	Non-electric Detonator
Initiation	Electric Cap and Detonator Cord Mine-wide Central Blasting

16.3.4.3 Development Mucking

Development rounds will be mucked using a 17-t class LHD. The LHD will muck blasted rock from the face to a remuck bay and subsequently remuck the rock and load a haul truck. For long development drives, remuck bays will be spaced 150 m apart, resulting in an average tramming distance of 75 m. The design basis for development mucking are summarised in Table 16-29.

Table 16-29: Development Mucking Design

Item	Value
Bucket Capacity (SAE Heaped)	8.6 m^3
Bucket Fill Factor	80%
Average Tramming Distance	75 m
Average Tramming Speed	6.5 km/hr
Load Bucket	90 sec
Position and Dump	60 sec

16.3.4.4 Ground Support Installation

Ground support installation will be completed using a mechanical bolter. Ground support requirements were identified for various rock domains that will be encountered. To minimise the inventory of ground support materials and to promote consistency and quality control with ground support installation a common primary ground support that will be accommodate most ground conditions encountered was selected. The primary ground support will include 2.4 m long resin rebar installed on a 1.5 m by 1.5 m staggered pattern with welded-wire mesh screen installed on the back, shoulders and walls to within 1.25 m of the floor. An allowance for shotcrete application to 10% of all development as part of primary ground support was included to accommodate local poor-quality ground. In addition, further allowance for shotcrete as secondary support to 10% of all development in waste rock is included.

Secondary ground support consisting of cable bolts will be applied to larger spans at intersections and infrastructure excavations. Where possible, four-way intersections will be avoided in the mine design. At intersections, there will be 6 m long cable bolts installed on a 2.5 m x 2.5 m pattern.

16.3.5 Vertical Development

Vertical raise development will consist primarily of ventilation raises and will be constructed using raiseboring methods carried out by a qualified mining contractor.

16.3.5.1 Surface Ventilation Raises

The main fresh air and return air raises to surface will be 6.0 m in diameter. The collar for each raise will require pre-supporting through a layer of loose sand overburden and a layer of weathered sediments that are highly fractured and of low strength. The pre-supported collar will be established by constructing a ring of concrete secant piles. The secant piles will also provide the foundation for the raisebore setup and the base for ventilation duct installation. The depth of secant piling for each raise was determined from core logging data from nearby diamond drill holes and are summarised in Table 16-30.

Table 16-30: Surface Ventilation Raise Collar Secant Pile Depth

Raise	Secant Pile Depth (m)
CC-1	20.0
CC-2	21.0
CC-3	21.3
CC-4	25.9
SC-1	40.1
SC-2	8.6
SC-3	16.2
NC-1	40.0
NC-2	40.8
NC-3	60.5
NC-4	42.0

The piling depths are deeper for ventilation raises in the North Complex due to the thickness of the weathered sediments and become shallower toward the South Complex.

16.3.5.2 Underground Internal Ventilation Raises

Internal ventilation raises will be raisebored and will connect to each production level. Internal ventilation raises above 800 Level will be 6.0 m diameter, while below 800 Level twin 4.0 m diameter raises will be used (based on geomechanical factors). The underground internal ventilation raise accesses will include a station for raisebore set-up and gear and rod storage. Internal ventilation raises that are equipped with an escapeway for egress will include ground support.

16.3.6 Mining Method Selection

At the start of the DFS, an initial mine design was prepared for LSLOS without backfill, with permanent sill and rib pillars left in place to maintain overall rock mass stability. As this mine design progressed, the low extraction ratio (due to the required size of the permanent sill and rib pillars) and identified geomechanical risks, initiated evaluations around changing to a mining method and mine design that includes backfill. Based on the evaluations, the introduction of paste backfill was identified to significantly mitigate geomechanical risks, improve confidence that the longhole mining method will be successful in execution, and make practical the achievement of the following additional benefits.

- Increased percentage of extracted Mineral Resource.
- Increase in ore production rate.

- Increased mine life.
- Improved economics (arising from the increases in extraction percentage, production rate, and mine life).
- Reduced volume of tailings in the TSF.

The mine design for the DFS is based on the Longhole with paste backfill mining method.

16.3.6.1 Sublevel Longhole Stoping with Backfill Mining Method

A combination of transverse and longitudinal Longhole approaches will be used to extract the Mineral Resource. Longhole requires dividing the Mineral Resource targeted for production into individual stopes and establishing mining sublevels to access the stopes and position development to facilitate drilling, blasting, and extracting the blasted material from between the sublevels. Once mining of a stope is complete, the stope will be backfilled with paste backfill. Longhole is a non-entry method, meaning that during mining, personnel will be prohibited from entering the open portion of a stope.

A transverse approach consisting of primary and secondary stopes will be applied to areas where the average true thickness (perpendicular to dip) of the Mineral Resource is 15 m or greater. In the transverse approach, stopes are accessed and developed perpendicular to the strike of the ore body. For areas where the true thickness is less than 15 m, a longitudinal approach requiring less waste rock development will be used. In the longitudinal approach, stopes are developed along (i.e. parallel) the strike of the ore body.

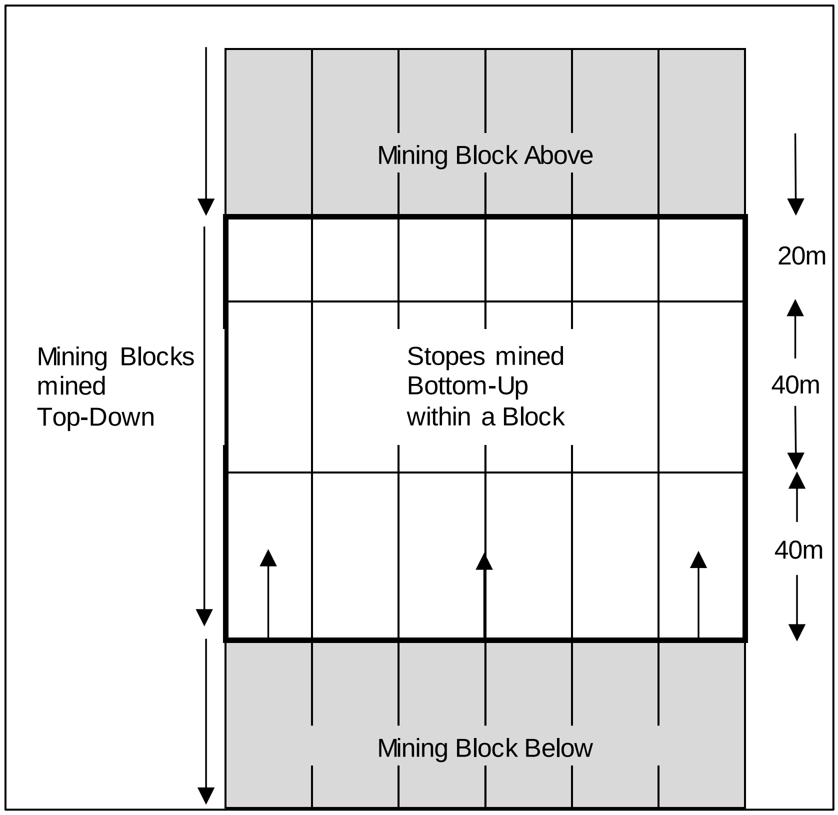
16.3.6.2 Sublevel Interval

The sublevel interval was evaluated and considered rock mechanics empirical design methods for excavation stability, the Mineral Resource geometry, stope productivity, and optimization of the waste rock to ore ratio. Specialty mine design software MSO was used to generate stope shapes at 20 m and 40 m vertical intervals. The 20 m vertical sublevel spacing was considered the minimum spacing to use when mining approaches a mined and backfilled stope block above, while a 40 m vertical interval was considered the maximum based on the production drilling hole length when accounting for drilling holes along the dip of the ore body. For the 40 m vertical interval stopes, to maintain maximum hole lengths at 30 m or less, production drilling will consist of uphole drilling from the bottom sill of the stope and downhole drilling from the top sill of the stope.

16.3.6.3 Mining Blocks

To achieve the planned production rate, simultaneous production will be required from multiple mining fronts. To establish multiple fronts, mining blocks will be established at 100 m vertical intervals. The 100 m vertical blocks will consist of two 40 m vertical H stopes (each stope drilled up and down) and one 20 m H vertical uppers stope that will be mined up to the backfilled stopes in the block above as shown in Figure 16-23. The mining block and/or stope heights may be adjusted to accommodate Mineral Resource geometry in certain areas. Within a mining block, stoping will progress from bottom-up, but the overall mining of blocks will progress top-down.

Figure 16-23: 100 m Vertical Mining Block



Each 100 m mining block will consist of a 40 m H bottom stope, a 40 m H middle stope, and a 20 m H top stope as shown in Figure 16-24.

The sequence of mining the bottom, middle, and top stope are summarised in Figure 16-25. The sill drifts for the middle stope and top stope will require ground support rehabilitation for re-entering once the stope is backfilled. This rehabilitation is anticipated to primarily be around the slot area of the stope.

Figure 16-24: 100 m Mining Block Stopes

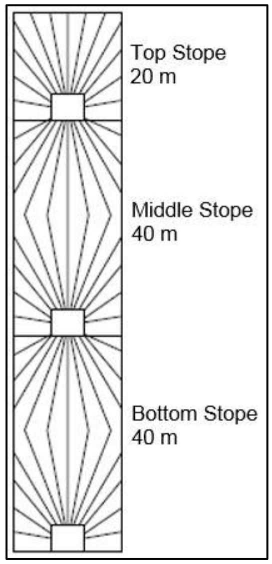
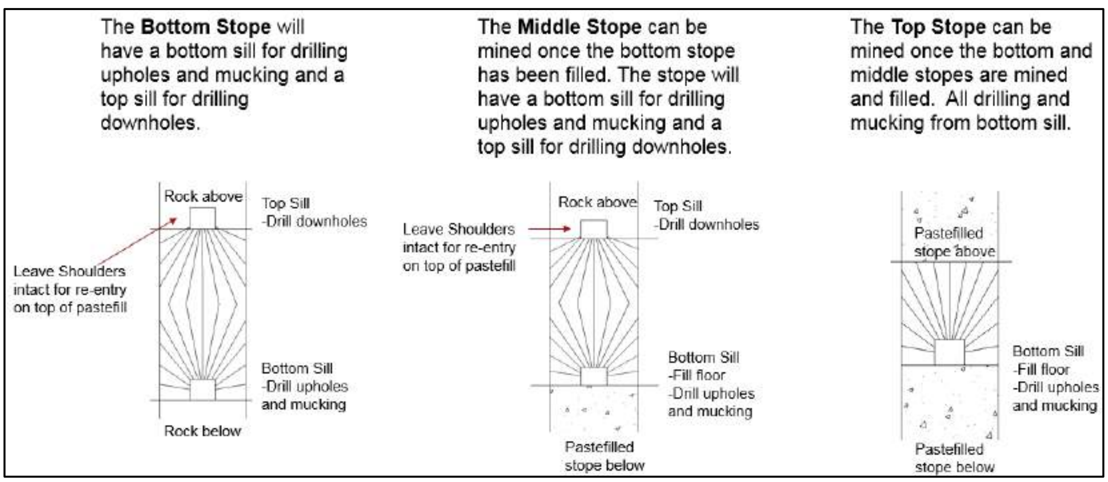


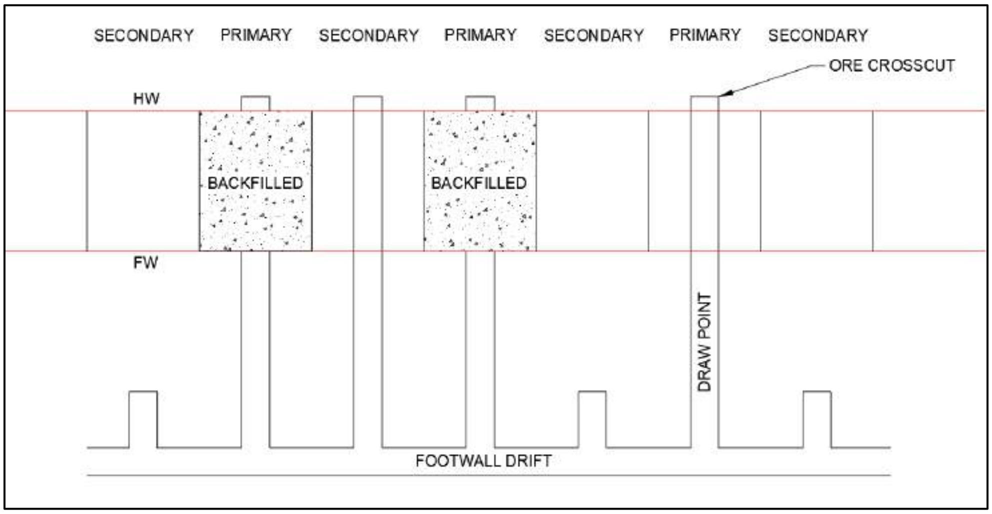
Figure 16-25: Bottom, Middle, and Top Stope Sequence



16.3.6.4 Transverse Longhole

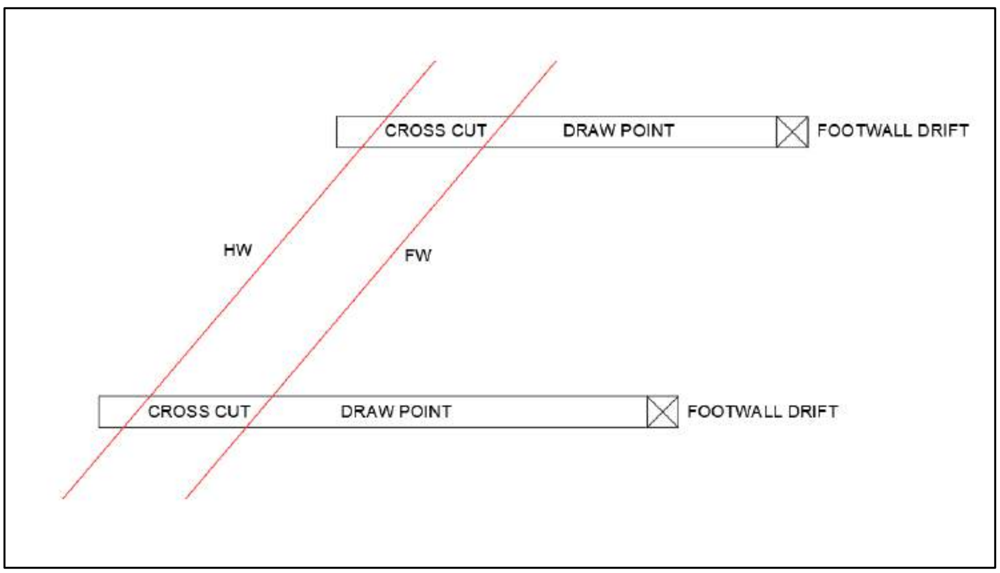
For Transverse Longhole Stoping, a drift will be established in the footwall (footwall drift) parallel to the strike of the ore body on each sublevel. Primary and secondary stopes will be defined at 20 m W intervals along strike and each stope will be accessed from the footwall drift with a crosscut developed through the centre of the stope from the footwall to the hanging wall. The mining of the stope will progress from the hanging wall to the footwall. A simplified level plan showing a series of primary and secondary transverse stopes along strike is shown in Figure 16-26.

Figure 16-26: Simplified Level Plan – Transverse Longhole



A simplified section view through a transverse longhole stope is shown in Figure 16-27.

Figure 16-27: Simplified Section View – Transverse Longhole



The design parameters for transverse stopes is summarised in Table 16-31.

Table 16-31: Transverse Longhole Stope Design Parameters

Item	Parameter
Maximum Stope Height (vertical)	40 m
Primary Stope Width (along strike)	20 m
Secondary Stope Width (along strike)	20 m
Minimum Stope True Thickness (hanging wall to footwall)	15 m*
Minimum Inclination of Stope Footwall	38.0º
Stope Access / Drawpoint Dimensions	5.0 m W x 5.0 m H
Stope Ore Crosscut Dimensions	6.0 m W x 5.0 m H

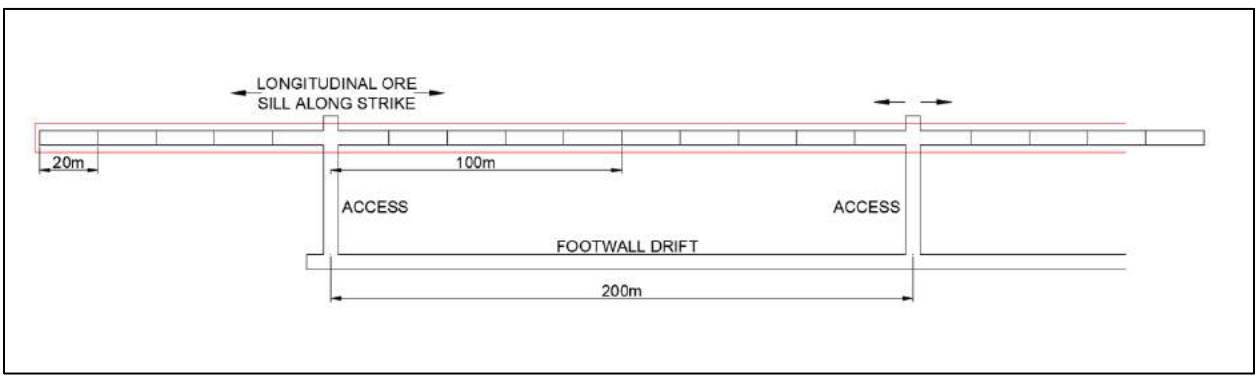
Note:

- *Some transverse stopes may be less than 15 m true thickness.

16.3.6.5 Longitudinal Longhole

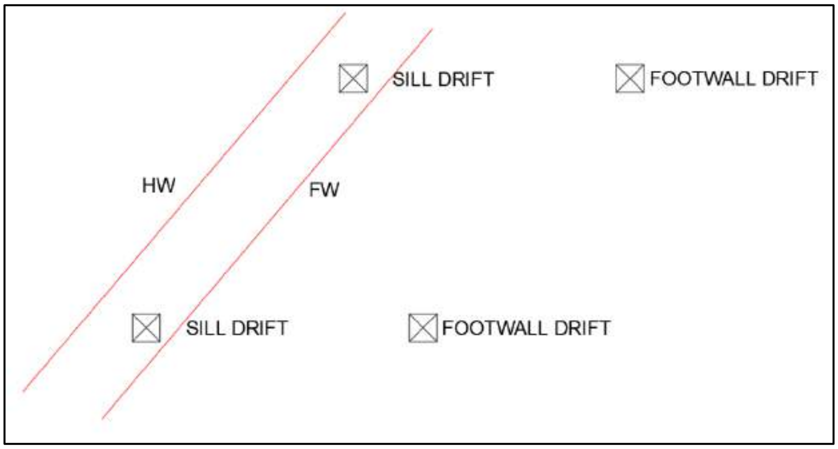
A longitudinal approach will be used in areas where the true thickness of stopes averages less than 15 m. Similar to transverse mining, sublevels in longitudinal areas will require a footwall drift; however, rather than access each individual stope, access to the Mineral Resource will be developed at approximately 200 m intervals along strike. From the access, a sill drift will be developed in each direction along the strike of the ore body through a series of stopes as shown in Figure 16-28. Stoping will start at the end of each sill and retreat to the access. Each stope will be 20 m along strike and then backfilled prior to mining the adjacent 20 m stope. Although ground quality will allow for opening longer longitudinal stopes along strike, the sequence and schedule have been based on 20 m. This will allow sequencing flexibility, limit remote mucking distances, and the frequent stope 're-start' will reduce losses on the footwall. As the operation gains experience, there may be an opportunity to increase the strike length of individual stopes.

Figure 16-28: Simplified Level Plan – Longitudinal Longhole



A simplified section view through a longitudinal longhole stope is shown in Figure 16-29.

Figure 16-29: Simplified Section View – Longitudinal Longhole



The design parameters for longitudinal stopes is summarised in Table 16-32.

Table 16-32: Longitudinal Longhole Stope Design Parameters

Item	Criteria
Ore Sill Access Spacing (typical)	200 m
Maximum Stope Height (vertical)	40 m
Maximum Stope Length (along strike)	20 m
Maximum Stope True Thickness (hanging wall to footwall)	15 m*
Minimum Stope True Thickness (hanging wall to footwall)	2.4 m
Minimum Inclination of Stope Footwall	38.0º
Stope Access / Drawpoint Dimensions	5.0 m W x 5.0 m H
Ore Sill Dimensions (up to 6.0 m true thickness)	5.0 m W x 4.0 m H
Ore Sill Dimensions (greater than 6.0 m true thickness)	6.0 m W x 5.0 m H

Note:

- * Some longitudinal stopes may exceed 15 m true thickness.

16.3.7 Stoping

The height (40 m and 20 m) and strike length (20 m) of stopes will generally be consistent throughout the Mineral Resource; however, the true thickness of stopes (from hanging wall to footwall) will vary. Stope thickness data from the 3D mine model was used to generate eight representative stope sizes that were used to estimate stope cycles and productivities. The representative stope sizes are summarised in Table 16-33.

Table 16-33: Representative Stope Sizes

	Stope Height	Thickness Range	Thickness Used
Transverse	40 m	15 m to 30 m	21 m
Transverse	20 m	15 m to 30 m	21 m
Transverse	40 m	+30 m	48 m
Transverse	20 m	+30 m	48 m
Longitudinal	40 m	South Complex 2.4 m to 4 m	3 m
Longitudinal	20 m	South Complex 2.4 m to 4 m	3 m
Longitudinal	40 m	3 m to 15 m	8 m
Longitudinal	20 m	3 m to 15 m	8 m

Stoping activities include slot raise drilling, production drilling, production blasting, mucking, and backfilling.

16.3.7.1 Slot Raise Drilling

Slot raises will be drilled using an in-the-hole (ITH) drill and a Machine Roger V30 reaming head (or similar) for blind boring upholes and down reaming. An initial pilot hole will be drilled and reamed followed by the installation of the reaming head and a second pass of reaming to the final dimension of 760 mm (30 inches).

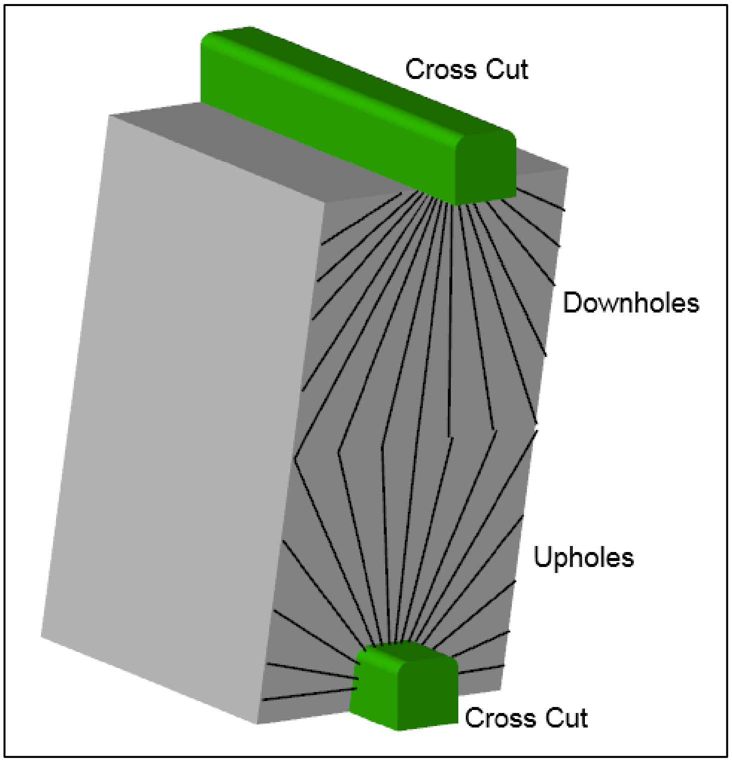
16.3.7.2 Production Drilling

Production drilling will be completed using electric-hydraulic top-hammer drills. The top-hammer drill was selected due to high penetration rates and suitability for 76 mm diameter holes that are 30 m or less in length. A combination of uphole drilling and downhole drilling will be used. The maximum production hole length will be approximately 30 m downholes in longitudinal stopes. The hole diameter will be 76 mm and the average hole length will be approximately 17.0 m. The 76 mm hole diameter can be applied to narrow longitudinal stopes and larger transverse stopes.

The production drills will be equipped with control systems and automated functions that improve safety, hole placement accuracy and precision, and drill productivity. Information (hole dip, dump, and length) from drilling designs provided by mine engineering will be programmed into the drill. Proper drill ring survey and initial drill set-up on a ring will be critical to achieve proper drilling results. Mine surveyors have been included in the labour to support production drilling. During drilling operations, quality checks on ring mark-up, drill set-up, hole accuracy (collar location, dip, azimuth), and breakthroughs will be conducted by mine engineering technicians. The estimated drilling rate for a drill is approximately 1 700 tonnes drilled per day.

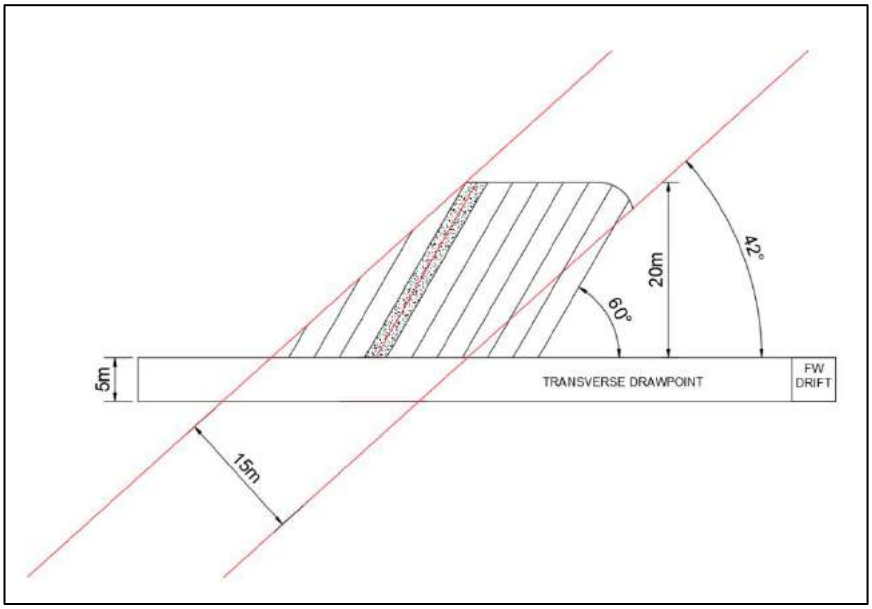
For the 40 m vertical H stopes, to reduce the hole length and potential for deviation, upholes will be drilled from the bottom sill of the stope and downholes drilled from the top sill. The uphole and downhole drilling concept in a transverse stope is demonstrated in Figure 16-30.

Figure 16-30: Uphole and Downhole Production Drilling



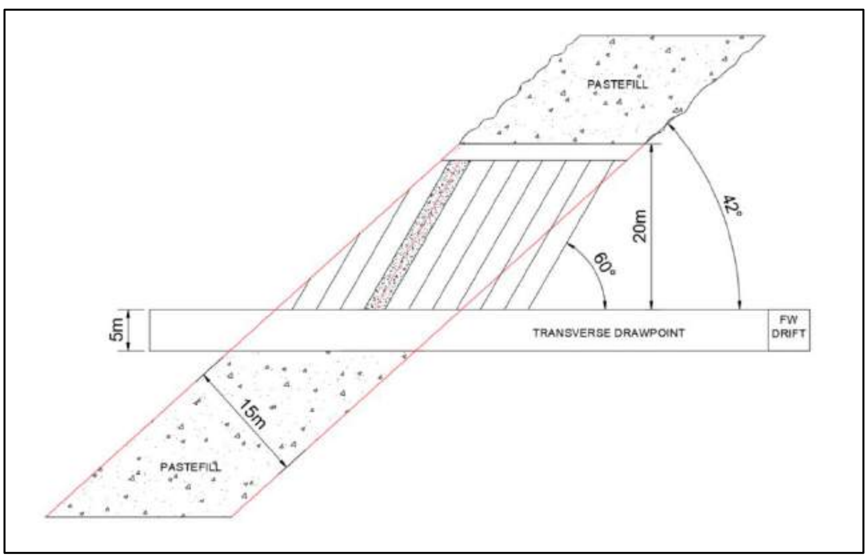
For transverse stopes, the uphole production rings will be designed at a 60º angle as seen in Figure 16-31 to mitigate the potential for an unstable intermediate brow that could be created if the production holes are drilled parallel to the dip of the stope. For holes that are collared in waste rock, only the portion of the holes in ore will be blasted, as determined by the planned stope limits.

Figure 16-31: Uphole Production Rings at 60º



For the 20 m vertical H stopes, upholes will be drilled from the bottom sill of the stope and drilled short of breaking through into the paste backfilled stope from the block above to minimise paste backfill dilution from the exposed back. The uphole drilling in a 20 m transverse uppers stope is demonstrated in Figure 16-32.

Figure 16-32: Transverse 20 m Uppers Drilling



Production drilling rings for the representative stope sizes were prepared to determine the drilling quantities and drill factors. An example of the drill rings for a 40 m transverse stope is shown in Figure 16-33 and the typical drilling on a ring is shown in Figure 16-34. The production drilling design parameters are summarised in Table 16-34 and Table 16-35.

Figure 16-33: Transverse Production Rings

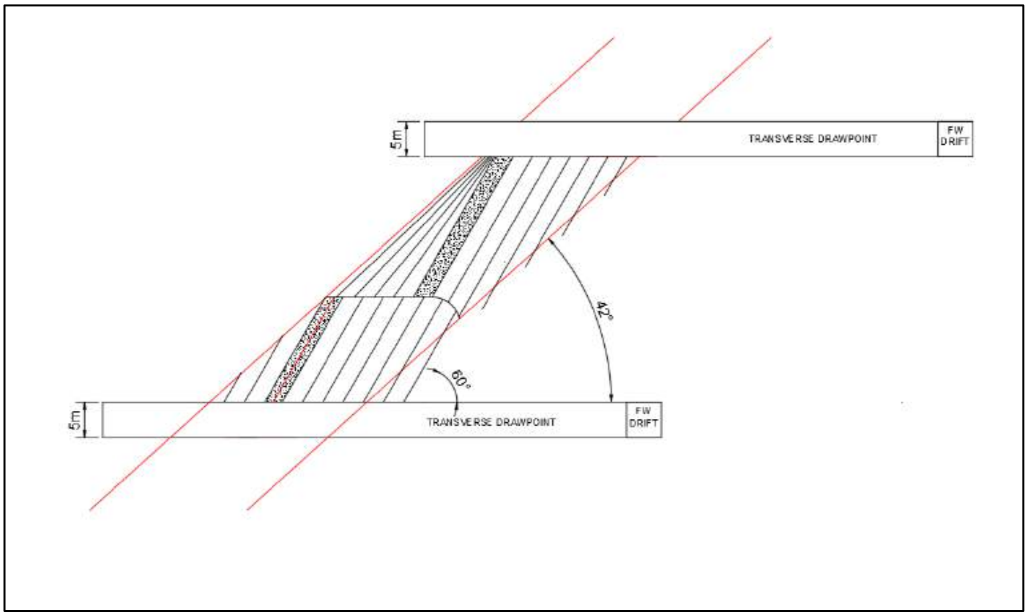


Figure 16-34: Typical Production Drilling Ring (along 60° ring dip) 40 m Transverse Stope

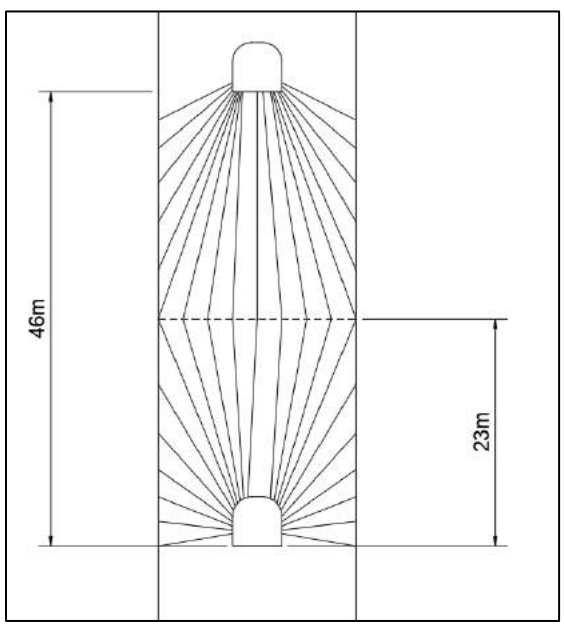


Table 16-34: Transverse Stope Production Drilling Parameters

Item	Transverse 40 m H 21 m Thick	Transverse 20 m H 21 m Thick	Transverse 40 m H 48 m Thick	Transverse 20 m H 48 m Thick
Hole Diameter	76 mm	76 mm	76 mm	76 mm
Ring Spacing	2.2 m	2.2 m	2.2 m	2.2 m
Hole Burden	2.5 m	2.5 m	2.5 m	2.5 m
Total Drilling	8 456 m	3 972 m	18 156 m	8 708 m
Stope Tonnes	67 000 t	32 200 t	149 200 t	71 700 t
Drill Factor	7.9 tpm	8.1 tpm	8.2 tpm	8.2 tpm
Average Hole Length	17 m	14 m	17 m	14 m

Table 16-35: Longitudinal Stope Production Drilling Parameters

Item	Longitudinal 40 m H 8 m Thick	Longitudinal 20 m H 8 m Thick	Longitudinal 40 m H 3 m Thick	Longitudinal 20 m H 3 m Thick
Hole Diameter	76 mm	76 mm	76 mm	76 mm
Ring Spacing	2.2 m	2.2 m	2.2 m	2.2 m
Hole Burden	2.5 m	2.5 m	2.5 m	2.5 m
Total Drilling	2 867 m	1 313 m	1 670 m	725 m
Stope Tonnes	26 600 t	12 400 t	11 400 t	5 100 t
Drill Factor	9.3 tpm	9.4 tpm	6.8 tpm	7.0 tpm
Average Hole Length	17 m	13 m	27 m	23 m

16.3.7.3 Longhole Blasting

Bulk emulsion will be used for production blasting. A mobile emulsion loading unit will be used to load the holes. The production blasting design basis is summarised in Table 16-36.

Table 16-36: Longhole Blasting Parameters

Item	Parameter
Explosives Type	Bulk Emulsion (Density 1 150 kg/m^3)
Detonator	Non-electric Detonator
Initiation	Electric Cap and Detonator Cord Mine-wide Central Blast System

The estimated powder factor for each typical stope size is summarised in Table 16-37 and Table 16-38.

Table 16-37: Transverse Longhole Powder Factor

Item	Transverse 40 m H 21 m Thick	Transverse 20 m H 21 m Thick	Transverse 40 m H 48 m Thick	Transverse 20 m H 48 m Thick
Hole Diameter	76 mm	76 mm	76 mm	76 mm
Total Drilling	8 456 m	3 972 m	18 156 m	8 708 m
Loaded Length	5 083 m	2 390 m	10 920 m	5 244 m
Total Emulsion	27 846 kg	13 092 kg	59 816 kg	27 358 kg
Stope Tonnes	67 000 t	32 200 t	149 200 t	71 700 t
Powder Factor	0.42 kg/t	0.41 kg/t	0.40 kg/t	0.38 kg/t

Table 16-38: Longitudinal Longhole Powder Factor

Item	Longitudinal 40 m H 8 m Thick	Longitudinal 20 m H 8 m Thick	Longitudinal 40 m H 3 m Thick	Longitudinal 20 m H 3 m Thick
Hole Diameter	76 mm	76 mm	76 mm	76 mm
Total Drilling	2 867 m	1 313 m	1 670 m	725 m
Loaded Length	1 724 m	790 m	1 005 m	436 m
Total Emulsion	9 448 kg	4 327 kg	5 504 kg	2 388 kg
Stope Tonnes	26 600 t	12 400 t	11 400 t	5 100 t
Powder Factor	0.36 kg/t	0.35 kg/t	0.48 kg/t	0.46 kg/t

16.3.7.4 Production Mucking

Blasted ore will be mucked from stopes using 17-t class LHDs. When the stope brow is closed the LHD will be operated with the operator in the cab. When the stope brow is open, the LHD will be operated on remote control with the operator stationed at a remote stand located a safe distance from the brow and away from the path of the moving LHD. The LHD will tram and dump into a remuck bay located within 150 m of the stope drawpoint. A second LHD dedicated to truck loading will re-handle the ore to load the trucks (to decouple stope mucking from truck haulage). The height of the drift at the truck loading area will accommodate the truck loading. The design parameters related to mucking are summarised in Table 16-39.

Table 16-39: Production Mucking Parameters

Item	Value
Bucket Volume (SAE heaped)	8.60 m^3
Bucket Fill Factor	80.00%
Actual Bucket Capacity	6.90 m^3
Ore SG *In Situ*	2.94 tpm^3
Swell Factor	40.00%
Broken Ore SG	2.09 tpm^3
Payload	14.40 t
Average Tramming Speed	6.00 km/hour
Average Tramming Distance to Remuck Bay	150.00 m
Mucking Cycle Time per Bucket	6.50 min
Mucking Fixed Time per Shift	25.00 min
Mucking Productivity per Day	1 600.00 tpd

16.3.7.5 Stope Results Evaluation

Following the completion of mucking and prior to backfilling, the empty stope cavity will be surveyed (i.e. 3D scanned image of the void) and mine engineering / geology will evaluate stope results versus the planned design (i.e. tonnes mined, external dilution, and recovery / ore left) and reconcile the grade of the stope versus the planned and sampled grades. This reconciliation exercise will allow the operation to adjust the stoping process as part of an overall site continuous improvement programme. The stope cavity survey will also be used for mine planning for adjacent stopes.

16.3.7.6 Backfill Cycle

A backfill barricade will be constructed at the stope drawpoint to contain the initial paste backfill plug poured. The barricade design will have drainage piping to allow stope decant water to drain and relieve pressure build-up in the stope.

The backfill component of the stope cycle is summarised in Table 16-40.

Table 16-40: Backfill Cycle Parameters

Item	Value
Backfill Barricade Construction	5 days
Paste Backfill Availability*	50%
Plug Cure time	3 days

Note:

- *Assumes the paste backfill plant is available, but a pour is occurring in another stope.

16.3.8 Mining Development

Each complex will have sublevels at 40 m and 20 m intervals. Due to the strike length of the ore body, sublevels may be accessed by more than one service decline. A long section view of the Central Complex showing the sublevels is shown in Figure 16-35. A long section view of the South Complex showing the sublevels is shown in Figure 16-36. A long section view of the North Complex showing the sublevels is shown in Figure 16-37.

Figure 16-35: Central Complex Long Section – Looking Northwest

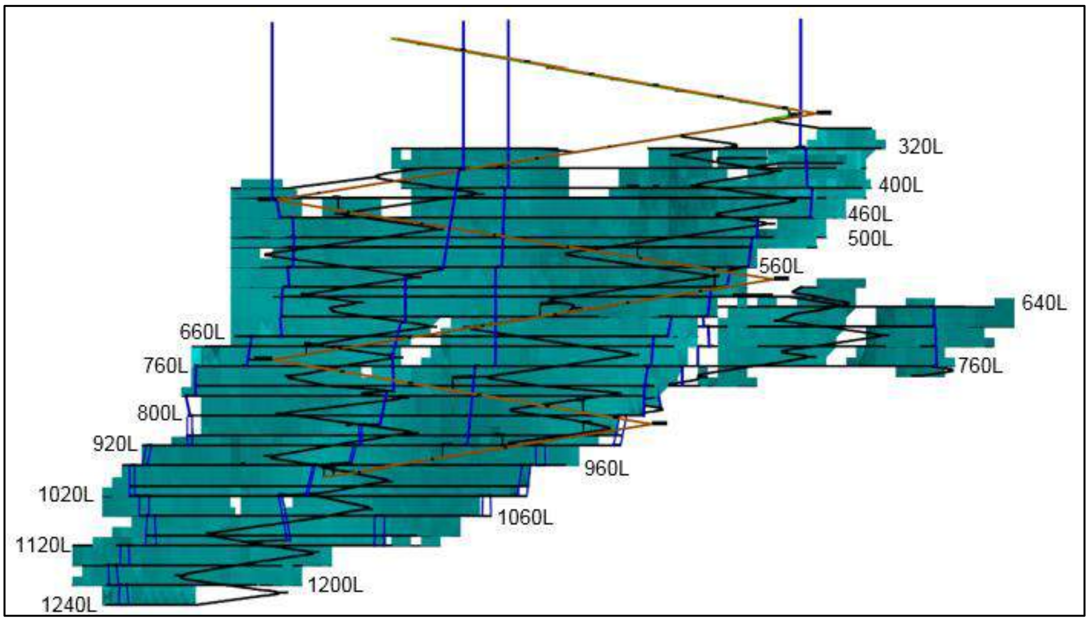


Figure 16-36: South Complex Long Section – Looking Northwest

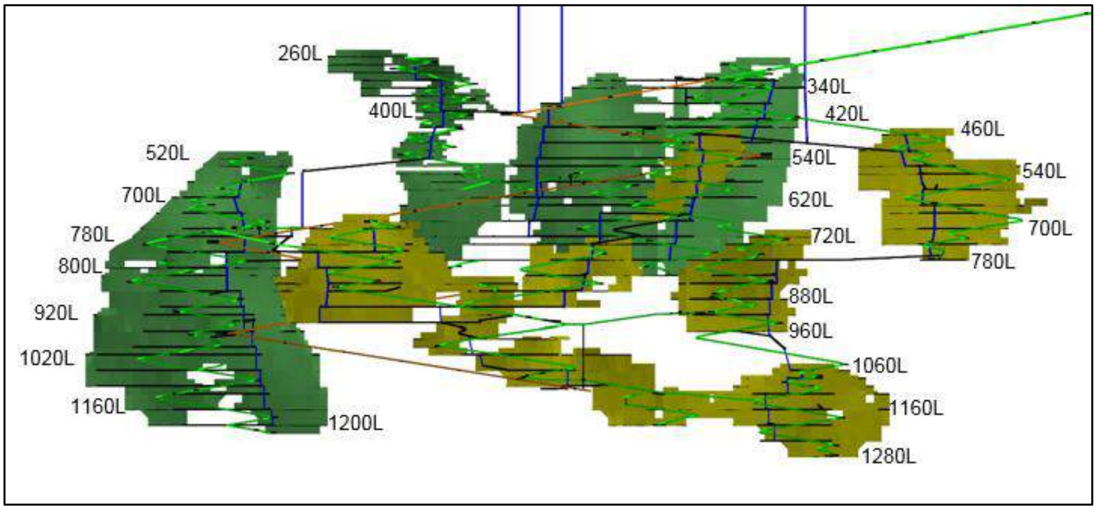
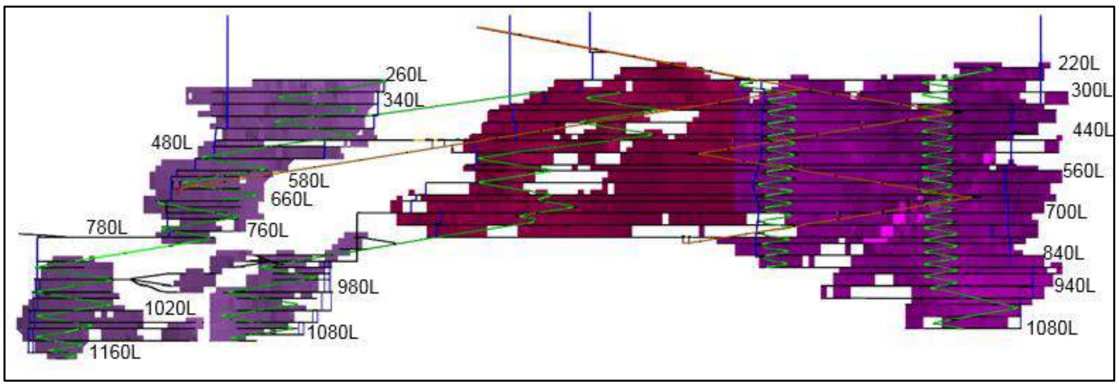


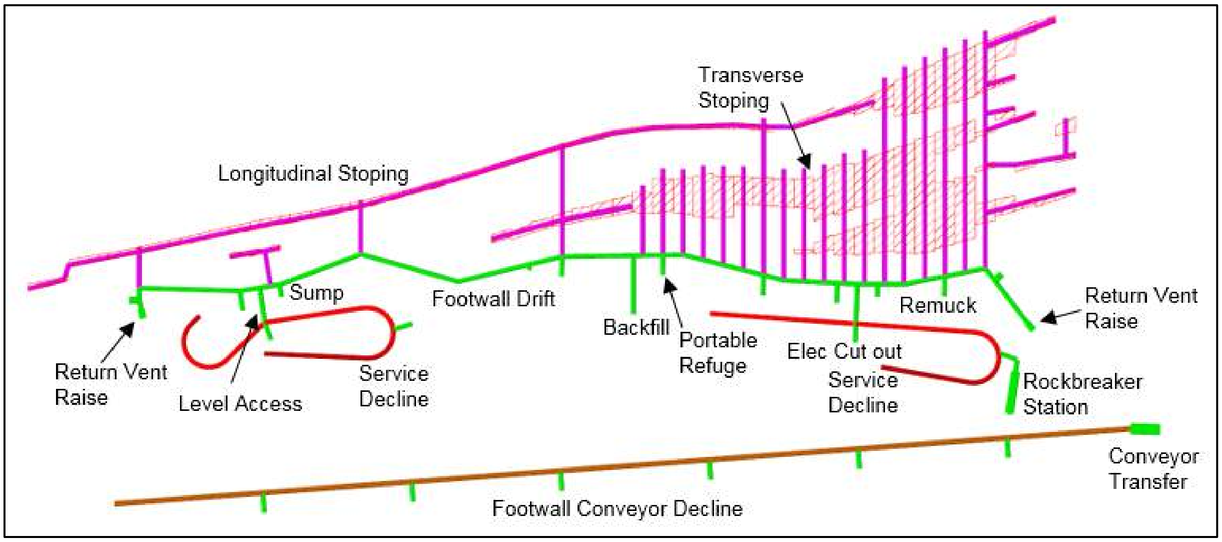
Figure 16-37: North Complex Long Section – Looking Northwest



16.3.8.1 Sublevel Development

Typical sublevel development is represented by 600 Level in the Central Complex shown in Figure 16-38.

Figure 16-38: Typical Sublevel Plan – Central Complex 600 Level



16.3.8.2 Development Quantities

The 3D mine model for each complex includes all decline, sublevel, and infrastructure development required to access and extract the Mineral Reserves. A summary of the development totals, by excavation type is included in Table 16-41.

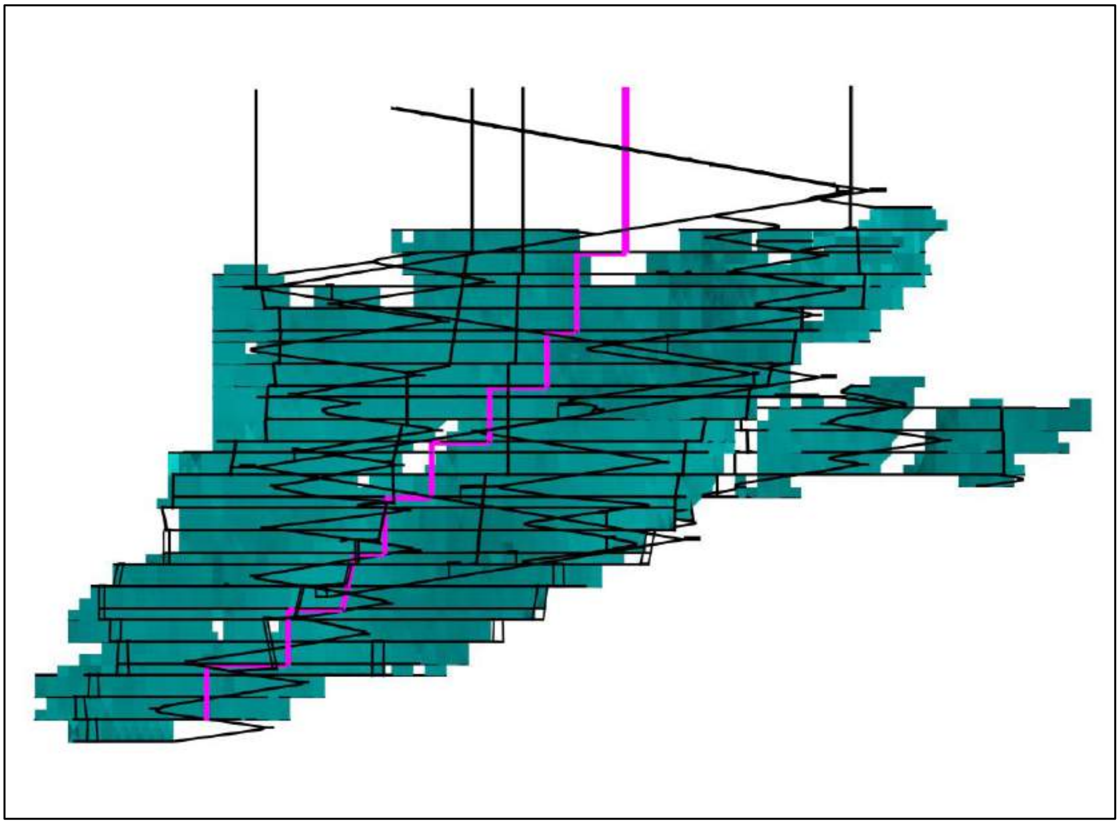
Table 16-41: Development Quantities by Excavation Type

Item	Central Complex (m)	South Complex (m)	North Complex (m)	Waterberg Total (m)
Main Decline Conveyor	1 764	1 417	1 352	**4 534**
Footwall Conveyor	4 183	5 354	5 696	**15 233**
Conveyor Transfer Station	157	157	94	**409**
Footwall Conveyor Access	564	810	1 874	**3 248**
Rock Breaker Station	483	178	517	**1 177**
Main Service Decline	1 766	1 408	1 148	**4 322**
Service Decline	14 603	29 017	25 202	**68 822**
Sublevel Access	1 910	4 139	5 617	**11 667**
Footwall Drift	27 570	17 167	51 533	**96 269**
Sump	552	1 311	1 339	**3 202**
Stope Access Cross Cut	76 378	16 969	83 893	**177 240**
Ore Longitudinal Sill 5wX4h	21 498	34 802	13 593	**69 893**
Ore Longitudinal Sill 6wX5h	10 708	12 035	32 962	**55 705**
Electrical Cut Out	962	1 805	1 889	**4 657**
Backfill Access	1 465	3 971	4 203	**9 639**
Diamond Drill Bay	4 601	2 476	8 598	**15 675**
Remuck Bay	5 206	6 208	8 427	**19 840**
Refuge Station/Waiting Place	122	229	247	**598**
Ventilation Access	6 566	7 911	8 281	**22 759**
Raisebore Room	1 144	1 549	1 271	**3 964**
Explosive Storage	18	36	39	**93**
Detonator Storage	95	132	220	**447**
Shop Large	110	112	117	**339**
Shop Small	566	428	651	**1 645**
Satellite Service Bay	89	94	122	**305**
Wash Bay	67	95	93	**254**
Fuel and Lube Bay	85	76	77	**238**
Satellite Fuel and Lube	46	76	93	**216**
Total	**183 279**	**149 963**	**259 148**	**592 390**

16.3.9 Mine Backfill – Underground

Mined stopes will be backfilled with paste backfill. Backfill will be delivered underground via 152 mm inside diameter ceramic lined pipe inserts installed in drill holes from surface. There will be three surface drill holes to service the Central Complex and two surface drill holes (one active and one spare) to service the South Complex. A network of internal underground drill holes and 152 mm pipe will deliver backfill to each sublevel and fill location. There will be backfill cut-out excavations on each level for the drill holes and piping inserts at the drill holes. The backbone of the paste backfill underground reticulation system for the Central Complex is shown in Figure 16-39.

Figure 16-39: Paste Backfill Underground Reticulation System Backbone – Central Complex Looking Northwest



A backfill barricade will be constructed at the stope drawpoint to contain the initial paste backfill plug. The barricade will be arch shaped, constructed from 350 mm thick 15.0 MPa concrete. Although paste backfill typically has little or no bleed water, the barricade design includes a drainage system to dissipate any pour pressure on the barricade and drain any free decant water to drain.

Further information on the surface paste backfill preparation plant is included in Section 18.

16.3.9.1 Backfill Strength Requirements

The backfill strength requirements for a stope will depend on if the backfill will be exposed due to mining adjacent stopes. Four cases of backfill exposure with varying strength requirements will be realised.

- Case 1 – Mining below a backfilled stope, exposing the backfill in the stope above.
- Case 2 – Mining beside a backfilled stope, exposing the backfilled end wall of the stope.
- Case 3 – Backfilling a secondary stope, that will not be mined beside or below.
- Case 4 – Mining a transverse stope from hanging wall to footwall in 'panels,' exposing the backfill wall along strike.

Case 1 Backfill

Within a mining block, stopes will be mined from bottom-up to directly beneath the backfilled stopes in the mining block above. The backfill in the stope above will be exposed and must have sufficient strength to remain intact. Prior to backfilling stopes that will be mined beneath, the stope floor must be properly mucked clean to ensure there will be no loose muck that will affect the fill quality. The stope cavity survey will be used to confirm the stope is mucked clean prior to backfilling.

The design parameters for Case 1 backfill strength is summarised in Table 16-42.

Table 16-42: Case 1 Backfill Design Parameters

Item	Value
Backfill Strength – Bottom Plug	2.0 MPa
Bottom Plug Thickness	Width: Height Ratio 1:1
Backfill Strength – Body of Stope	See Case 2, Case 3, or Case 4
Cure Time	28 days
Stope Width (along strike)	20 m

Case 2

Primary stopes will be mined and backfilled. When secondary stopes are mined adjacent to the primaries, the backfilled stope side wall will be exposed and must have sufficient strength to stand-up unconfined. The design parameters for Case 2 backfill strength is summarised in Table 16-43.

Table 16-43: Case 2 Backfill Design Parameters

Item	Value
Backfill Strength – Transverse (15 m to 60 m True Thickness)	0.35 to 0.76 MPa (Average 0.46 MPa)
Backfill Strength – Longitudinal (up to 15 m True Thickness)	0.35 MPa
Cure time	28 days
Stope Height	40 m Vertical, 60 m along dip

Case 3

Secondary stopes that will not be mined beneath or beside require only enough strength to be self-supporting and to provide a working base for an LHD or a longhole drill will when mining the next stope above. The design parameters for Case 3 backfill strength is summarised in Table 16-44. The secondary stopes will be capped with a layer of higher strength backfill.

Table 16-44: Case 3 Backfill Design Parameters

Item	Value
Backfill Strength	0.1 MPa
Cure Time	28 Days
Stope Height	40 m Vertical, 60 m along Dip

Case 4

Primary or secondary stopes in areas where the Mineral Resource is thick (from hanging wall to footwall) may have to be mined in panels to limit the backfill exposed in the back or wall. The design parameters for Case 4 Backfill strength is summarised in Table 16-45.

Table 16-45: Case 4 Backfill Design Criteria

Item	Value
Backfill Strength (20 m W)	0.46 MPa
Cure time	28 days
Stope Height	40 m Vertical, 60 m along Dip

16.3.9.2 Backfill System Requirements

The Central Complex and South Complex underground operations will be in production simultaneously and each will have independent backfill distribution infrastructure. The paste backfill plant / system will supply paste backfill to both complexes simultaneously. Future requirements will include distributing all paste backfill to the North Complex. The paste backfill pour rate allows for filling stopes 40% faster than the mine production rate to ensure capacity to catch-up if backfilling days are lost due to delays. The paste backfill pour rates for each complex are summarised in Table 16-46.

Table 16-46: Paste Backfill Pour Rates by Complex

Item	Central Complex	South Complex	North Complex
Paste Backfill Pour Rate	3 Lines 106 m^3/hr per Line	1 Line 106 m^3/hr per Line	4 Lines 106 m^3/hr per Line

Prior to paste backfill plant commissioning there will be approximately 135 000 tonnes of cemented rock fill used to fill the initial stopes in the Central Complex. Waste rock from development stockpiled on surface will be mixed with cement slurry on surface and backhauled in the 40-t capacity waste haul trucks.

During operations, as opposed to hauling to surface, some waste rock from development will be dumped into stopes that are in the filling cycle. The following factors were used the estimate the amount of waste rock disposed of in stopes

- No waste rock will be dumped into stopes during the first year of paste backfilling
- No waste rock will be dumped into the 20 m uppers stopes due to no access
- No waste rock will be dumped into the fill sill pillars
- Up to 30% of transverse secondary stope volume
- Up to 10% of transverse primary stope volume
- Up to 5% of longitudinal stope volume

The annual LOM backfill requirements for each complex are shown in Figure 16-40, Figure 16-41, and Figure 16-42.

Figure 16-40: Central Complex Backfill Requirements

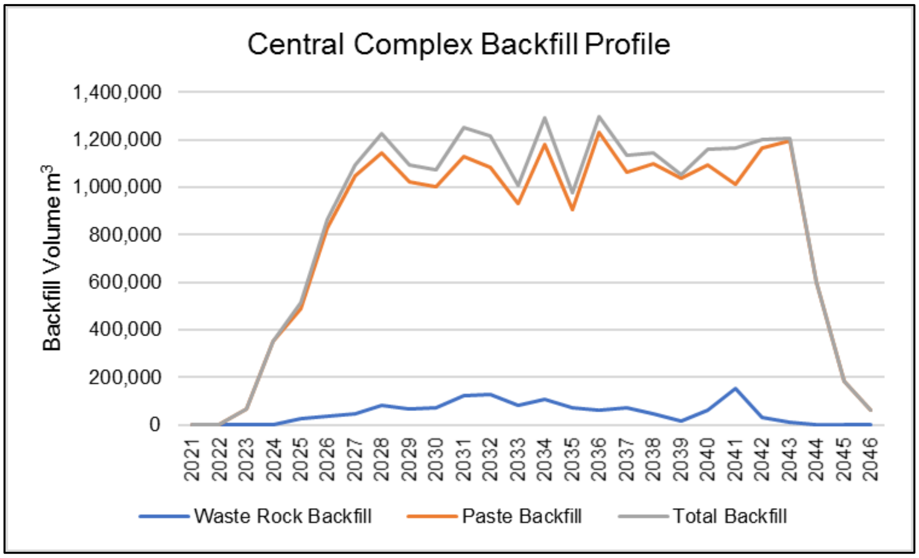


Figure 16-41: South Complex Backfill Requirements

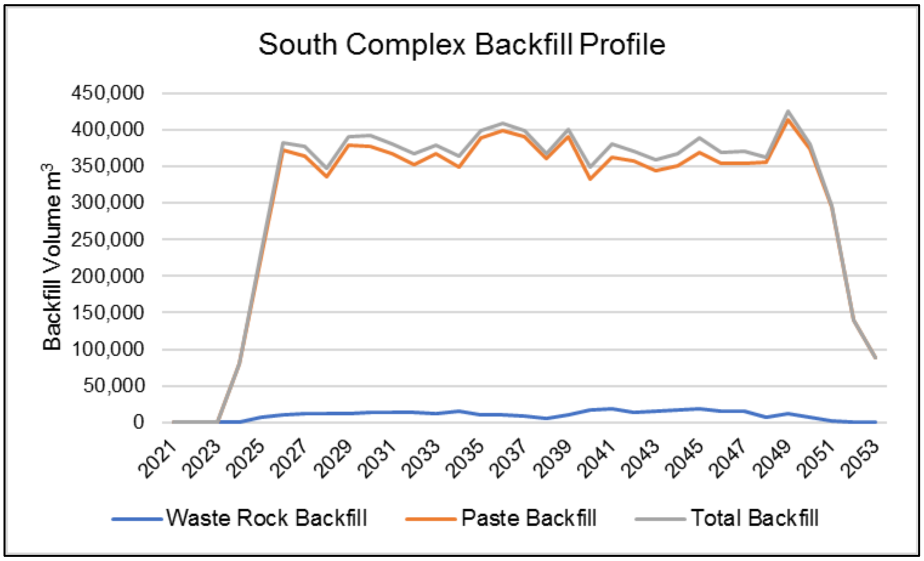
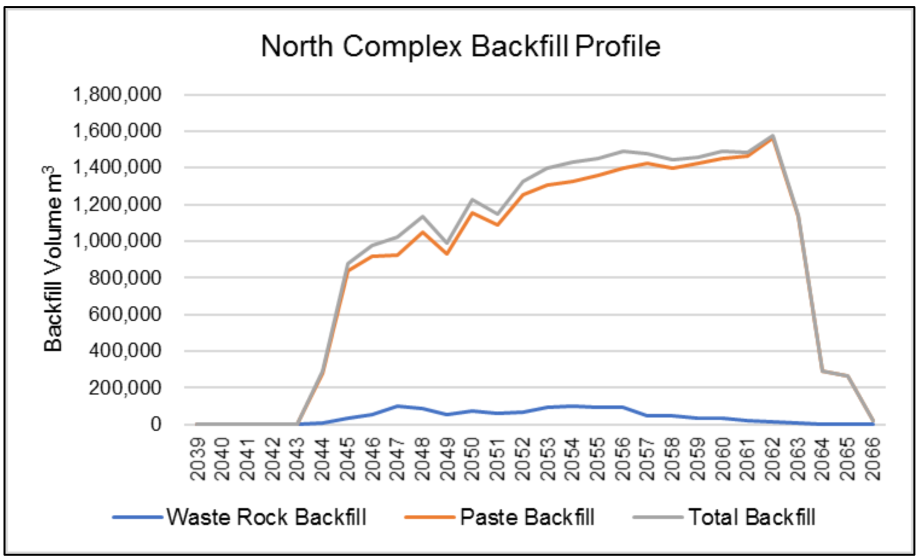


Figure 16-42: North Complex Backfill Requirements



16.3.10 Productivity Rates

The underground operations will operate two 10.5 hour shifts per day, seven days per week. The worker effective time per shift was estimated considering the amount of non-effective time or non-productive time during a shift. The estimated worker effective time per shift is summarised in Table 16-47.

Table 16-47: Estimated Worker Effective Time per Shift

Activity	Time	Unit
Morning Lineup in Lamp Room	5.0	Minutes
Vehicle Loading	5.0	Minutes
Travel Time to Working Area	20.0	Minutes
Shift Safety Meeting	15.0	Minutes
Travel Time to Working Face / Production Area	5.0	Minutes
Pre-use Inspection	15.0	Minutes
Legislated Breaks	30.0	Minutes
Re-fueling	20.0	Minutes
Wash and Grease at End of Shift	15.0	Minutes
Operator Unavailable and Other	20.0	Minutes
Travel Time from Working Face / Production Area to Surface Transportation	5.0	Minutes
Vehicle Loading	5.0	Minutes
Travel Time to Surface Lamp Room	20.0	Minutes
Total Non-effective Shift Time (minutes)	**180.0**	**Minutes**
Total Non-effective Shift Time (hours)	**3.0**	**Hours**
Total Shift Length (hours)	**10.5**	**Hours**
Total Effective Shift Length (hours)	**7.5**	**Hours**

16.3.10.1 Development Productivity

Lateral development advance rates were broken down into the components of the drill-blast-muck-bolt cycle and estimated from first principles. The rates reflect the advance that each jumbo and associated gear will achieve over extended periods of operation. These rates were benchmarked against other operations and experience of the project team members and review committee. The rates reflect long-term averages and include an efficiency allowance to account for interferences with other activities and conflicting priorities that occur during the operating period.

For the initial decline development in the poor ground conditions of the weathered Waterberg sediments, the advance rate for the jumbo (working at two faces) reflects drilling and blasting 3.0 m long rounds with shotcrete applied to the walls and back as secondary ground support. The resulting advance rate will average 3.2 m per day (combined for the two faces). Once the decline development reaches the sill rock unit, combined advance will be 6.2 m per day. This is approximately 186 m per month total advance (includes the decline face advance as well as remuck bays and the lateral connections between the two declines). During this initial decline development, the focus will be on development with minimal interference with other activities. There will also be opportunity for in-shift blasting during the initial decline development.

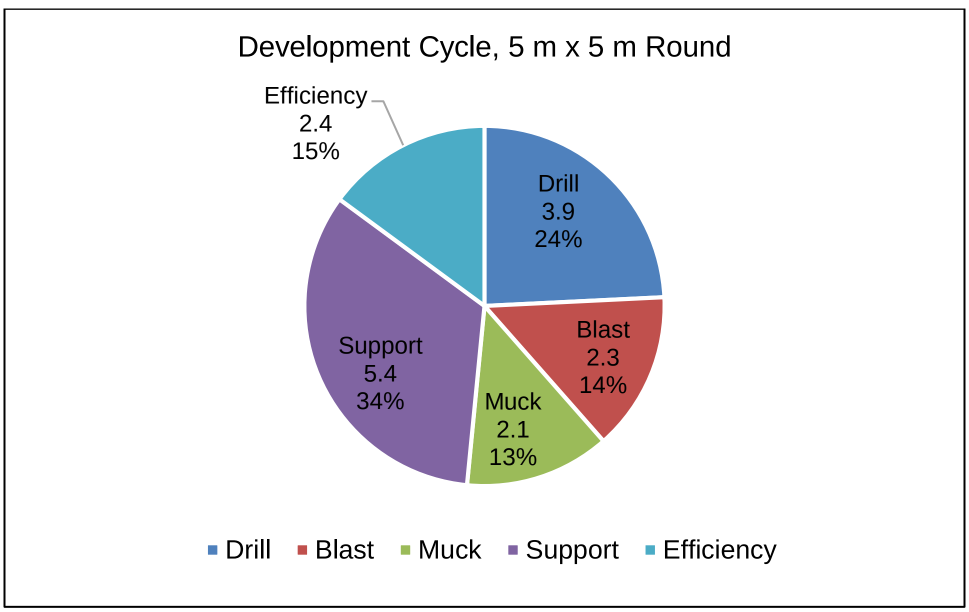
Once decline development reaches the Mineral Resource depth and ventilation infrastructure is established and workplaces become available, additional jumbos will be incrementally added. In general, each jumbo will have multiple workplace headings to advance. The estimated average long term daily advance rate per jumbo will be 6.2 m per day. To achieve this, each jumbo will average 1.63 development rounds per day.

The breakdown of the development cycle for a 5 m x 5 m waste rock heading in good quality ground is summarised in Table 16-48 and Figure 16-43.

Table 16-48: Development Cycle for 5 m x 5 m Round (Good-quality Ground)

Item	Hours
Drill	3.9 hrs
Blast	2.3 hrs
Muck	2.1 hrs
Ground Support	5.4 hrs
In-cycle Efficiency (85%)	2.4 hrs
Total Cycle	**16.1 hrs**
Single Heading	3.7 m/day
Two Headings	4.9 m/day
Multiple Headings	6.2 m/day

Figure 16-43: Development Cycle for 5 m x 5 m Round



The lateral development advance rates are summarised in Table 16-49.

Table 16-49: Lateral Development Advance Rates

Heading Type	System Advance		
	Single Heading (m/day)	Double Heading (m/day)	Multiple* Heading (m/day)
Service Decline 5.0 m W x 5.0 m H (sediments)	N/A	3.2	N/A
Conveyor Decline 5.5 m W x 5.0 m H (sediments)	N/A	3.2	N/A
5.5 m W x 5.0 m H (footwall waste)	3.5	4.6	5.8
5.0 m W x 5.0 m H (footwall waste)	3.7	4.9	**6.2**
6.0 m W x 5.0 m H (ore)	3.5	4.6	5.8
5.0 m W x 4.0 m H (ore)	4.3	5.6	7.2

Note:
- *Maximum advance in any face 75 m/month

Vertical development (i.e. raises) will be developed using raiseboring methods. The vertical development advance rates (excluding mobilisation and set-up times) are summarised in Table 16-50.

Table 16-50: Vertical Development Advance Rates

Raise Size	Pilot Hole (m/day)	Ream (m/day)
Surface 6 m diameter	16.0	4.0
UG 6 m Diameter	16.0	4.0
UG 4 m Diameter	16.0	5.0

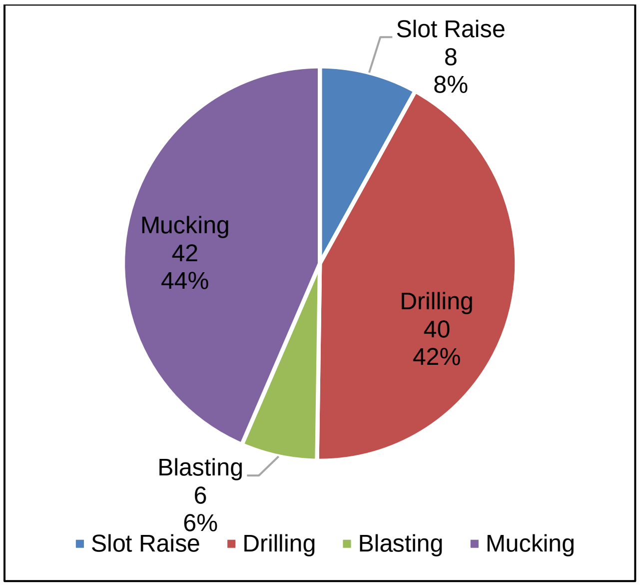
16.3.10.2 Stope Productivity

Stope production rates were broken down into the components of the drill-blast-muck (DBM) and backfill cycle and estimated from first principles. The DBM productivity was estimated accounting for parallel activities that can occur in-cycle and in parallel with other stopes. For example, although a stope cannot be blasted until the adjacent stope is backfilled, the slot raise and production drilling can be done in parallel with most other activities. A breakdown of the DBM cycle for a 21 m thick and 40 m H transverse stope is summarised in Table 16-51 and Figure 16-44.

Table 16-51: Drill-Blast-Muck Cycle for 21 m Thick, 40 m High Transverse Stope

Item	Days
Slot Raise	8 days
Production Drill	40 days
Blast	6 days
Muck	42 days
Total DBM Cycle	**96 days**
Total Mined Tonnes	67 000 t
Days with Parallel Activities	26 days
Tonnes per Day	954 tpd

Figure 16-44: Drill-Blast-Muck Cycle Days for 21 m Thick, 40 m High Transverse Stope



Stoping DBM productivities were broken into four groups according to stope thickness and the average of each of those groups were used as representative stope sizes. The representative stope productivities are summarised in Table 16-52.

The backfill component of the stope was created as a separate cycle and task in the production scheduling software. A breakdown of the backfill cycle for a 21 m thick and 40 m H transverse stope is summarised in Table 16-53 and Figure 16-45.

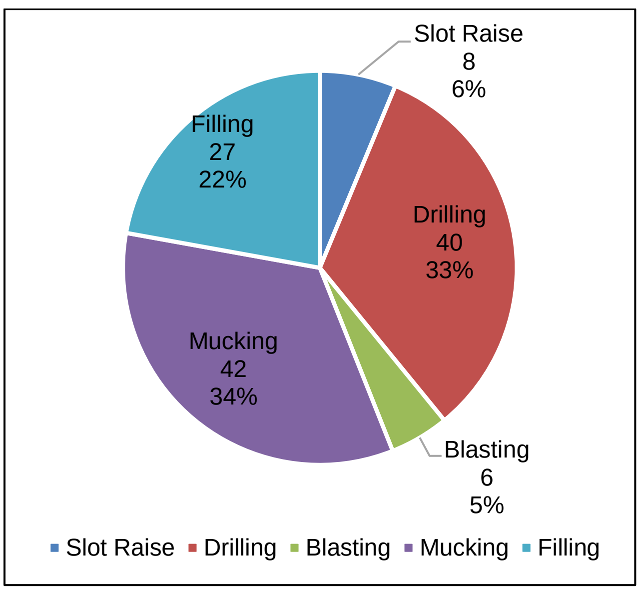
Table 16-52: Drill-Blast-Muck Cycle for Representative Stope Sizes

Type	Thickness Range (m)	Average Thickness (m)	Stope Height (m)	DBM (tpd)
Transverse	15-30	21	40	954
			20	747
	30+	48	40	1 015
			20	786
Longitudinal	4-15	8	40	789
			20	701
	2.4-4	3	40	523
			20	487

Table 16-53: Backfill Cycle for 21 m Thick, 40 m High Transverse Stope

Item	Days
Cavity Monitor Survey	1 day
Barricade Construction and Cure	5 days
Paste Backfill Plug Pour	4 days
Paste Backfill Plug Cure	3 days
Paste Backfill Body Pour	14 days
Total Backfill Cycle	**27 days**

Figure 16-45: Total Cycle Days for 21 m Thick, 40 m High Transverse Stope



The paste backfill cure time required for a stope before mining the next stope in sequence will vary depending on whether the next stope will be mined above (and only needs a backfill floor to work on), or adjacent (exposing a fill wall), or mining below (exposing backfill in the back). To account for vary cure time, the delay for backfill cure was accounted for using dependencies the Deswik production schedule.

16.3.11 Mine Development and Production Schedules

All mine development and production scheduling was completed using Deswik scheduling software (Deswik.Sched) with the schedule interactively linked to the Deswik 3D mine model. All development and production scheduling is based on dependencies linked within the mine model.

16.3.11.1 Development Scheduling

Mine development for each complex is broken down into three main phases of activity.

Phase 1 – Development of Main Declines
The first phase of development includes the twin decline development from surface until the first surface ventilation raise is commissioned and flow through ventilation is established. During this period, development will consist of the service and conveyor declines, remuck bays, and ventilation drifts connecting the two declines.

Phase 2 – Development after Flow-through Ventilation is Established
The second phase of development includes initial sublevel and infrastructure development including establishing the remaining surface ventilation raises. The priority during this phase is to commission the complete ventilation system so that ventilation can be increased, and additional development crews can be mobilised.

Phase 3 – Development after all Ventilation Raises are Commissioned
The final phase occurs after all ventilation raises are commissioned for steady state ventilation flow though. Additional development crews are then added to meet the production ramp-up period to full production.

The LOM development schedules for each complex are shown graphically in Figure 16-46 to Figure 16-48. The dip in the development profile in the South Complex in 2035-2036 is due to deferring accessing the F-South Zone.

Figure 16-46: Central Complex Development Profile

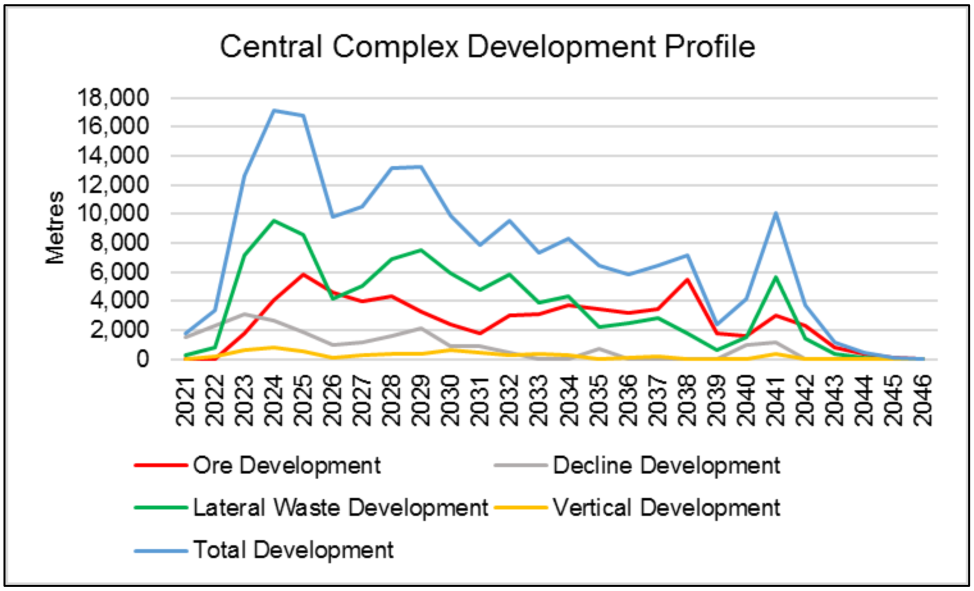


Figure 16-47: South Complex Development Profile

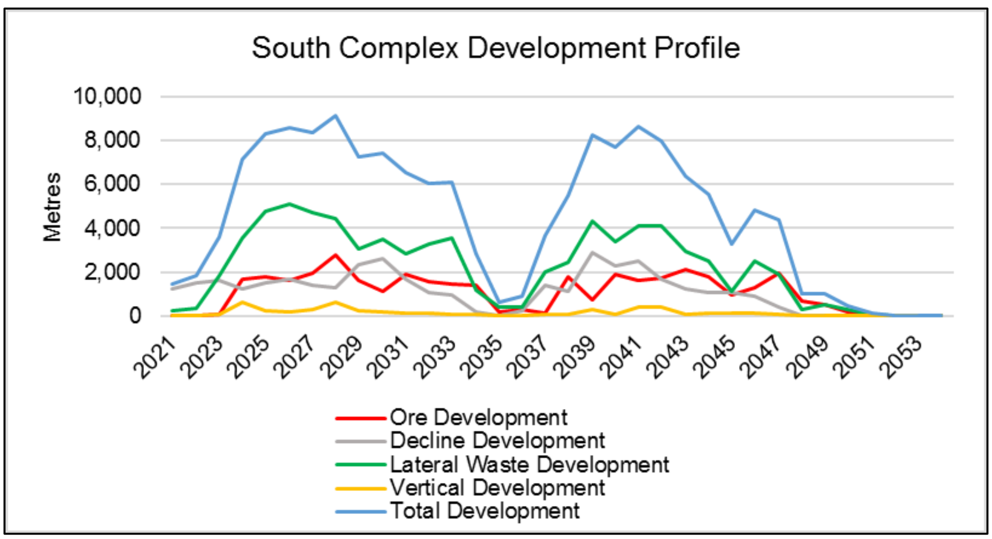
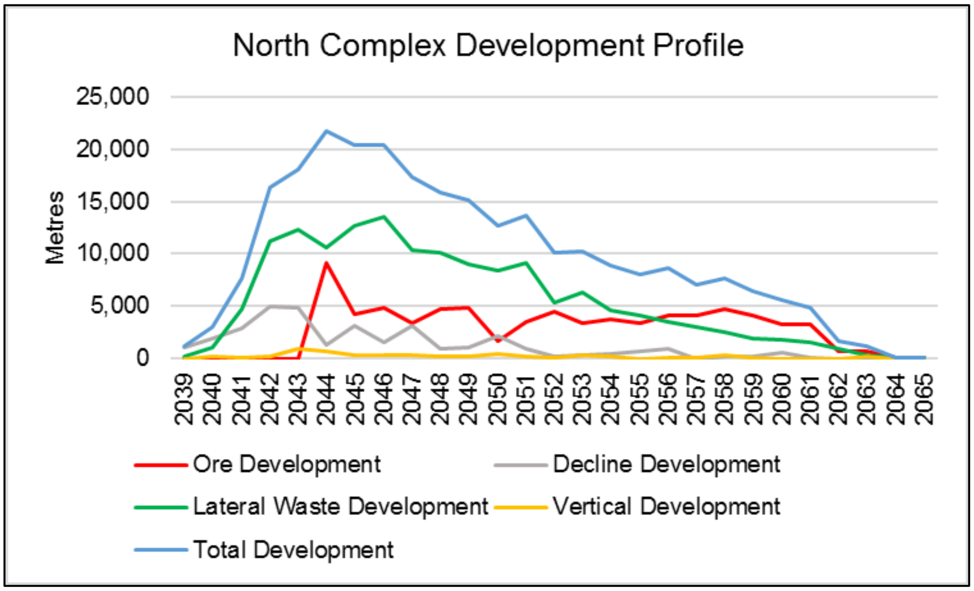


Figure 16-48: North Complex Development Profile



16.3.11.2 Production Scheduling

The production plan for the LOM focused on optimizing the ramp-up period and maximizing productivity. Each complex was scheduled independently as a stand-alone operation. The breakdown of tonnes and grade recovered by mining method and zone is summarised in Table 16-54.

Table 16-54: Life-of-Mine Production Summary

	T Zone	F-Central	F-South	F-North	F-Boundary North	F-Boundary South
Ore Tonnes – Stope Total	15 610 201	65 326 918	14 482 019	50 274 701	16 888 572	11 922 776
Ore Tonnes – Transverse	1 689 200	46 538 873	2 302 529	38 755 421	7 318 698	508 303
Ore Tonnes – Longitudinal	13 921 001	18 788 045	12 179 491	11 519 279	9 569 874	11 414 473
Ore Tonnes – Development	1 290 363	4 804 419	1 171 942	3 347 199	1 399 862	988 084
Ore Tonnes – Total	**16 900 564**	**70 131 337**	**15 653 961**	**53 621 900**	**18 288 434**	**12 910 859**
Grade 4E (g/t)	4.05	3.09	3.29	3.18	3.13	3.23
Grade Pt (g/t)	1.18	0.88	1.06	0.88	0.98	0.97
Grade Pd (g/t)	2.04	2.02	2.03	2.09	1.93	2.06
Grade Rh (g/t)	0.03	0.05	0.05	0.05	0.05	0.05
Grade Au (g/t)	0.80	0.14	0.15	0.16	0.17	0.15
Grade Cu (%)	0.18	0.07	0.04	0.10	0.10	0.07
Grade Ni (%)	0.09	0.18	0.13	0.20	0.23	0.19

Note:

- 4E = PGE (Pt + Pd +Rh) and Au. Totals may not add due to rounding.

The following criteria were applied during production ramp up and for LOM production scheduling.

- Proximity to Surface
- Measured Mineral Resource Classification
- Higher Grade
- High Productivity

Although targeting measured Mineral Resource material was prioritised during the production ramp-up period, this was not at the expense of sterilizing indicated Mineral Resource material or impeding the ability to optimise ramp up.

Initial production will come from the Central and South Complexes operating simultaneously, with the North Complex phased in once production in Central and South begins to ramp down. There will be approximately six years of ramp up from the start of the decline development to achieve steady-state production of approximately 400 000 tpm or approximately four years of ramp up from first ore until achieving steady state. The Central Complex steady-state production will average approximately 300 000 tpm (10 000 tpd), while the South Complex will average 100 000 tpm (3 333 tpd). Later in the mine life, the North Complex will ramp up to maintain 400 000 tpm production. The ramp-up and steady-state production tonnage profiles are shown in Figure 16-49 and Figure 16-50.

Figure 16-49: Production Tonnage by Month during Ramp Up

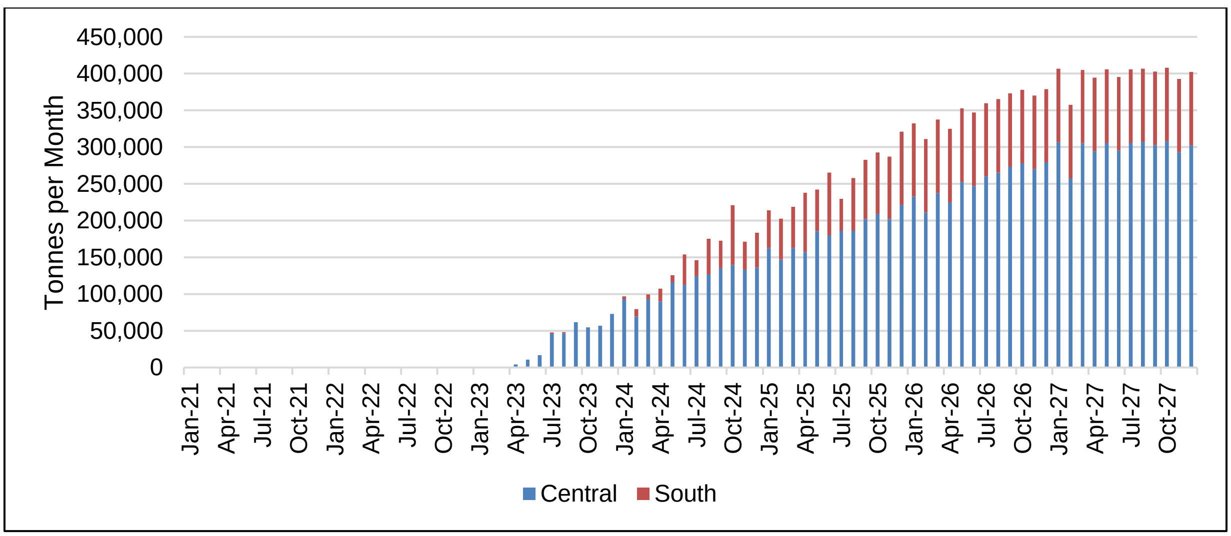
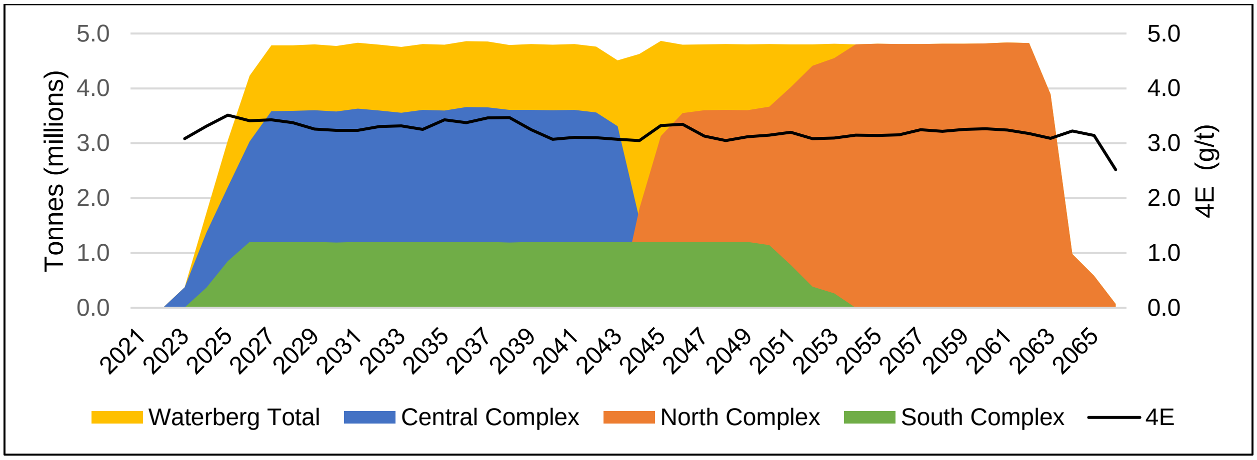


Figure 16-50: Annual Production Tonnage Profile



The production ramp-up period includes establishing capital development ahead of the mining front to allow delineation diamond drilling and mine planning ahead of production and also to provide access to sufficient developed Mineral Reserves for flexibility in the stope sequence. The developed Mineral Reserve increases annually and provides alternate sources of production if required. The developed reserve will continue to provide mitigation to maintain the production profile as the operation matures to steady state. If a problem occurs in a stope there will be flexibility to move to another stope in the active area or on another active level.

Production Sequencing

Each complex was divided into 100 m vertical mining blocks (consisting of two 40 m H stopes and one 20 m H stope) and the stopes within each mining block were sequenced depending on the stoping method (transverse or longitudinal).

The transverse stopes were mined in a primary-secondary sequence according to the rules outlined below and demonstrated in Figure 16-51.

a. Cannot start drilling a primary stope above until the stope below is filled and cured and sill rehabilitation is complete.
b. Cannot start drilling a bottom secondary until both adjacent middle primaries are filled.
c. Cannot start drilling middle secondary until both adjacent top primaries are filled.
d. Cannot start drilling any top stopes until the bottom stope from the block above has 28 days of paste backfill curing.
e. In some cases, there will not be an adjacent primary above, if so, cannot start drilling the adjacent stope until the previous stope has 21 days of paste backfill curing.

The longitudinal stopes are accessed approximately every 200 m along strike and are retreated back to a central access according to rules outlined below and demonstrated in Figure 16-52.

- Cannot start drilling stope above until the stope is filled and sill rehabilitation complete.

- Cannot start drilling adjacent bottom stope until previous middle stope is filled.
- Cannot start drilling adjacent middle stope until previous top stope is filled.
- Cannot start drilling any top stopes until the bottom stope from the block above has 28 days of paste backfill curing.
- In some cases, there is no previous longitudinal stope above. If so, cannot start drilling the adjacent stope until the previous stope has 21 days of paste backfill curing.

Figure 16-51: Transverse Stope Sequencing Rules – Longitudinal View

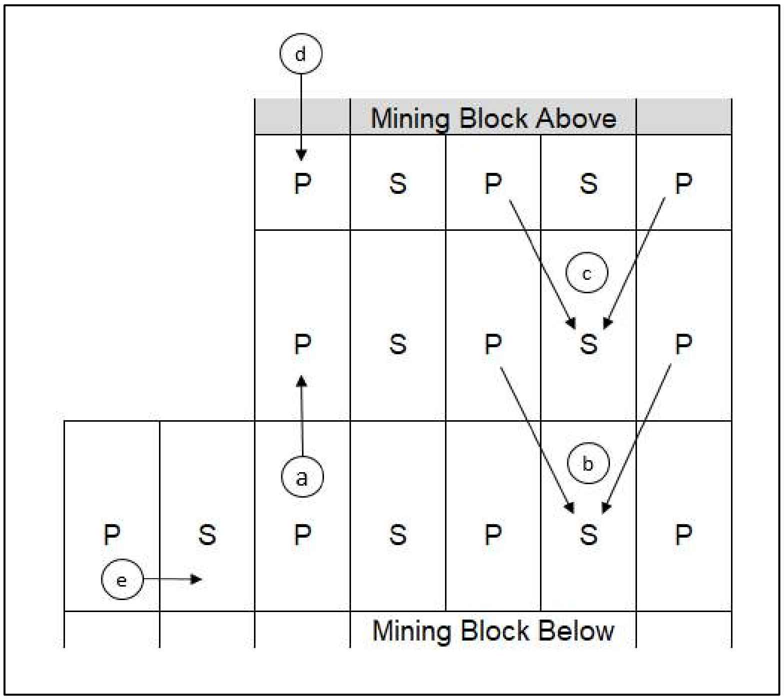
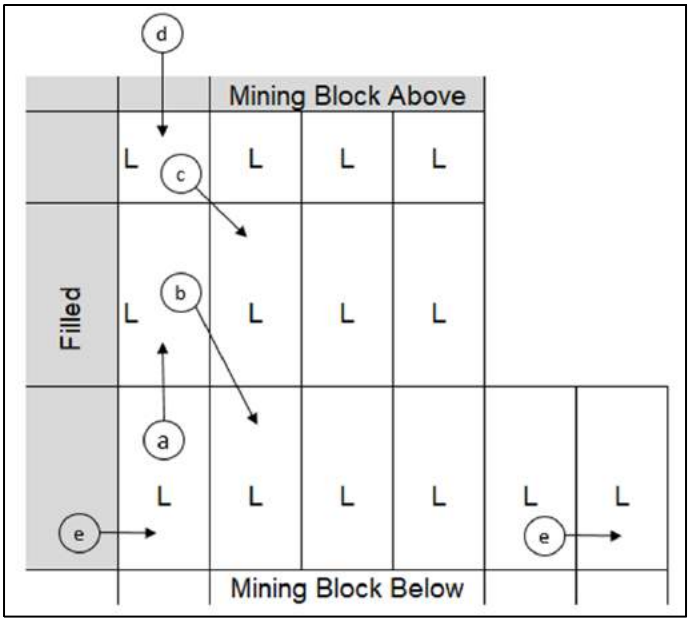


Figure 16-52: Longitudinal Stope Sequencing Rules – Longitudinal View

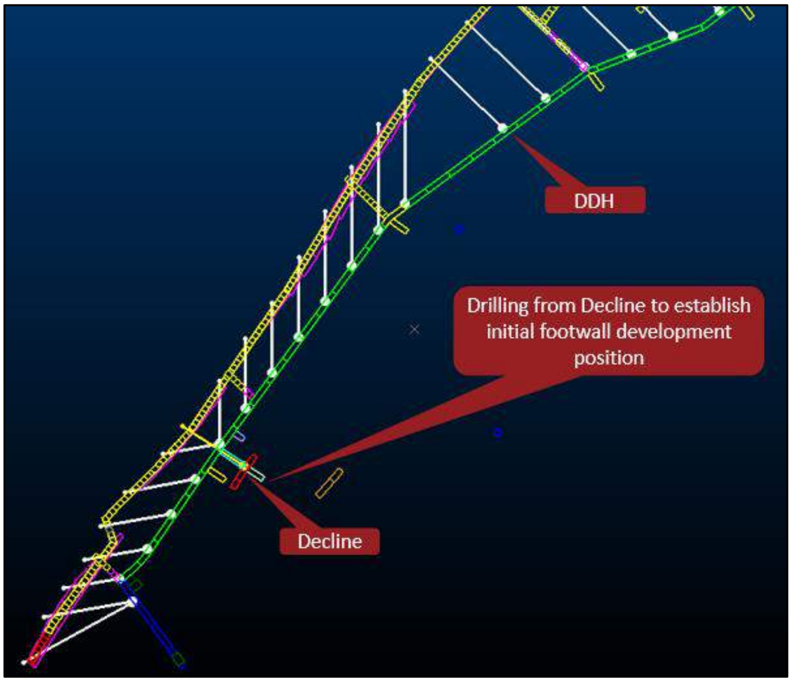


16.3.12 Delineation Diamond Drilling

Mineral Resource definition drilling will be completed from both surface and underground. The main objective of the Mineral Resource definition drilling is to upgrade indicated Mineral Resources to measured Mineral Resources. Such infill surface Mineral Resource definition will be undertaken in initial years until the mine is established to allow access for underground Mineral Resource definition drilling well in advance of stoping. Capital provision is made for infill Mineral Resource definition drilling to depths of approximately 700 m below surface.

In each complex there will be underground diamond drilling programmes to upgrade the Mineral Resource and continuously delineate all stopes for mine planning and grade control. The delineation diamond drilling will be completed from drill cut-outs spaced along the footwall drifts on sublevels and from other pre-developed excavations, including remuck bays in the declines. Sufficient mine development will be scheduled and in place ahead of the advancing production fronts to ensure adequate time for definition diamond drilling and subsequent Mineral Resource model updates and mine planning. Diamond drilling will be completed from the service decline and footwall drift to define the placement of sublevel infrastructure and stope sills. This drilling is demonstrated on 460 Level in the Central Complex in Figure 16-53.

Figure 16-53: Delineation Diamond Drilling – Central Complex 460 Level (Plan View)



A typical diamond drilling section showing multiple sublevels in a longitudinal mining area is shown in Figure 16-54.

In thicker transverse mining areas, stope delineation and grade control drilling can be completed from the stope crosscuts as shown in Figure 16-55.

Figure 16-54: Typical Diamond Drilling Section View – Longitudinal Mining Area

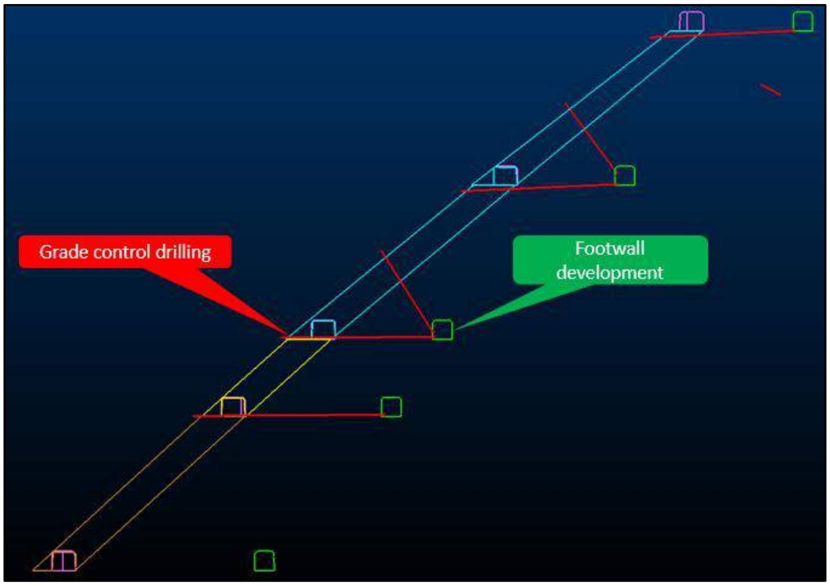
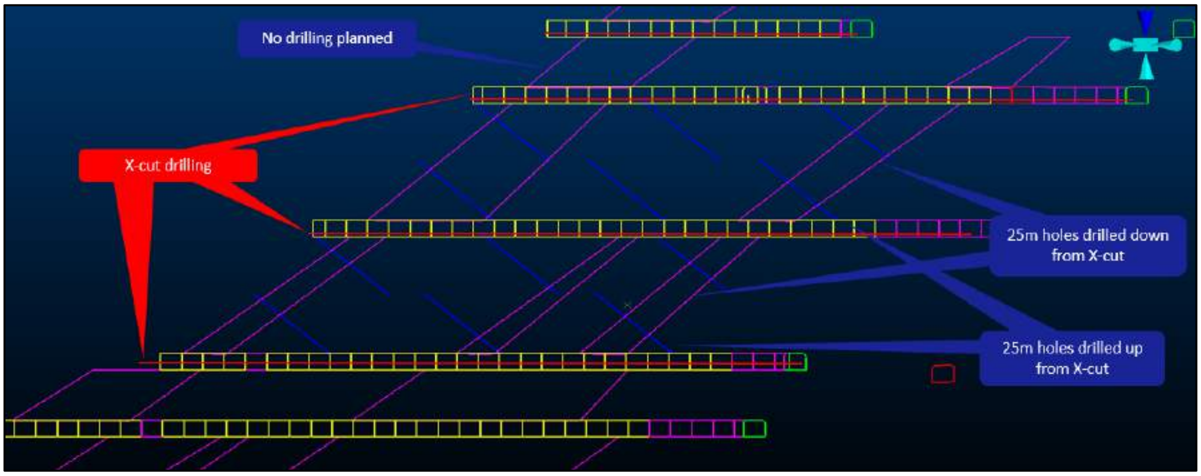


Figure 16-55: Delineation Drilling from Stope Crosscuts



It is proposed that over the LOM, some 478 km of delineation drilling will be completed (an average of almost 1 000 m per month) or some 390 tonnes of ore per metre drilled.

16.4 Mine Ventilation and Refrigeration Design

Ventilation to each complex will be provided by surface fresh air and return air ventilation raises and the declines. The ventilation systems will be a “pull” system with large surface fans located at the exhaust raises. The ventilation system will be designed to provide flow-through ventilation with fresh air pulled from the service declines and fresh air raises located near the centre of each sublevel and return air exhausting to surface via return air raises located at the extremities of sublevels. The ventilation in the conveyor declines will have fresh air pulled from the portals and exhausted without being used to ventilate other mine workings. Doors at each of the sublevel connections to the conveyor decline will prevent mixing of the conveyor ventilation air with the rest of the mine workings. The underground mobile equipment fleet will be diesel powered and mine air cooling will be implemented to maintain underground working temperatures within designed thresholds listed in Section 16.4.8.

The main ventilation fans are located at the exhaust raises on surface to reduce heat gain in the fresh air supply and to provide better control of the airflow through minimizing leakage. The fresh air intake raises, where the bulk-air coolers (BAC) for cooling will be located, will have stench gas release on the BAC intake fans for warning in the event of an emergency.

16.4.1 Ventilation and Refrigeration Assumptions and Design Criteria

Assumptions and design criteria for the ventilation system are provided in Table 16-55.

All the main fans will be equipped with variable frequency drives (VFDs), to provide the capability to modulate the airflow being exhausted from each raise. South African regulations for mine ventilation and industry best practices were considered in assessing the ventilation requirements.

Underground internal ventilation raises will be 6.0 m diameter down to 800 Level; however, below 800 Level twinned 4.0 m diameter raises will be used for geotechnical stability.

Table 16-55: Ventilation and Cooling Design Criteria

	Item	Design Value
Sizes	Service Decline and Access Drifts Size	5 m H x 5 m W
	Conveyor Decline Size	5 m H x 5.5 m W
	Drift Profile	Arched
	Duct Material	Fabric (<500 m)
		PVC (>1 000 m)
	Duct Size	1 220 mm
	Ventilation Raises	4 m or 6 m Diameter (Ø)
	Fan Station width	2 x Fan Diameter
	Fan Station Length before Fans	5 x Fan Diameter
	Fan Station Length after Fans	5 x Fan Diameter
Surface Conditions	Surface Summer Design Wet-bulb Temperature	20.0°C
	Surface Summer Design Dry-Bulb Temperature	30.0°C
	Surface Rock Temperature	24.2°C
	Barometric Pressure	88 kPa
Heat / Airflow Requirements	Geothermal Gradient	1.8°C per 100 m
	Wetness Fraction	0.15
	Maximum Wet-bulb Globe Temperature (WBGT) (airways with personnel)	29.0°C
	Maximum WBGT (only cabbed equipment)	33.5°C
	Engine Efficiency	37%
	Engine Load	60%
	Airflow Requirement	0.08 $m^3/s/kW$
Velocity Thresholds	Main Airways	6.5 m/s
	Airways without Personnel	10 m/s
	Conveyor Declines	5 m/s
	Intake / Exhaust Raises	20 m/s
	Workshops	0.4 m/s
Friction Factors	Raisebored Airways	0.005 kg/m^3
	Average Blasted Main Airways	0.012 kg/m^3
	Fabric Ducting	0.003 kg/m^3
	PVC Ducting	0.002 kg/m^3

16.4.2 Airflow Requirements

Airflow requirements for the different underground mining crews / functions are detailed in Table 16-56. Airflow requirements are for the peak production and development periods to highlight the maximum airflow requirements. The airflow required takes into consideration the mobile equipment utilization factor and is rated at 0.08 m^3/s per engine kW rated, with utilization factors applied. The equipment shows the requirement for development, production, haulage, and miscellaneous auxiliary equipment. Leakage for North and Central Complexes was calculated at 10%, while South Complex was allocated 25% leakage due to the fresh air being distributed onto the service declines and having more transfer drifts than the other complexes. The required total flow is approximately 1 124 m^3/s, 688 m^3/s, and 1 229 m^3/s at full production for the Central, South, and North Complexes, respectively.

Table 16-56: Airflow Requirements (North, Central and South Complexes)

			North Complex		Central Complex		South Complex	
Item	Engine Power kW/unit	Utilization %	Total Units (each)	Total Vent (m^3/s)	Total Units (each)	Total Vent (m^3/s)	Total Units (each)	Total Vent (m^3/s)
Development Crew								
2-Boom Jumbo	55	15	11	7	7	5	4	3
LHD – 17T	285	60	11	150	7	96	4	55
Mechanical Bolter	58	15	12	8	9	6	5	3
Explosives Loader	55	40	5	9	4	7	2	4
Production Crew								
Slot Drill - ITH	120	5	1	1	1	1	1	1
Production Drill	120	5	6	3	7	3	3	1
LHD - 17T	285	90	6	123	7	144	3	62
Explosives Loader	130	40	3	13	2	8	2	8
Blockholer	120	5	1	1	1	1	1	1
Haulage Fleet								
LHD – 17T	285	90	3	62	4	82	2	41
50T Truck (Production)	515	90	7	260	9	334	3	111
40T Truck (Development)	388	90	7	196	5	140	4	112
Construction and Services								
Shotcrete Sprayer	92	20	4	6	2	3	1	1
Concrete Transmixer	129	30	4	12	1	3	1	3
Scissor Lift	78	50	10	31	7	22	5	16
Cassette Truck	103	40	3	10	2	7	2	7
Boom Truck - Material	103	80	3	20	3	20	1	7
Boom Truck - Construction	103	10	2	2	1	1	1	1
Service LHD	310	50	4	50	3	37	3	37
Water Tanker	129	30	1	3	1	3	1	3
Telehandler	75	20	3	4	2	2	1	1

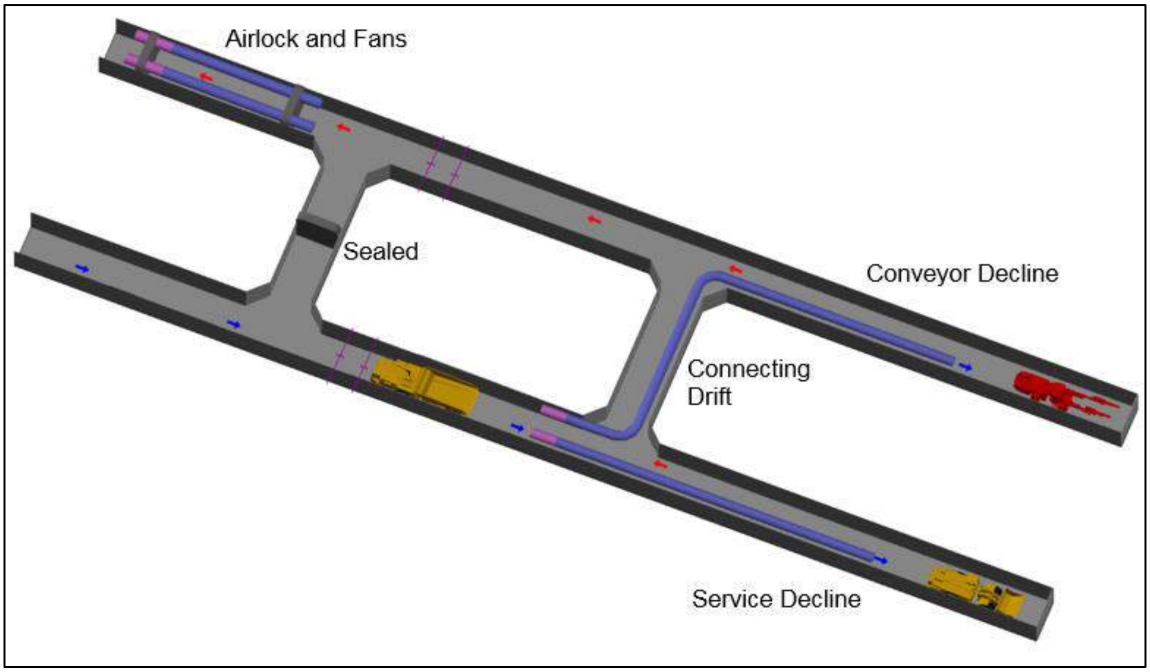
Item	Engine Power kW/unit	Utilization %	North Complex Total Units (each)	North Complex Total Vent (m^3/s)	Central Complex Total Units (each)	Central Complex Total Vent (m^3/s)	South Complex Total Units (each)	South Complex Total Vent (m^3/s)
Grader	109	20	1	2	1	2	1	2
Forklift	109	10	2	2	1	1	1	1
Cable Bolter	110	10	1	1	1	1	1	1
Maintenance								
Mechanic Truck	115	25	4	9	2	5	1	2
Millwright Service Truck	115	25	3	7	2	5	1	2
Conveyor Service Truck	115	25	2	5	2	5	2	5
Electrician Tractor	115	25	3	7	2	5	1	2
Fuel / Lube Truck	115	50	3	14	2	9	1	5
Telehandler	75	15	2	2	1	1	1	1
Personnel Carriers								
30 Person	106	20	4	7	3	5	2	3
Small Services	75	40	5	12	3	7	2	5
Tractors	115	40	6	22	5	18	4	15
Pick-Ups	115	30	22	61	13	36	11	30
Subtotal Mobile Equipment								
Development Crews				175		114		64
Production Crews				139		156		72
Haulage				517		555		264
Miscellaneous Equipment				286		196		149
Leakage			**10%**	**112**	**10%**	**102**	**25%**	**138**
Total Vent Requirements				**1 229**		**1 124**		**688**

16.4.3 System Description

16.4.3.1 Decline Development

The main service and conveyor declines from surface will be developed simultaneously with fresh air through the service decline and exhaust air through the conveyor decline. To establish the flow-through ventilation system between the two declines, an airlock will be installed near the entrance of the conveyor decline with 2 x 230 kW fans mounted across the bulkheads creating the negative pressure required to promote the ventilation flow. To create the ventilation loop, all the connecting drifts between the service decline to the conveyor decline (apart from the last one that was created closest to the advancing face) will be sealed as illustrated in Figure 16-56. For ventilation, the heading auxiliary fans will be mounted just before the last connecting drift in the service decline with ducting going to each heading. The fans are rated at 56 kW each, pushing 23 m^3/s to the face, sufficient for the operation of an LHD.

Figure 16-56: Decline Development – Ventilation Schematic – Isometric View



16.4.3.2 Heading Development

For development headings up to 500 m, fabric ducting will be used to provide the auxiliary ventilation required, while for longer lengths, rigid ducting will be required to minimise frictional pressure loss and allow additional fans to be installed in series. For headings with a truck and an LHD, twin ducting will be required to provide the appropriate airflow.

In the case of shorter headings (<500 m with fabric ducting), the ventilation will be supported by a 112-kW auxiliary fan at each duct. For the longer headings with rigid ducting, ventilation will be supported up to 1 000 m with a 112-kW fan at each duct after which another fan in series will be required.

16.4.3.3 Central Complex

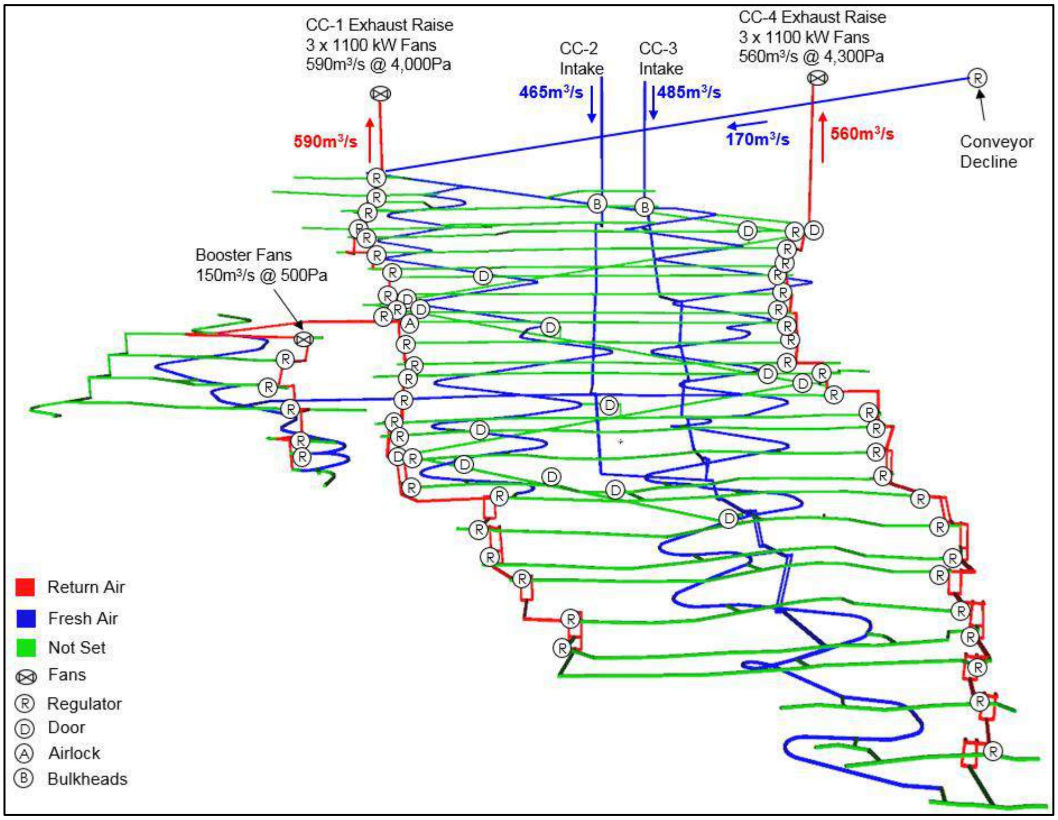
The ventilation system for the Central Complex will be comprised of four 6.0 m diameter raisebored surface raises, two for exhaust and two for intake.

The ventilation system will be established in four main stages. During each stage the backbone of the ventilation system will continue to expand through the addition of internal ventilation raises that will connect between sublevels. In addition, the increased ventilation will support increased development, construction, and production activities.

- Stage 1 – Main decline development and establish CC-1 exhaust raise (150 m^3/s).
- Stage 2 – Establish initial sublevels and CC-2 and CC-3 fresh-air intake raises (510 m^3/s).
- Stage 3 – Establish CC-4 exhaust raise (1 140 m^3/s).
- Stage 4 – Full complex developed (1 120 m^3/s).

The final ventilation system for the Central Complex (Stage 4) is shown in Figure 16-57.

Figure 16-57: Central Complex – Stage 4 - Longitudinal Looking Southeast



16.4.3.4 South Complex

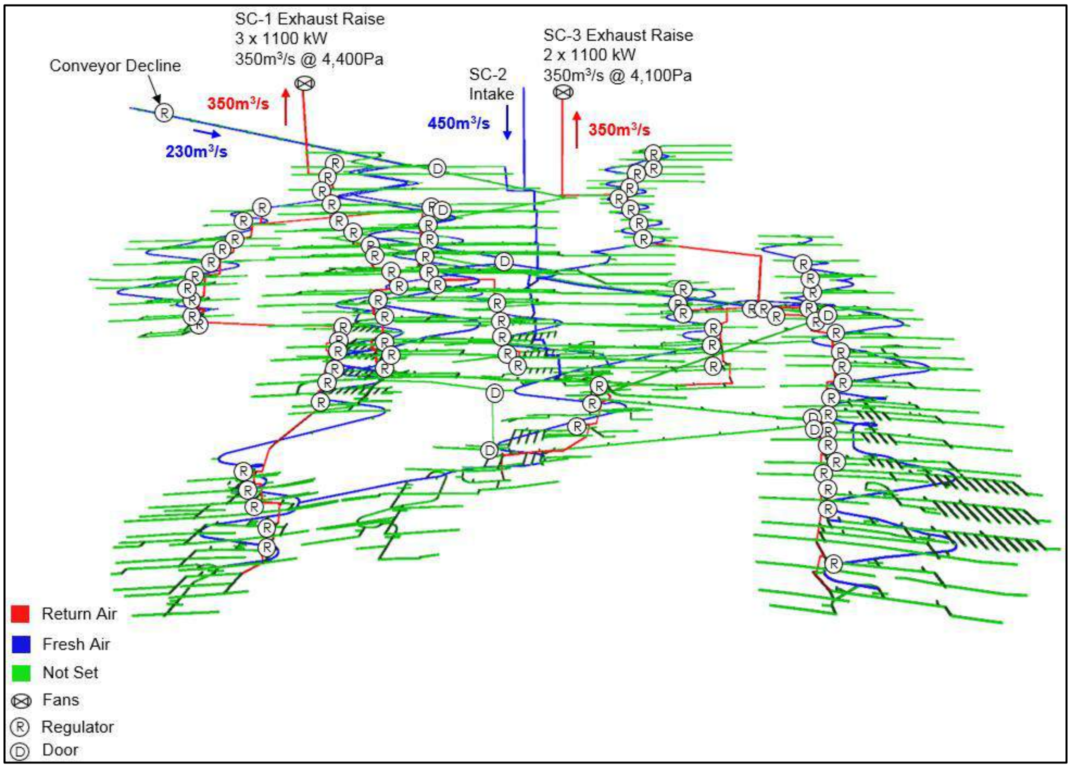
The ventilation system for the South Complex will be comprised of three 6.0 m diameter raisebored surface raises, two for exhaust and one for intake. Since most of the levels will have only access by a single ramp, all the internal raises will be equipped with escapeways for secondary egress. On levels with two internal raises, one for fresh air and one for exhaust, only one of the internal raises will be equipped with the escapeway.

The ventilation system will be established in four main stages. During each stage the backbone of the ventilation system will continue to expand through the addition of internal ventilation raises that will connect between sublevels. the increased ventilation will support increased development, construction, and production activities.

- Stage 1 – Main decline development and initial sublevel development (130 m^3/s).
- Stage 2 – Establish SC-1 exhaust raise and SC-2 intake raise (620 m^3/s).
- Stage 3 – Establish SC-3 exhaust raise (670 m^3/s).
- Stage 4 – Full complex developed (680 m^3/s).

The final ventilation for the South Complex is shown in Figure 16-58.

Figure 16-58: South Complex – Stage 4 - Longitudinal Looking Southeast



16.4.3.5 North Complex

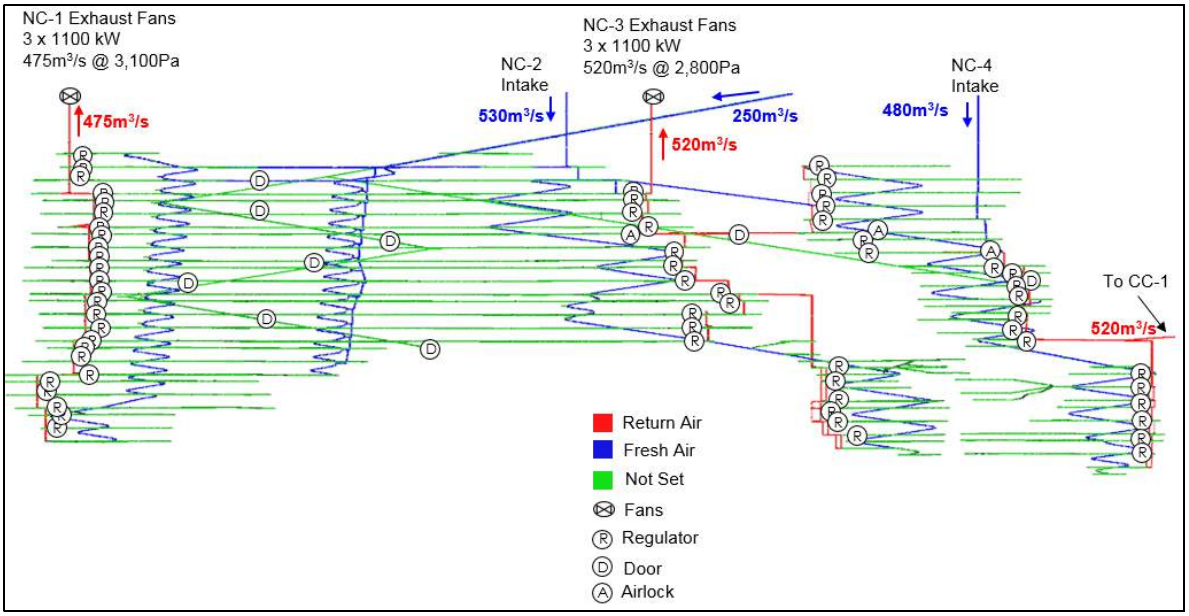
The ventilation system for the North Complex will be comprised of four surface raises, two for exhaust and two for intake.

The ventilation system will be established in five main stages. During each stage, the backbone of the ventilation system will continue to expand through the addition of internal ventilation raises that will connect between sublevels. The increased ventilation will support increased development, construction, and production activities.

- Stage 1 – Main decline development and temporary use of NC-2 as exhaust (300 m^3/s).
- Stage 2 – Establish NC-1 and NC-3 exhaust raises and convert NC-2 to intake (780 m^3/s).
- Stage 3 – Establish NC-4 Intake raise (1 050 m^3/s).
- Stage 4 – Expansion of Stage 3 (1 160 m^3/s).
- Stage 5 – Full complex developed. Connect to Central Complex CC-1 (1 260 m^3/s).

The final ventilation for the North Complex is shown in Figure 16-59.

Figure 16-59: North Complex – Stage 5 - Longitudinal Looking Southeast



16.4.4 Main Surface Fans

The main surface fan pressure requirements were estimated from the VentSIM ventilation models based on the required airflows. From these parameters, the fan motor ratings were assessed. The main fan sizes and ratings were standardised where possible across all installations for ease of maintenance and to reduce spare requirements on site. The main surface fan requirements are summarised in Table 16-57.

Table 16-57: Main Surface Fan Requirements

Type	Number of Fans	VFD Capable	Peak Airflow per Fan (m^3/s)	Peak Pressure per Fan (Pa)	Motor Rated Power (kW)
North Complex					
NC-1 Main Exhaust Fans	3	Y	200	3 100	1 100
NC-3 Main Exhaust Fans	3	Y	200	2 700	1 100
Central Complex					
CC-1 Main Exhaust Fans	3	Y	200	4 000	1 100
CC-4 Main Exhaust Fans	3	Y	190	4 200	1 100
South Complex					
SC-1 Main Exhaust Fans	3	Y	120	4 400	1 100
SC-3 Main Exhaust Fans	2	Y	175	4 400	1 100

All main fans will be located on surface and mounted in a horizontal arrangement. All fans will have a trifurcated fan arrangement, except for SC-3 which will be bifurcated. All fans will be equipped with VFDs to provide variable airflow underground depending on the airflow requirements at that stage of mining.

16.4.5 Auxiliary Fans

The auxiliary fan quantities were derived from the production and development schedule for each mine complex (Table 16-58). The auxiliary fans for development and production are rated at 112 kW and 56 kW, respectively. The rating selection considers available headroom in the drift for fan installation and distance the fan will be pushing the air through the duct. A single development fan will be able to support an LHD to 500 m using fabric ducting (or 1 000 m with rigid ducting). For headings with an LHD and truck operating, twin ducting will be required with the same ventilation length limits – after which another fan in series will be required. A single production fan will be able to support ventilation for an LHD to about 250 m.

Table 16-58: Auxiliary Fan Requirements

Type	North Complex	Central Complex	South Complex
Development Fans (112 kW)	24	14	10
Production Fans (56 kW)	15	12	8
Decline Airlock Fans (230 kW)	2	2	2
Booster Fans (230 kW)	1	2	-

16.4.6 Ventilation Controls

Ventilation controls will be used to control airflow throughout the mine and optimise ventilation system performance. These controls will include airlocks, drop-board regulators, and overhead doors. The overhead doors will be used primarily to isolate airflow in the conveyor drift from the rest of the mine workings. These doors will also prevent contamination of the air in the event of a fire in the conveyor drift.

The main control system for the mine from level to level will be provided by the drop board regulators at either the fresh or return raise access. The regulator opening will be adjusted according to required airflow.

16.4.7 Heat Loads

The heat loads were estimated for each complex to determine the surface cooling infrastructure requirements. Heat loads were derived from diesel mobile equipment, ventilation air auto-compression, strata heat, and electrical loads (from underground fans, conveyors, and other electrical loads). It is estimated that mine air cooling will not be required until mining depths reach 700 m below surface.

The heat and cooling loads for each of the North, Central and South Complexes are summarised in Table 16-59 and in Figure 16-60 to Figure 16-62, respectively. The major component of heat will be derived from the mobile diesel equipment (which has direct correlation on the airflow requirement underground and air-cooling potential).

Table 16-59: Summary of Heat Loads

	Unit	North Complex	Central Complex	South Complex
Diesel Equipment				
Total Engine Rating	kW	13 967	13 372	6 877
Diesel Total Heat	kW	22 649	21 684	11 152
Auto-compression				
Auto-compression Heat	kW	9 400	9 750	4 800
Strata Heat				
Strata Heat	kW	6 200	5 600	4 800
Broken Rock				
Production Rate	tpd	13 400	10 000	3 400
Broken Rock Heat	kW	2 132	1 591	541
Other Sources				
General Electrical Equipment Heat	kW	3 760	2 740	2 243
Conveyor Belt Heat	kW	4 366	3 514	1 951
Total Heat	kW	48 507	44 879	25 486
Natural Air Cooling	kW_R	-30 022	-27 337	-16 793
Refrigerated Air Cooling	kW_R	18 485	17 542	8 694

Figure 16-60: North Complex – Heating and Cooling Load Summary (48.5 MW_R)

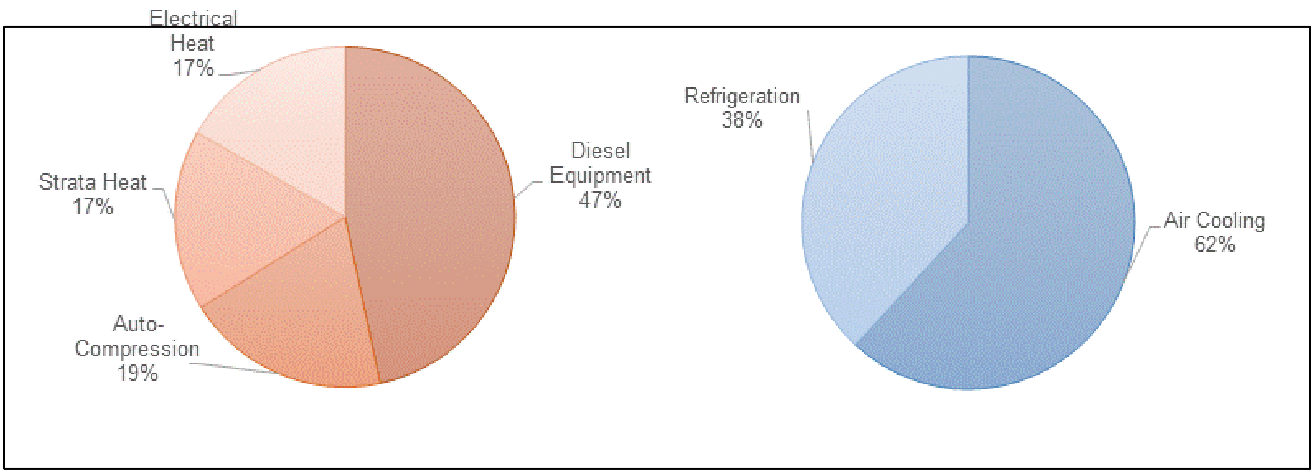


Figure 16-61: Central Complex – Heating and Cooling Load Summary (44.9 MW_R)

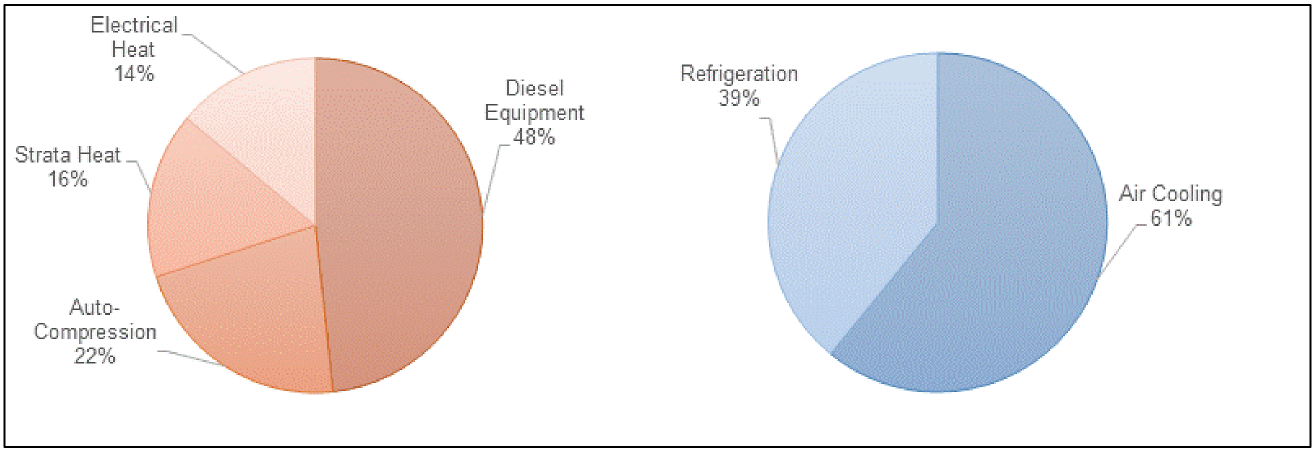
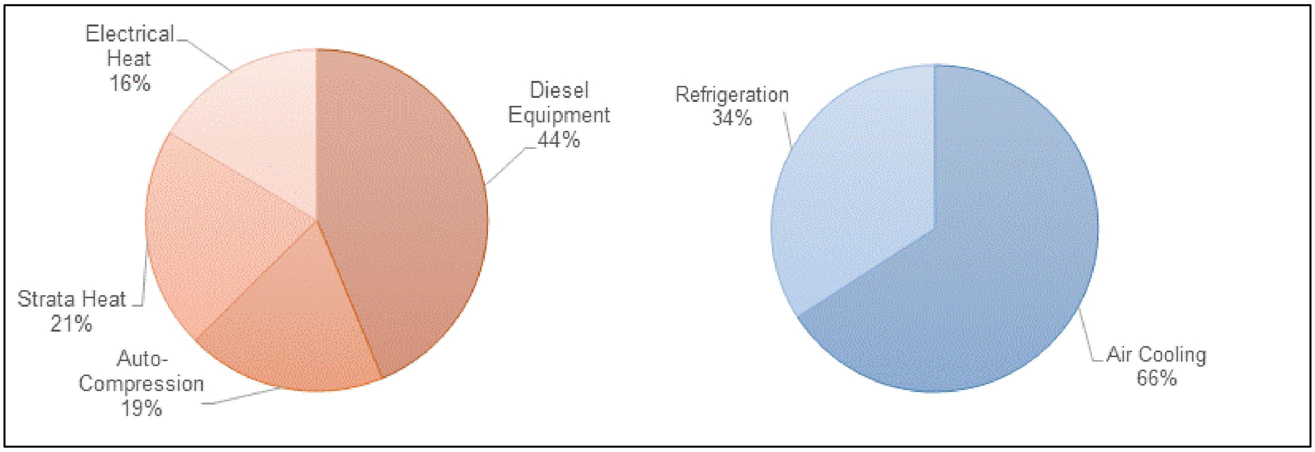


Figure 16-62: South Complex – Heating and Cooling Load Summary (25.4 MW_R)



16.4.8 Refrigeration

The heat loads will be countered by a combination of refrigerated air and uncooled air. The maximum operating reject temperature was based on 28.5°C WBGT. The required cooling duty is determined by the difference between overall heat load and natural cooling effect of the uncooled ventilation. Based on the heat loads outlined above, the cooling requirement will be 10 MW_R for each of the intake raises.

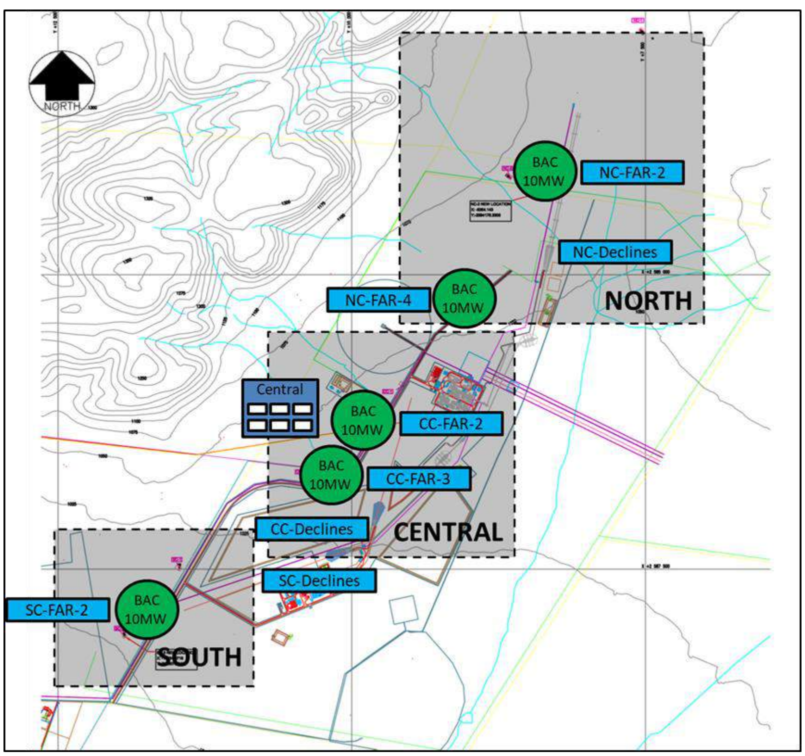
Since Central and South Complexes will be mined first, with North Complex being mined when Central and South Complexes are near completion, the full cooling requirement will not be required from the onset. The timing of mine air-cooling requirements is summarised in Table 16-60.

Table 16-60: Summary of Cooling Duty and Operation Period

Name	Size	Cooling Duty	Airflow Quantity	Schedule
Central Complex				
Declines	5 m x 5 m	No Cooling	120 m^3/s	2030 – 2049
CC- FAR-2	6 m Ø	10 MW_R	480 m^3/s	
CC-FAR-3	6 m Ø	10 MW_R	480 m^3/s	
North Complex				
Declines	5 m x 5 m	No Cooling	140 m^3/s	2048 – 2065
NC-FAR-2	6 m Ø	10 MW_R	530 m^3/s	
NC-FAR-4	6 m Ø	10 MW_R	480 m^3/s	
South Complex				
Declines	5 m x 5 m	No Cooling	140 m^3/s	2033 – 2055
SC-FAR-2	6 m Ø	10 MW_R	430 m^3/s	

To satisfy the cooling requirement, a central 30 MW_R refrigeration plant will be located at the Central Complex with piping to the BACs within each complex. This cooling distribution concept is outlined in Figure 16-63.

Figure 16-63: Schematic of Refrigeration Plan and Distribution of Cooling



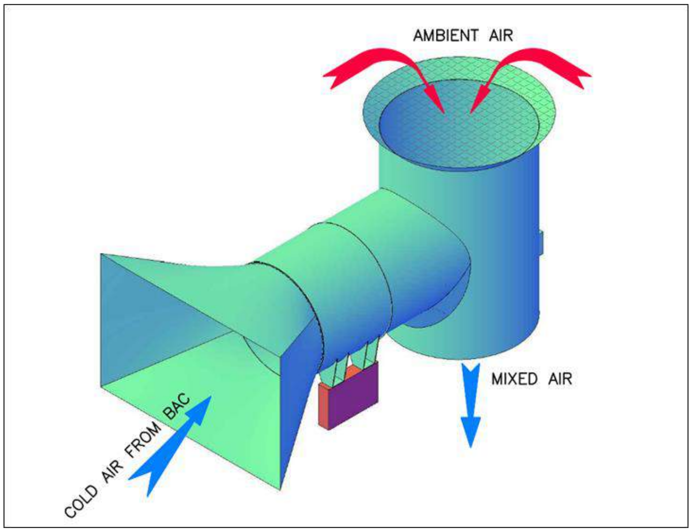
Not to Scale

16.4.9 Bulk-air Coolers

Each BAC will be sized for a nominal air-cooling duty of 10.0 MW_R. BACs will be concrete horizontal spray heat exchangers. For all 10.0 MW_R BACs, two-stage spray chambers will be used with chilled water sprayed in the first stage and resprayed in the second stage prior to returning to the refrigeration plant room via the warm water dam to be re-cooled.

The quantity of air through each BAC will be controlled by fans installed on the inlet. The fans are sized to overcome the BAC pressure only and will not push the ventilation system. Not all the air entering each intake raise will be cooled and some ambient air will bypass the BAC and mix with the cold air from the BAC at the top of the intake raise. The raise top arrangement will be designed to allow for this mixing of air and will be as shown in Figure 16-64.

Figure 16-64: Typical Shaft Top Arrangement for Bulk-air Coolers



There will be stench gas systems incorporated into the shaft top arrangements. These stench gas systems will be installed on the side of the vertical duct portion with a connection into the airstream. In the event of an emergency the system will be triggered delivering stench gas into the fresh air stream and in turn underground.

Refrigeration Plant

The water used at the BACs will be cooled by three pairs of refrigeration machines with the evaporators configured in lead-lag (series). Each lead-lag pair will deliver nominal capacity of 11.8 MW_R.

Each condenser and evaporator will be of the shell-and-tube type with water flowing through the tubes and refrigerant on the shell side. The condenser circuits will operate in a parallel arrangement and the water will split evenly to each refrigeration machine operating.

The BAC and refrigeration machines will typically operate continuously and at full load during hot summer conditions and part load during cooler conditions. For these cooler periods, the return water temperature will drop and the refrigeration machine load will be automatically reduced by pre-rotational guide vanes to maintain the predetermined set point.

The heat generated by the refrigeration machines will be rejected to a condenser water stream. This condenser water will flow to the heat rejection facility where it will be rejected to ambient air by means of six CCTs, each with a nominal heat rejection capacity of 7.0 MW_R and one CCT will be required per refrigeration machine operating.

16.5 Labour

The management, supervisory / technical, and skilled operators labour related to the underground mine for each complex is categorised in the following groups.

- Management
- Safety and Training
- Mine Engineering
- Geology
- Maintenance / Services / Construction / Material Handling
- Development
- Production
- Haulage

16.5.1 Labour Requirements

The estimated labour requirements are made up of Owner and Contractor labour. The labour requirements include a three-shift rotation (i.e., Rotation A, B, C) for certain staff and operations positions.

The peak and steady-state Owner's labour requirements for each complex are summarized in Table 16-61.

Table 16-61: Owner's Peak and Steady-state Underground Labour

Position	Central Complex Peak (2029)	Central Complex Steady State (2033)	South Complex Peak (2027)	South Complex Steady State (2033)	North Complex Peak (2046)	North Complex Steady State (2056)
Management	**3.5**	**3.5**	**3.5**	**3.5**	**4.0**	**4.0**
UG Mine Manager	1	1	1	1	1	1
UG Maintenance Resident Engineer	1	1	1	1	1	1
Safety, Health, Environment, and Quality (SHEQ) Manager	0.5	0.5	0.5	0.5	1.0	1.0
Technical Services Manager	1	1	1	1	1	1
Safety	**7**	**7**	**7**	**7**	**7**	**7**
SHEQ Officer	1	1	1	1	1	1
Compliance Safety Officer – Development	3	3	3	3	3	3
Compliance Safety Officer – Production	3	3	3	3	3	3
Mine Engineering	**30**	**30**	**25**	**25**	**28**	**35**
Engineer	1	1	1	1	1	1
TMM Engineer	3	3	3	3	3	5
UG Engineer	4	4	3	3	3	6
Ventilation and Hygiene Officer	1	1	1	1	1	1
Ventilation and Hygiene Assistant	3	3	2	2	2	3
Longhole Drilling & Blasting Operator	4	4	3	3	4	5
Senior Surveyor	1	1	1	1	1	1
Surveyor	4	4	3	3	4	4
Survey Helper	4	4	3	3	4	4
Rock Engineer	1	1	1	1	1	1
Backfill Engineer	1	1	1	1	1	1
Rock Engineer Assistant	3	3	3	3	3	3
Geology	**15.5**	**15.5**	**12.5**	**12.5**	**18.0**	**20.0**
Chief Geologist	1	1	1	1	1	1
Senior Resource Geologist	1	1	1	1	1	1
Senior Geologist	3	3	2	2	3	4
Diamond Drill Coordinator / Supervisor	0.5	0.5	0.5	0.5	1	1
Geologist – Core Logging	3	3	2	2	4	4
Geologist – UG Sampling, Mapping, Grade Control	5	5	4	4	5	6
Geology Helper – Core Handling	2	2	2	2	3	3
Maintenance / Services / Construction / Material Handling	**257**	**212**	**183**	**173**	**287**	**271**
Maintenance General Foreman	1	1	1	1	1	1
Maintenance Planner	3	3	3	3	3	3
Mechanic Supervisor	3	3	3	3	3	3
Surface Ventilation & Cooling Plant Maintenance	1	1	1	1	1	1
Lead Mechanic	3	3	3	3	3	3
Mechanic – UG Shop	52	37	30	26	67	55
Mechanic – Surface Shop for UG Fleet	12	10	8	9	12	12
Millwright Supervisor	1	1	1	1	1	1
Welder	3	3	3	3	3	3
Millwright	18	14	11	10	23	19
Electrical & Instrumentation Supervisor	1	1	1	1	1	1
Lead Electrician	3	3	3	3	3	3
Electrician	14	10	7	7	18	15
Instrumentation Technician	9	7	5	5	11	9
Construction / Services / Bulk Material Handling Supervisor	1	1	1	1	1	1
Cable Bolter Operator	3	3	3	3	3	3
UG Construction Worker	6	6	6	6	6	6
Construction Helper (Cable Bolt, Transmixer, etc.)	9	9	9	9	9	9
UG Backfill Construction Worker	12	12	12	12	12	12

Position	Central Complex Peak (2029)	Central Complex Steady State (2033)	South Complex Peak (2027)	South Complex Steady State (2033)	North Complex Peak (2046)	North Complex Steady State (2056)
Bulk Material Handling Operator	3	3	3	3	3	3
Conveyor Attendant	12	12	12	12	4	12
Rock Breaker Operator	12	12	9	9	12	15
Bit Sharpener	0	0	0	0	0	0
UG Labourer - Mine Services	69	51	42	36	81	75
Surface Labourer – Material Movement	6	6	6	6	6	6
Development	**82**	**40**	**49**	**40**	**127**	**61**
Mine Overseer – Development	1	1	1	1	1	1
Shift Boss – Development	6	3	3	3	9	6
Jumbo Operator	21	9	12	9	33	15
LHD Operator	21	9	12	9	33	15
Bolter Operator	24	12	15	12	36	18
Explosives Loading Operator	9	6	6	6	15	6
Production	**64**	**64**	**40**	**34**	**61**	**82**
Mine Overseer – Production	1	1	1	1	1	1
Shift Coordinator (Dispatch)	3	3	3	3	3	3
Shift Boss – Production	6	6	3	3	3	6
Slot Raise Driller	6	6	6	6	3	6
Production Driller	18	18	9	9	15	24
Blaster – Production	6	6	6	3	9	9
Blaster – Production	6	6	6	3	9	9
LHD Operator – Production	18	18	6	6	18	24
Haulage	**54**	**33**	**18**	**21**	**48**	**51**
LHD Operator – Truck Loading / Waste Handling	15	12	6	9	15	15
Haul Truck Operator – Production	18	18	6	6	21	33
Haul Truck Operator – Development	21	3	6	6	12	3
Grand Total	**513**	**405**	**338**	**316**	**580**	**531**

16.5.2 Overall Labour Profile

The Labour plan uses contractor labour for initial development and as the project period ends, the contractor labour is ramped down in a systematic way. It is assumed that a large portion of the contractor labour force will transition to the Owner's team. All production activities are completed by the Owner over the LOM. The LOM contractor labour will include raisebore operators and diamond drillers.

The contractor and Owner's labour profile for the Central Complex showing ramp up, steady state, and ramp down are represented graphically in Figure 16-55 and Figure 16-56.

Figure 16-65: Central Complex Underground Labour Ramp Up

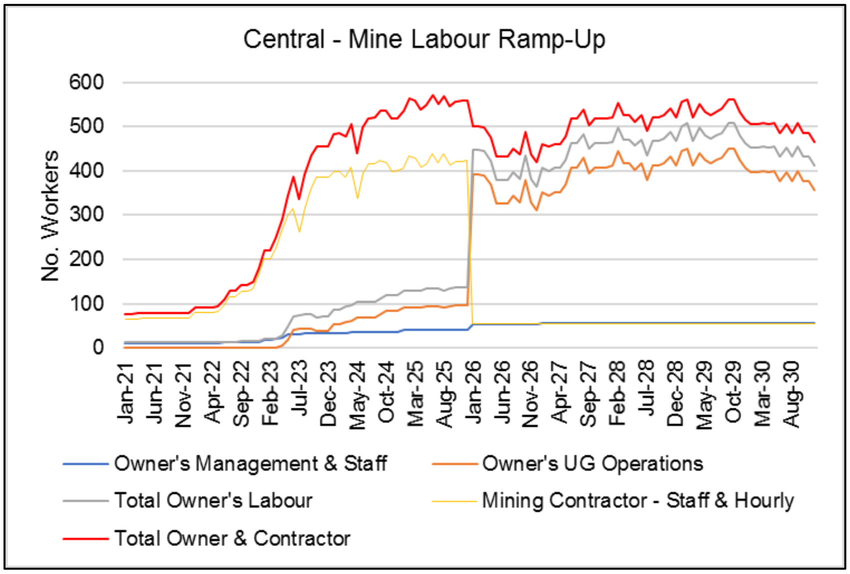
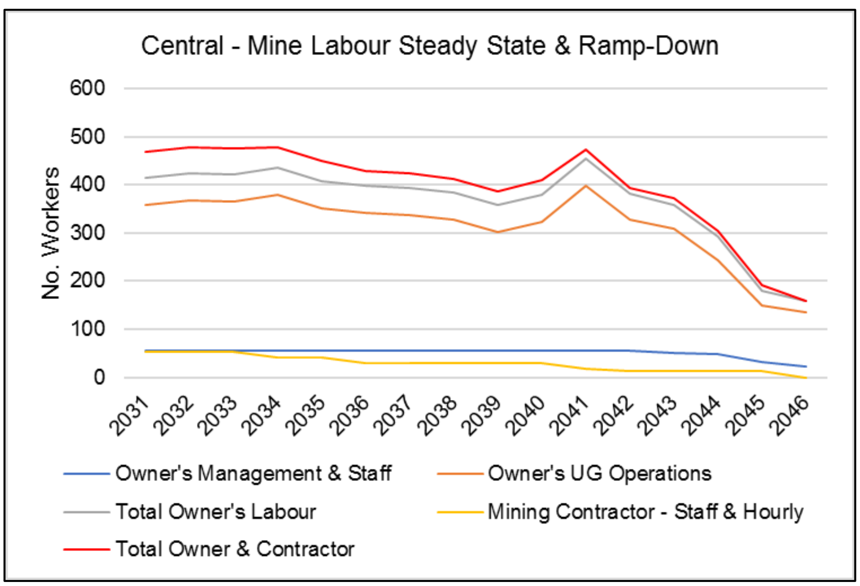


Figure 16-66: Central Complex Underground Labour Steady State and Ramp Down



The contractor and Owner's labour profile for the South Complex showing ramp up, steady state, and ramp down are represented graphically in Figure 16-67 and Figure 16-68.

The contractor and Owner's Labour profile for the North Complex showing ramp up, steady state, and ramp down are represented graphically in Figure 16-69.

Figure 16-67: South Complex Underground Labour Ramp Up

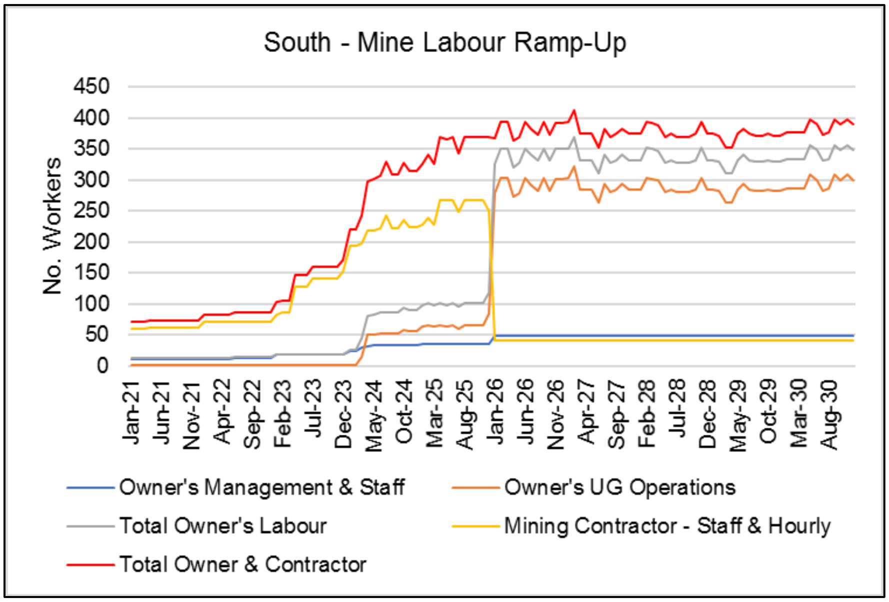


Figure 16-68: South Complex Underground Labour Steady State and Ramp Down

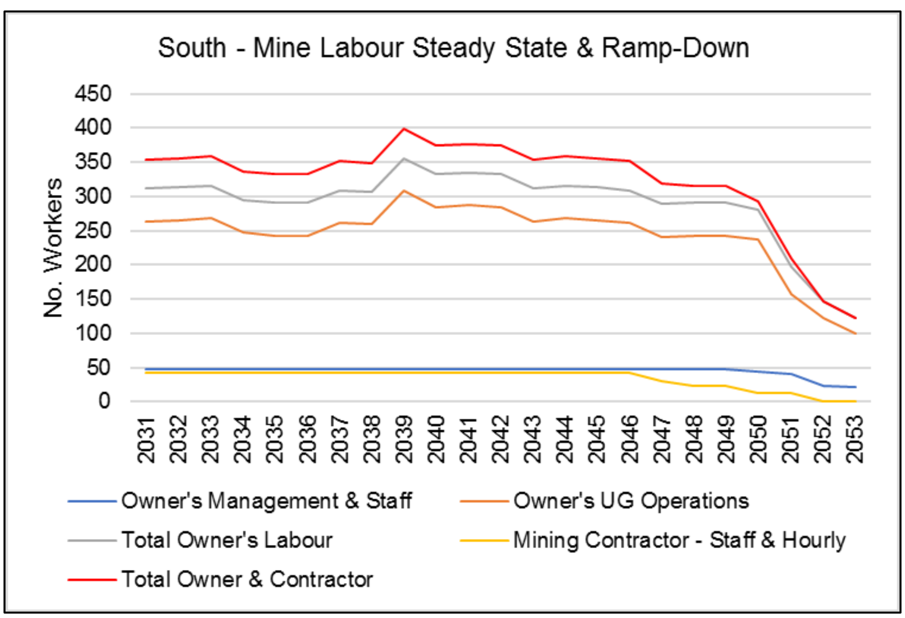
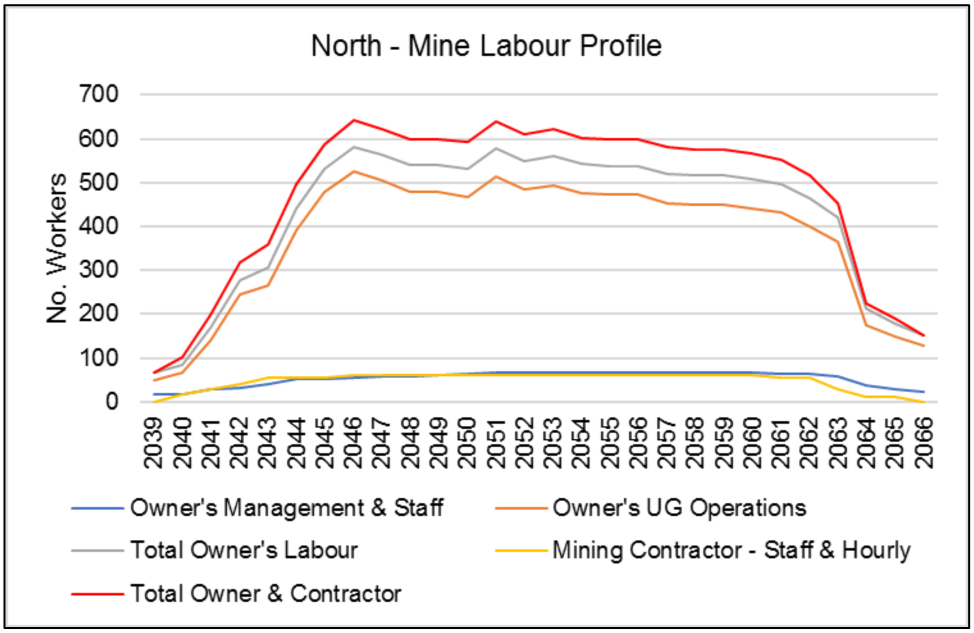


Figure 16-69: North Complex Underground Labour Profile



16.6 Mobile Equipment

The Waterberg Project will be highly mechanised using a diesel-powered mobile equipment fleet.

During the project period, a mining contractor will complete the development for the Main declines and initial sublevel development to establish key infrastructure, position underground diamond drills, and prepare for stope production. During this period, the mining contractor will use mobile equipment provided by the Owner.

The fleet will include development, production, and auxiliary equipment commonly used in the global mining industry. The type of mobile equipment and intended purpose is listed in Table 16-62.

Table 16-62: Mobile Equipment Type and Purpose

Unit	Purpose
Development	
2-Boom Jumbo	Drill Development Rounds
LHD – 17-t Class	Muck Development Rounds – Load Haul Trucks
Mechanical Bolter	Install Ground Support
Mobile Explosives Loader	Explosives Transfer and Charging
Production	
Slot Drill – ITH	Drill Slot Raises, Paste Backfill Holes, Drain Holes, Service Holes
Production Drill – Top Hammer	Drill Production Holes
LHD – 17-t Class	Stope Mucking – Equipped for Remote Control
Mobile Explosives Loader	Explosives Transfer and Loading
Blockholer	Drill and Blast Oversize Material – Equipped Remote
Truck Haulage	
LHD – 17-t Class (loading trucks)	Remuck ore and load trucks. Rehandle waste rock
50-t Trucks Production Ore	Haul from level to rock breaker/grizzly station
40-t Trucks Development	Haul development ore and waste rock
Construction and Services	
Shotcrete Sprayer	Ground support and construction
Concrete / Shotcrete Transmixer	Transport wet concrete/shotcrete from surface
Scissor Lift – Services (Pipe, Vent, etc.)	Install pipe and ventilation services
Scissor Lift – Construction	General Construction
Scissor Lift – Backfill	Install / Remove Piping – Construct Barricades
Cassette Truck – Material Movement	Move Material from/to Surface
Boom Truck – Material Movement	Move Material from/to Surface
Boom Truck – Construction	General Construction
Service LHD	Clean Sumps – Move Material – Equipped for Bucket, Forks, Basket Attachments
Water Tanker (Dust Suppression)	Dust Suppression in Ramps
Telehandler	Construction
Grader	Maintain Roadways
Forklift	Move Material
Cable Bolter (Drill and Install)	Drill and Install Cable Bolts
Maintenance	
Mobile Equipment Mechanic Truck	Service Equipment in the Field
Millwright Service Truck	Service Pumps, Vent Fans, Rock Breakers
Conveyor Service Truck	Service Conveyors
Electrician Tractor	Service Equipment – Install Cable – Field Service
Fuel / Lube Truck	Transfer Fuel / Lubes to Equipment in the Field and to Satellite Fuel Bays
Telehandler	Maintenance
Personnel Carriers	
Personnel Carrier – Large – 30 Person	Bus Style – Transfer Workers to Waiting Places
Personnel Carrier – Small – Services	Distribute Workers to Workplaces
Surveyor Tractor	Equipped with Basket
Geology Tractor	
Diamond Drill Contractor Tractor	
Pick-up Truck – Mine General Foreman Prod	Toyota Landcruiser or Equivalent
Pick-up Truck – Mine General Foreman Dev	
Pick-up Truck – Development Supervisor	
Pick-up Truck – Production Supervisor	
Pick-up Truck – Construction Supervisor	
Pick-up Truck – Maintenance Supervisor	
Pick-up Truck – Supervisor	
Pick-up Truck – Technical Services	
Pick-up Truck – Contractor	

16.6.1 Fleet Size

The fleet size for each complex was determined based on the underground development, production, construction, maintenance, and services activities to achieve the development and production schedule.

16.6.1.1 Development Fleet

The development fleet for each complex is determined from the total scheduled advance metres and the performance that each jumbo can achieve considering the development heading size, ground support requirements, and the number of working faces available. Generally, except for initial decline development, each jumbo will have multiple workplaces to cycle development rounds.

Each jumbo will be matched with an LHD and a mechanical bolter and there will be an additional mechanical bolter in the fleet dedicated to ground rehabilitation. The number of development emulsion explosive loading units was determined based on capacity to load two development rounds per shift (or approximately one explosives loader per two development crews).

16.6.1.2 Production Fleet

The production fleet for each complex was determined from the total scheduled stope tonnes, stope cycle productivities, and performance that each production drill and LHD can achieve.

ITH drills will be required for drilling the slot raises for stopes using the Machine Roger V30 reaming head. The ITH drill will have a portable compressor located at the drill site. The ITH will also be used to drill service holes for paste backfill distribution, drain holes, and electrical holes (for running cable from level to level).

Top-hammer production drills will be used for production drilling 76 mm diameter longholes. Each production drill will average approximately 1 700 tonnes drilled per day.

Each 17-t capacity production LHD will average 1 600 tpd mucking from the stope and dumping into a remuck located within approximately 150 m from the stope. The LHDs for re-handling ore from the remuck and loading trucks is included in the haulage fleet.

Two emulsion explosive loading units have been included to provide flexibility to load two stopes simultaneously.

16.6.1.3 Truck Haulage

All waste rock and ore will be trucked from the development or production area to an identified dump point.

Development Waste Rock Haulage

Development waste rock will be loaded into 40-t class haul trucks and hauled to a remuck for subsequent placement into a stope or onto a grizzly by an LHD for conveying to surface, or to a surface dump point located near the box cut. The haulage rate (tpd) from each sublevel to surface or dump points was estimated and applied to the tonnes generated from each level based on the development schedule.

Ore Haulage

Ore will be loaded into 50-t capacity trucks and hauled to rock breaker / grizzly stations for sizing and loading onto the conveyor system. The haulage fleet for each complex was determined from the total scheduled stope tonnes from each sublevel and the distances to grizzly / rock breaker stations. The capacity of each rock breaker was estimated to be 2 500 tpd (base on input from vendors and benchmarking operations). For each sublevel, a primary (i.e. preferred) dump point was identified as well as an alternate dump point (i.e. further haul distance). If the capacity of a rock breaker was reached (based on multiple trucks hauling to same location), the alternate route was considered in the haulage rate.

16.6.1.4 Construction, Services, Maintenance, and Personnel Carriers Fleet

The auxiliary equipment fleet for construction, services, and maintenance, and for personnel movement was estimated based on the level of development, construction, and production activities.

16.6.2 Peak and Steady-state Fleet Size

The peak and steady-state mobile equipment fleet for each complex is summarised in Table 16-63.

The mobile equipment profile showing ramp-up, steady state, and ramp down for each complex are represented graphically in Figure 16-70 through Figure 16-74.

Table 16-63: Peak and Steady-state Mobile Equipment by Complex

Item	Central Complex Peak (2029)	Central Complex Steady State (2033)	South Complex Peak (2028)	South Complex Steady State (2033)	North Complex Peak (2051)	North Complex Steady State (2057)
Development	**25**	**15**	**18**	**12**	**32**	**15**
2-Boom Jumbo	7	4	5	3	9	4
LHD Development 17-t Class	7	4	5	3	9	4
Mechanical Bolter	8	5	6	4	10	5
Explosives Loader	3	2	2	2	4	2
Production	**17**	**16**	**8**	**9**	**18**	**22**
Slot Drill – ITH	2	2	2	2	1	2
Production Drill – Top Hammer	6	5	2	3	6	8
LHD Production 17-t Class	6	6	2	2	7	8
Explosives Loader	2	2	1	1	3	3
Blockholer	1	1	1	1	1	1
Truck Haulage	**15**	**11**	**7**	**7**	**16**	**18**
LHD Truck Load/Waste Rehandle 17-t Class	5	4	3	3	6	5
50-t Trucks Production Ore	6	6	2	2	8	12
40-t Trucks Development	4	1	2	2	2	1
Construction and Services	**28**	**23**	**23**	**19**	**35**	**30**
Shotcrete Sprayer	3	2	3	2	4	2
Concrete / Shotcrete Transmixer	3	2	3	2	4	2
Scissor Lift – Development Services	3	2	2	1	3	2
Scissor Lift – Construction	2	1	2	1	3	1
Scissor Lift – Backfill	2	2	2	2	2	3
Cassette Truck	2	2	1	1	2	3
Boom Truck – Material Movement	3	2	1	1	3	3
Boom Truck – Construction	1	1	1	1	2	2
Service LHD	3	3	3	3	4	4
Water Tanker (Dust Suppression)	1	1	1	1	1	1
Telehandler	2	2	1	1	3	3
Grader	1	1	1	1	1	1
Forklift	1	1	1	1	2	2
Cable Bolter (Drill and Install)	1	1	1	1	1	1
Maintenance	**12**	**12**	**10**	**10**	**13**	**17**
Mobile Equipment Mechanic Truck	3	3	2	2	3	4
Millwright Service Truck	2	2	2	2	3	3
Conveyor Service Truck	2	2	2	2	2	2
Electrician Tractor	2	2	2	2	2	3
Fuel / Lube Truck	2	2	1	1	2	3
Telehandler	1	1	1	1	1	2
Personnel Carriers	**30**	**29**	**26**	**25**	**35**	**34**
Personnel Carrier – Large – 30 Person	3	3	2	2	4	4
Personnel Carrier – Small – Services	3	3	2	2	5	5
Surveyor Tractor	2	2	2	2	2	2
Geology Tractor	2	2	2	2	2	3
Diamond Drill Contractor Tractor	1	1	1	1	1	1
Pick-up Truck – Mine General Foreman Prod	1	1	1	1	1	1
Pick-up Truck – Mine General Foreman Dev	1	1	1	1	1	1
Pick-up Truck – Development Supervisor	2	1	2	1	3	1
Pick-up Truck – Production Supervisor	2	2	1	1	2	2
Pick-up Truck - Construction Supervisor	1	1	1	1	1	1
Pick-up Truck – Maintenance Supervisor	1	1	1	1	1	1
Pick-up Truck – Other Supervisors	7	7	7	7	7	7
Pick-up Truck – Technical Services	3	3	2	2	4	4
Pick-up Truck – Contractor	1	1	1	1	1	1
Total Mobile Equipment Fleet – Operating	**127**	**106**	**92**	**82**	**149**	**136**

Figure 16-70: Central Complex Mobile Equipment Ramp-up

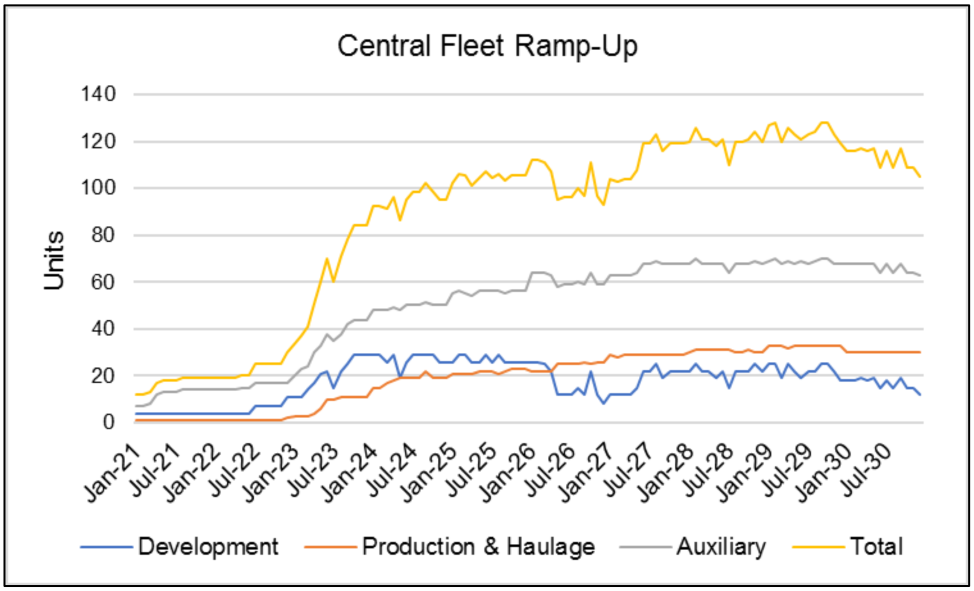


Figure 16-71: Central Complex Mobile Equipment Steady State to Ramp-Down

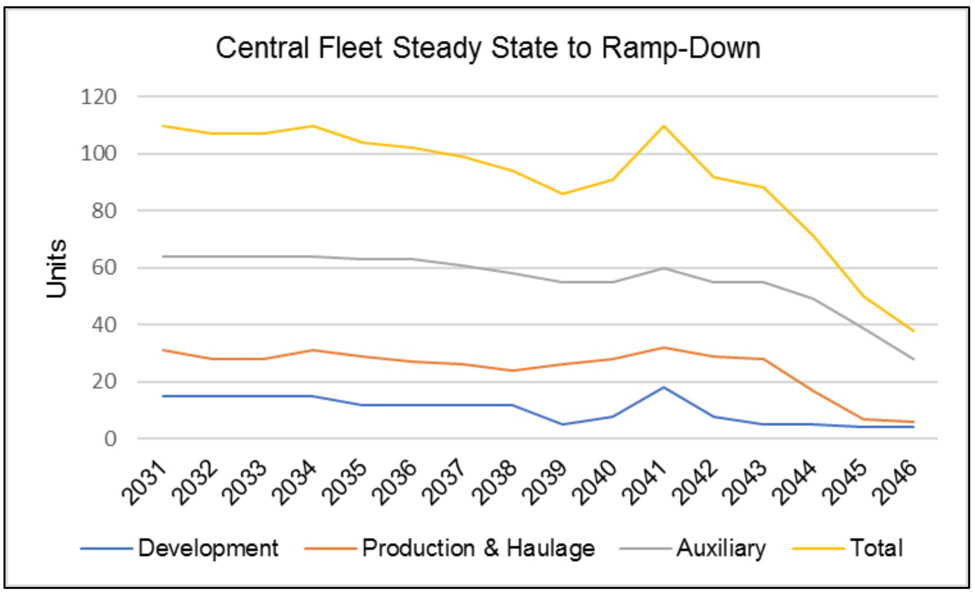


Figure 16-72: South Complex Mobile Equipment Ramp Up

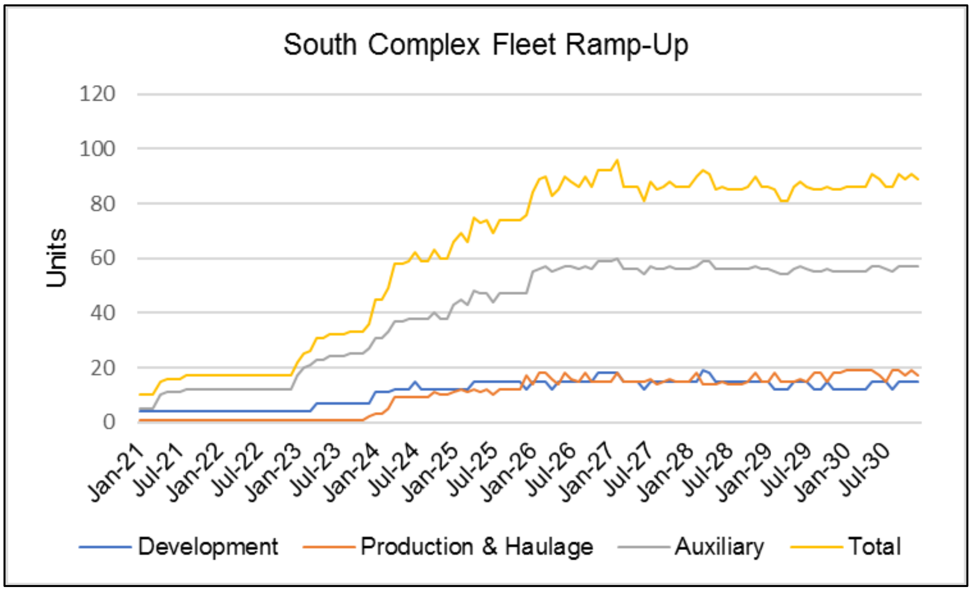


Figure 16-73: South Complex Mobile Equipment Steady State to Ramp Down

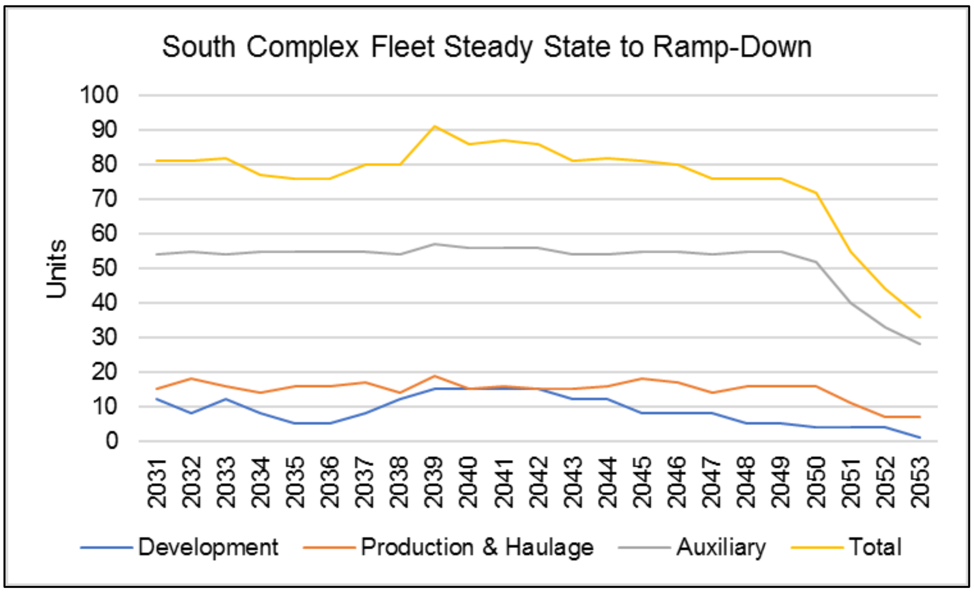
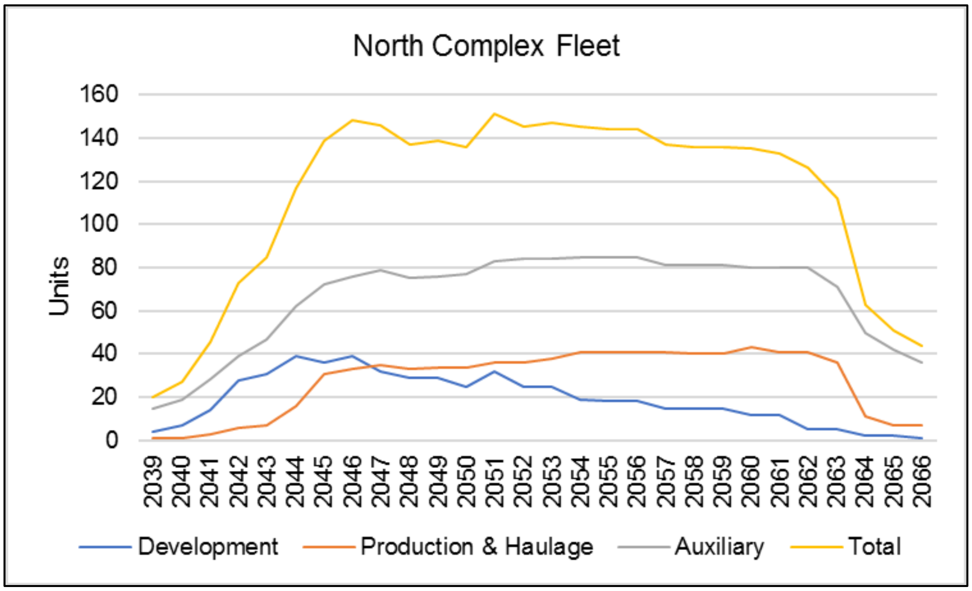


Figure 16-74: North Complex Mobile Equipment Profile



16.7 Underground Infrastructure

Following is the underground infrastructure to support mining operations for each complex

- Refuge Stations and Latrines
- Ore and Waste Rock Handling Systems
- Mine Dewatering
- Maintenance Facilities
- Explosives Handling and Distribution
- Fuel and Lubrication
- Mine Services (service water, fire water, potable water, compressed air,)
- Electrical Distribution and Communications

16.7.1 Refuge Stations

Permanent and portable refuge stations will be required underground to ensure personnel have a safe location to retreat to during mine emergencies. The maximum distance personnel will walk to a refuge station in an emergency is 500 m. Refuge stations will comply with current regulations and legislation, including the Mine Health and Safety Act of 1996 (Act No. 29 of 1996).

16.7.1.1 Permanent Refuge Stations

Permanent refuge stations / waiting places will be located near the main workshops and the satellite workshops. There are four permanent refuge stations in the North Complex, three in the Central Complex, and four in the South Complex. The permanent refuge stations located near the Main Workshops will be equipped with a compressed air line from surface.

In addition to being used during emergency conditions, permanent refuge stations will be used as lunchrooms and waiting places and will also be equipped with an office area.

Permanent refuge facilities will be designed for a capacity of 24 people for 24 hours during emergency conditions and will include the following items.

- Uninterruptible Power Supply of up to 24 Hours (without reliance on mine power)
- Breathable Air (Oxygen) Supply (compressed air) and /or Oxygen Generator
- Self-rescuers (quantity equal to the capacity of the station)
- Shelving with Emergency Food and Water Supply
- Carbon Dioxide and Carbon Monoxide Scrubbers
- Communications Equipment
- Air Conditioning Equipment
- Inside and Outside Environmental Gas Monitor
- Portable Latrine with Supplies
- Service Water Hose Rack
- Lighting with Battery Backup
- Seating for 24 People
- Sink with Potable Water and Water Heater
- Fire Extinguisher and Portable Eye Wash
- First-aid Equipment

16.7.1.2 Portable Refuge Stations

Portable refuge stations will be located at key areas and near the working face in headings being developed away from the complex's main infrastructure. Portable refuge stations will be used during emergency conditions only.

Portable refuge stations will be self-contained manufacturer-supplied and located in purpose-built or repurposed excavations. Each portable refuge station is capable of housing 16 people for 36 hours and will have similar features as the permanent refuge stations, except service water supply piping, sink, and office area will not be included. Portable refuge stations will be supplied with oxygen by bottled systems and not through a compressed-air line.

16.7.1.3 Latrine Stations

Latrine stations will be located on select sublevels in all three complexes. Each latrine station will have a toilet(s) and a sink with potable water

16.7.2 Ore and Waste Handling Systems

16.7.2.1 Ore Handling

Ore will be mucked from the stopes and ore development headings using 17-t class LHDs. LHDs will muck from the stopes and dump into a nearby remuck bay and a separate LHD will be dedicated to remucking the ore and loading the 50-t capacity haul trucks. The ore will be subsequently remucked and loaded into a 50-t capacity haul trucks.

The trucks will haul the ore from the remuck to the nearest available rock breaker station. The rock breaker stations will be located at strategic locations depending on the ore tonnages distribution (i.e. more frequent rock breakers in the higher tonnage areas). The South Complex will have 300 x 300 mm grizzly openings versus 400 x 400 mm grizzly openings in the Central and North Complexes. Grizzly sizes were selected to meet the daily production requirements of 10 000 tpd, 3 400 tpd, and 13 400 tpd for the Central, South, and North Complexes, respectively. Sunken grizzly designs capable of handling approximately two truckloads will be provided complete with 75 mm thick wear liners, fixed heavy-duty rock breakers, control booths, automatic lubrication systems, and hydraulic power packs with integral Ansul fire protection systems. Rock breaker station accesses will have roll-up doors to prevent ventilation bypass. Ventilation fans and dust suppression will be provided at each station.

The number of rock breaker stations at each complex are summarised in Table 16-64.

Table 16-64: Rock Breaker Stations

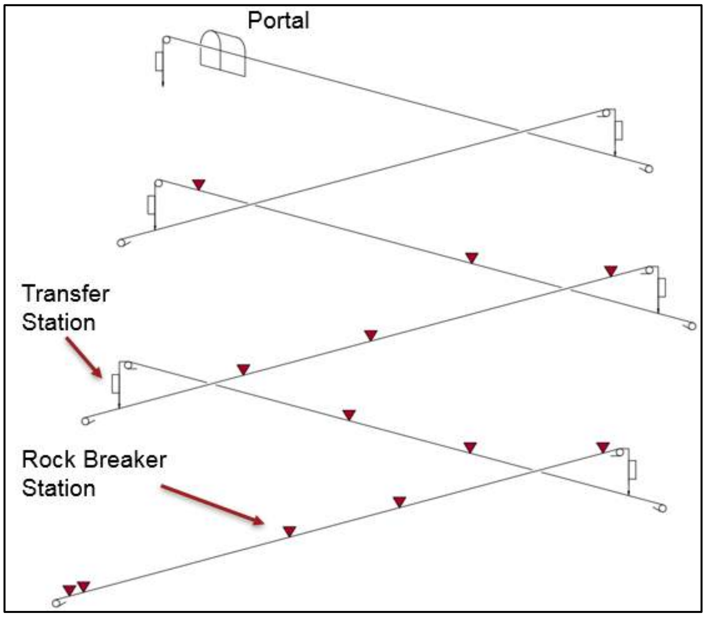
Complex	Number of Rock Breaker Stations
Central	12
South	6
North	15

Beneath each grizzly station at the conveyor level will be a transfer station comprised of a 3.0 m x 3.0 m surge bin (approximately 200-t capacity), transfer chute, vibrating feeder, and belt tramp metal magnet. The chutes will have solid ore bed depth control / maintenance doors operated with hydraulic cylinders. Chutes beneath the ore pass will be fitted with 75 mm thick wear liners. Maintenance platforms will be placed around the overhead-supported vibratory feeders and the tramp metal magnet. Dumping of the magnets and positioning of the bed depth control doors will be performed manually. The vibratory feeder flow control will be automated with feedback from a local belt scale and conveyor bed depth monitor. To meet mine production requirements, the South Complex will require two to three stations in operation at any one time while the Central and North Complexes will require four to six stations operating.

For areas producing ore before the first rock breaker stations is established, ore will be hauled by truck to surface.

All complexes will have similar ore handling systems, that include rock breaker stations for sizing ore and feeding a series of conveyors located in a dedicated decline developed in the footwall that ascend from the lower elevations of the mine to surface at 15.8% gradient (9º). Transfer stations will be required to change the conveyor direction as the system traverses the extents of the complex while ascending. A schematic demonstrating the footwall conveyor system for the Central Complex is shown in Figure 16-75.

Figure 16-75: Schematic of Footwall Conveyor System – Central Complex



Each system is designed to meet production requirements based on available total effective shift length per day, planned maintenance, and equipment reliability based on unplanned downtime. System utilization, based on a 24-hour day, will range between 48.5% and 52.6% for the three complexes. The shift work time and material handling equipment sizing parameters for each mining complex are summarised in Table 16-65.

Table 16-65: Material Handling Equipment Sizing Parameters

Item	Central Complex	South Complex	North Complex
Production Data			
Effective Shift Time Per Day (Hours)	15.00	15.53	15.53
Conveyor Operating Days per Year	353	353	353
Daily Ore Throughput (tpd)	10 000	3 400	13 600
Ore Bulk Density (SG)	2.07	2.07	2.07
Ore Moisture Content % w/w)	3	3	3
Sized Ore P80	400	300	400
Operating Data			
Conveyor and Feeder Reliability (%)	98	98	98
Quantity of Inline Conveyors and Feeders	6	9	6
System Reliability (%)	0.89	0.83	0.89
Weekly Planned Maintenance (Hours)	8	10	8
Quarterly Planned Maintenance (in addition to weekly) (Hours/Qtr)	8	8	8
Yearly Planned Maintenance (in addition to weekly and quarterly) (Hours/Yr)	8	8	8
Total Yearly Planned Maintenance (Hours/Yr)	456	560	456
Total Available Shift Hours per Year	5 295.0	5 483.3	5 483.3
Available Production Hours per Year	4 286.6	4 104.8	4 453.4
Effective Production Hours per Day	12.14	11.63	12.62
Hourly Production Target (tph)	823	292	1 078
Overall System Utilization (based on 24-hour day) (%)	50.6	48.5	52.6
Equipment Sizing			
Equipment Design Factor (%)	20	20	20
Conveyors (tph)	988	351	1 293
Feeders (tph)	988	351	1 293
Actual Design (tph)	1 000	350	1 300

Conveyor systems for each mining complex are a switch-back design from the lower mining levels to the surface portal at 15.8% gradient (9° gradient). Designs were completed for the varying angles for changes of direction at the conveyor transfer points. Conveyors were sized based on the grizzly openings size in each complex as they are required to handle large lump sizes. The conveyor belts at the Central and North Complexes will be 1 200 mm W while the South Complex belts will be 1 050 mm W. Conveyor belts will be Mine Safety and Health Administration (MSHA) rated fire-retardant Anti-Static type. The following three controls will be in place to ensure the belt and motors do not become overloaded beyond the belt or drive system capacity.

- Belt Scales at each Feeder Station
- Belt Level Detection at each Feeder Station
- Amperage Monitoring of the Drives Interlocked with the Feeders

All conveyors will have variable speed control drives to provide appropriate motor load sharing as the drives are typically dual or quad drive arrangements. Belt construction is typically steel cord due to the long belt lengths; however, there are some multi-fabric belts, where applicable. The transfer station for each conveyor will be complete with maintenance platforms, overhead cranes, and guarding.

Conveyors will be a stringer-style design complete with outboard guarding for personnel safety and heavy-duty CMEA E greased sealed-for-life idlers. Conveyors will be chain hung from the back. Fire protection sprinklers and fire hose reels will be provided along the entire length of each belt. Belt catch mechanisms and roll pack protection will be provided on each conveyor. Tension release on the back stop will be provided for personnel safety. Conveyor take ups are a winch style take up most suitable for underground. Final surface termination of the underground systems will be in the vicinity of the portals. At the South Complex, the conveyor terminates at the surface jaw crushing station. The Central and North Complex systems report to a separate transfer conveyor.

16.7.2.2 Waste Rock Handling

LHDs will be used to muck waste rock from development headings. The LHDs will load material into haul trucks for transport via the service decline to a surface stockpile, to a remuck for disposing of into a mined stope or for batching through a rock breaker onto the conveyor system when not transferring ore. During the production period, mined-out stopes will be utilised, whenever appropriate, to dispose of development waste rock. It is estimated that 18% of waste rock will be disposed of in stopes as backfill.

16.7.3 Mine Dewatering

The mine dewatering system will be a 'dirty' water system with minimal settling of fines underground. The settling of fines will be managed on surface.

Each complex will have similar dewatering design philosophies and equipment. Each system is designed to meet the total dewatering requirements for the complex with a 1.5 safety factor to accommodate upset conditions. The sources of water will include groundwater, service water from drilling, dust suppression, backfill, and potable water. Rainfall at the portal will be collected in a portal sump and pumped with a submersible pump to a pond on surface to prevent rainwater from entering the conveyor and main service ramps.

The dewatering systems for the complexes will contain the following three main elements.

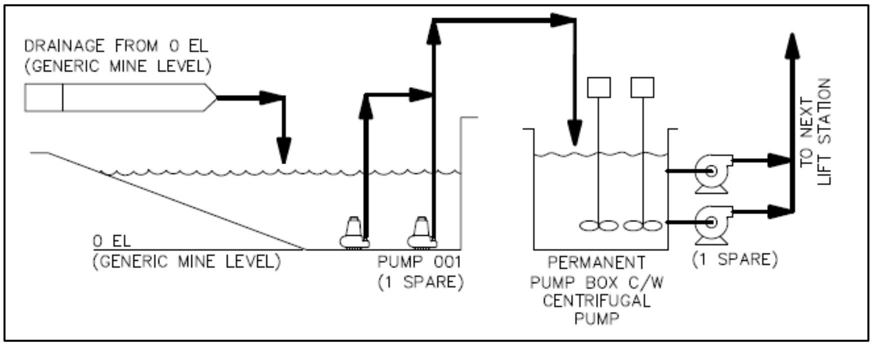
- Sublevel collection sumps with temporary submersible pumps and subsequent drill holes to gravity drain to collection and transfer sumps on lower sublevels. Active workplaces and rock mass inflows will drain to these collection sumps.
- Sublevel collection and transfer sumps with submersible pumps to transfer water to Pump Boxes
- Pump Boxes with Horizontal Centrifugal Pumps – Located in the conveyor decline and transfer water to surface

These sumps will collect and stage water to surface in the following general order as development progresses deeper in each area on the complexes.

16.7.3.1 Stage 1 Pumping

Stage 1 Generic Level 0 – A collection sump is constructed with submersible pumps, which feed directly into the Level 0 Pump Box for pumping to surface. A Stage 1 pumping schematic is shown in Figure 16-76.

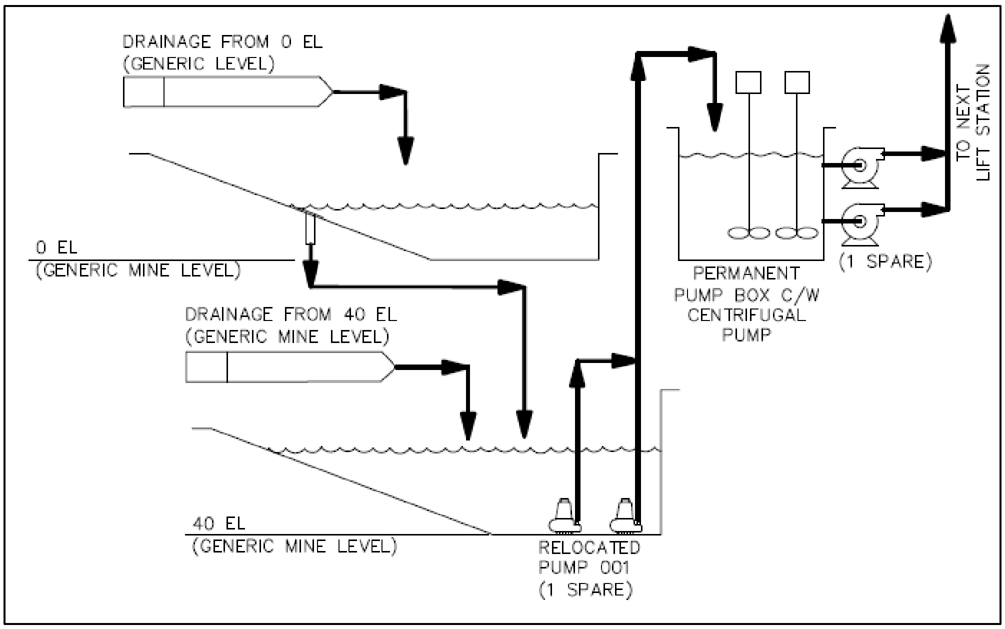
Figure 16-76: Stage 1 Pumping Schematic



16.7.3.2 Stage 2 Pumping

Stage 2 Generic Level 40 (20 m to 40 m below Level 0) – The submersible pumps from the Level 0 sump will be removed and drill holes will be drilled to allow water to gravity flow to the Level 40 sump. Submersible pumps will be relocated to the new sump at Level 40 and will pump up to the Level 0 Pump Box for pumping to surface. A Stage 2 pumping schematic is shown in Figure 16-77.

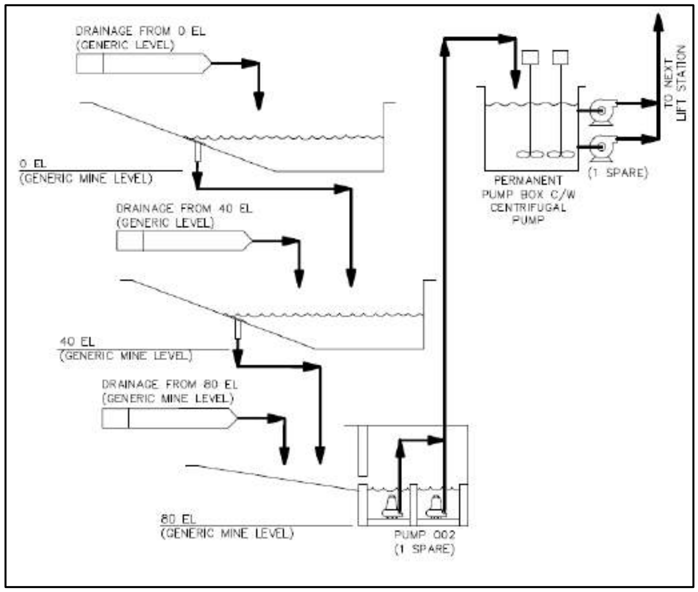
Figure 16-77: Stage 2 Pumping Schematic



16.7.3.3 Stage 3 Pumping

Stage 3 Generic Level 80 (60 m to 80 m below Level 0) – Collection and transfer sump with submersible pumps will be constructed. The submersible pumps from the Level 40 sump will be removed and drill holes drilled to allow water to gravity flow to the Level 80 sump. The Level 80 collection and transfer sump will be equipped with submersibles pumping up to the Level 0 Pump Box for pumping to surface. A Stage 3 pumping schematic is shown in Figure 16-78.

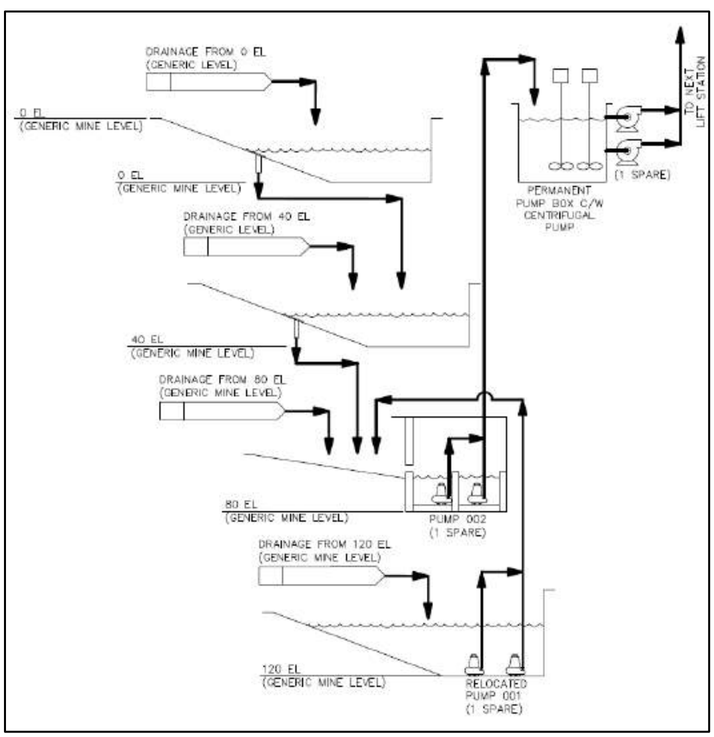
Figure 16-78: Stage 3 Pumping Schematic



16.7.3.4 Stage 4 Pumping

Stage 4 Generic Level 120 (100 m to 120 m below Level 0) – A floor sump with submersible pumps will be constructed. Submersible pumps will pump dirty water from this sump to the Level 80 collection and transfer sump. A Stage 4 pumping schematic is shown in Figure 16-79.

Figure 16-79: Stage 4 Pumping Schematic



Following Stage 4, the process repeats for the remaining sublevels. The main pump box stations with centrifugal pumps will be located in the conveyor decline at approximately every 160 m vertical elevation. The pump boxes and centrifugal pumps will cascade dirty water up the conveyor decline to surface for settling.

The dewatering requirements for each complex were estimated for the period of peak average inflows and service water usage. The dewatering requirements and the number of pump stations are summarised in Table 16-66.

Table 16-66: Peak Average Water Inflows and Quantity of Equipment

	Central Complex	South Complex	North Complex
Groundwater Inflow	1 085 L/min	1 498 L/min	1 146 L/min
Service Water Inflow	1 151 L/min	767 L/min	1 290 L/min
Potable Water Inflow	39 L/min	58 L/min	62 L/min
Backfill	292 L/min	97 L/min	390 L/min
Total Water Inflow	2 567 L/min	2 420 L/min	2 888 L/min
Quantity of Type 1 Pump Box Station (250 kW)	0	0	6
Quantity of Type 1 Pump Box Station (200 kW)	5	4	0
Quantity of Type 2 Pump Box Stations (90 kW)	3	2	5
Quantity of Type 2 Pump Box Stations (55 kW)	2	3	5
Quantity of Collection Transfer Sumps (30 kW)	14	13	20

16.7.4 Maintenance Facilities

Mobile equipment that frequently travels to surface as part of normal operation will be serviced at the surface maintenance shop, while equipment that is generally confined underground will be serviced in underground maintenance shops. The type equipment that will be serviced on surface versus underground are summarised in Table 16-67.

Table 16-67: Mobile Equipment Service Location

Equipment Type	Surface Shop	Underground Shop
Drills (Jumbo, Bolter, Blockholer, Production, Cable Bolter)	0%	100%
Explosives Loader	0%	100%
LHD	0%	100%
50-t Haul Truck	0%	100%
40-t Haul Truck	50%	50%
Shotcrete Sprayer	0%	100%
Transmixer	100%	0%
Scissor Lifts	0%	100%
Cassette Trucks, Boom Trucks, Water Tanker, Fuel Lube	100%	0%
Maintenance Service Vehicles	50%	50%
Grader	50%	50%
Personnel Carriers	75%	25%

The estimated number of mobile equipment units that will be serviced and/or undergoing minor repairs at any given time is estimated to be 15% of the total fleet, and it is assumed that 80% of these units will be serviced/repaired in a shop with the remaining serviced in the field. The average number of units serviced in bays in underground shops in each complex are summarised in Table 16-68.

Table 16-68: Average Mobile Equipment Serviced in Service Bays

Complex	No. Bays
Central Complex	9
South Complex	7
North Complex	12

During the initial decline development at each complex all mobile equipment will be serviced in the field or at the surface shop. Once development reaches the underground workings a satellite shop will be established to facilitate routine servicing and minor repairs.

There will be two types of underground workshop configurations at each complex; a Main Workshop that will be located near the centre of underground activity, and smaller Satellite Workshops located closer to work areas where travel distances to the Main Workshop are extensive. The number and location of the workshops in each complex are summarised in Table 16-69.

Table 16-69: Workshop Locations by Complex

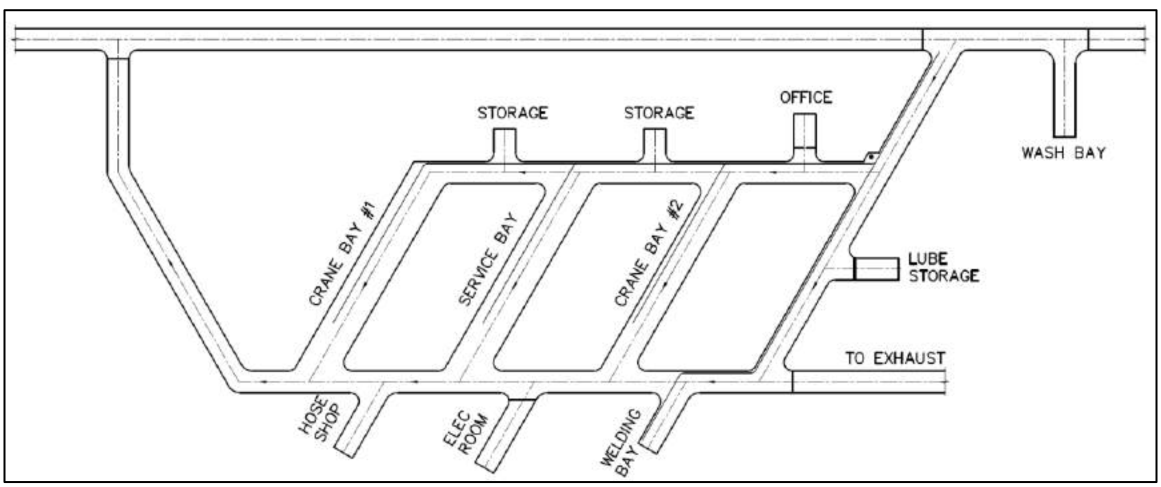
	Central Complex	South Complex	North Complex
Main Workshop	620 L	580 L	460 L
Satellite Workshop	2 x 400 L 960 L	320 L 700 L 920 L	2 X 260 L 2 x 780 L

The Main Workshop will have a compressed-air supply from the surface plant and service water, potable water, and fire water services will be supplied from surface via piping routed through the conveyor and main service declines. Fire detection and suppression equipment that interfaces with each complex's central alarm system will be provided for the workshops.

16.7.4.1 Main Workshop

The Main Workshops will be multi-bay facilities that can service up to six vehicles, each including a service bay, two crane bays, welding bay, office, hose shop, electrical equipment room, lubricant storage, and additional storage bays. The Main Workshops will be located in areas with sufficient room for potential expansion. The key features of the Main Workshop are shown in Figure 16-80.

Figure 16-80: Key Features of Main Workshops



Two 25-t cranes will be provided in each crane bay to enable multiple vehicles to be serviced at the same time.

A ramp with removable grating for access to the underside of mobile equipment, a trench drain, sump, and oil water separator will be installed in each service bay. The largest piece of equipment to be serviced in this workshop will be a 50-t haul truck.

Ventilation for the workshop will be flow-through to a nearby exhaust raise. Fire rated roll-up fire doors will be provided at the entrance and exit of the crane bays and service bays.

The lube storage bay will have fire-rated double man doors.

A wash bay, main fuel and lube station, permanent refuge / waiting station with two latrines, tire storage bay, parking, and other storage bays will be located near each main workshop.

16.7.4.2 Satellite Workshop

Smaller single-bay satellite workshops will be located near working areas at select levels in each complex. These workshops each have a 25-t crane, service water and compressed-air hose reels, communications, fire roll up doors, and fire-suppression sprinklers. These workshops are intended to support servicing and minor repairs for limited-travel equipment.

Service water and fire water will be supplied from surface to the satellite workshops via piping routed through the declines. A portable compressor will be provided in each satellite workshop to supply compressed air for tools.

Wash bays, satellite fuel and lube bays, permanent refuge / waiting station with latrines, parking, and storage areas are located on the same level as the satellite workshops.

16.7.4.3 Wash Bay

There will be a Wash Bay located adjacent to the Main Workshop areas and the Satellite Workshops for cleaning vehicles prior to maintenance.

16.7.5 Fuel and Lubrication

There will be main fuel and lubricant stations and satellite stations fuel and lubricant stations located underground. These stations will support diesel fuel and lubricant storage and distribution for diesel-powered mobile equipment used for underground development, production, construction, and movement of materials and personnel.

One main fuel and lubricant station will be located in each complex, while smaller satellite fuel and lubricant stations will be located near the satellite workshops and work areas. Four satellite fuel and lubricant stations will be in the North Complex, three in the Central Complex, and three in the South Complex.

There will be mobile fuel/lubricant trucks in the mobile equipment fleet to deliver fuel and lubricants to equipment such as jumbos, mechanical bolters, and longhole drills.

16.7.5.1 Main Fuel and Lubricant Bays

The main underground fuel and lubricant stations will be centrally located in each complex near the main underground workshops. Each bay will accommodate two vehicles to charge diesel fuel and lubricants to mobile equipment simultaneously.

Each underground main fuel and lubricant station will have two 60 000-liter horizontal, double-walled fuel storage tanks, two fuel and lubricant distribution bays with four lubricant totes, lubricant hose reels, a fuel pump, trench drain with sump, instrumentation and controls, fire water hose reel, fire detection / suppression, and safety items.

Total fuel storage underground is limited to a maximum of two days consumption (approximately 30 000 L per complex).

Ventilation for the main fuel and lubricant bays will be flow-through to a nearby exhaust raise.

Fuel will be transferred from the surface storage tanks on demand in measured batches via a pipeline in the main service decline to storage tanks at the main fuel station near each main workshop. Utility vehicles will transport lubricant containers from surface.

Fire water services will be supplied to the main fuel and lubricant bays from surface via piping routed through the conveyor drift to a local fire hose and sprinkler system. Fire doors will be provided at the entrance and exit to the main fuel and lubricant bays. Fire detection and suppression equipment that interfaces with the complex's emergency alarm system will be included in all the main fuel and lubricant storage bays.

16.7.5.2 Satellite Fuel and Lubricant Bays

Satellite fuel and lubricant bays will be located near satellite workshops and working areas on other levels in each complex. Satellite bays will be smaller than the main fuel and lubricant facilities. Each satellite fuel and lubricant bay will feature four self-contained units (SatStats or similar) to provide storage and dispensing of diesel fuel and lubricants for mobile equipment in the area.

The self-contained units will have 110% spill containment for all fluids stored in them and have integral fire suppression. External fluid containment and fire suppression will not be required at this facility.

16.7.6 Explosives Handling and Distribution

Underground storage magazines for explosives and detonator materials will be centrally located to the mining areas, away from the underground infrastructure and work areas. Three types of magazines, emulsion explosives, packaged explosives, and detonators will be separated by a minimum of 20 m of rock.

All explosives will be stored, stacked, and labeled to facilitate a first-in / first-out inventory control system. Each magazine will be designed with a locking gate. The location of the explosive / detonator facility will be a minimum of 100 m from any work area or blasting area and at least 25 m from the main travel way.

Explosives and detonator materials in specialised containers will be transported by utility vehicles from surface via the main service decline to the underground magazines. Emulsion containers will be unloaded using monorails and all other materials will be unloaded using boom trucks, as required. Special trucks will be used to transport explosive materials from the underground magazines to the workplace. Empty emulsion storage bins will be returned to surface for cleaning and refill, as required.

16.7.7 Mine Services

Mine services will include service water, fire water, potable water, and compressed air.

16.7.7.1 Service Water

Service water will be supplied from the portals through 150-mm diameter piping routed through the conveyor and main service drifts.

The underground service water consumption is based on the amount of water estimated to be used by the mobile equipment, underground facilities, and processes.

Estimated steady state underground service water consumption is summarised in Table 16-70.

Table 16-70: Estimated Underground Service Water Requirements

Facility Description	North Complex Average Flow L/Day	Central Complex Average Flow L/Day	South Complex Average Flow L/Day
Development			
Face Drilling	222 480	133 560	133 560
Primary Ground Support	144 000	96 000	96 000
Mucking	54 000	32 400	37 800
Washing	18 000	10 800	10 800
Production			
Secondary Support	7 680	7 680	7 680
Slot Drilling	115 200	115 200	57 600
Drilling	460 800	345 600	172 800
Mucking	307 200	230 400	115 200
Miscellaneous			
Raiseboring	14 400	14 400	28 800
Infill Drilling	57 600	65 280	28 800
Equipment Cleaning	14 400	14 400	14 400
Miscellaneous Washing	38 400	192 000	153 600
Dust Suppression	160 800	160 800	103 200
Leakage	242 244	238 920	144 036
Total	**1 857 204**	**1 657 440**	**1 104 276**

16.7.7.2 Fire Water

Underground fire-related systems will meet MSHA requirements.

Fire water services will be supplied from surface via 200-mm piping routed from the portal via the main service and conveyor declines. Fire water is used underground for fire-suppression hose reels and sprinkler systems over the full length the conveyors. Fire water systems are also used in the main workshop areas, satellite workshops, and main fuel and lube bays.

Fire detection and suppression equipment will interface with the emergency alarm system and will be included in areas with high risk for fire. These areas include the entire length of the conveyors (above and below the conveyors), main workshops, main fuel and lubricant storage and distribution areas, and satellite workshops.

Fire water hose reels with 30-m hoses will be located every 60-m along the length of the conveyors.

Satellite fuel and lube bay self-contained units will be equipped with integral fire suppression; and do not require fire water.

Electrical mine load centres and substations require clean agent fire suppression, such as FM200.

16.7.7.3 Potable Water

Treated potable water will be supplied from the portal at surface via 50-mm piping routed through the main service decline. Potable water will be provided to underground sinks in latrines, workshops, permanent refuge stations / waiting places, and water bottle filling stations.

Personnel will fill appropriate water containers and carry their own water supply to work areas. Estimated average potable water usage per day is provided in Table 16-71.

Table 16-71: Estimated Average Daily Potable Water Usage by Complex

Facility Description	North Complex Average Flow L/Day	Central Complex Average Flow L/Day	South Complex Average Flow L/Day
Refuge Station Sinks / Bottle Fills	9 792	7 344	7 344
Main Workshop Sinks	9 792	9 792	9 792
Latrine Sinks	61 200	34 272	58 752
Leakage (10%)	8 078	5 141	7 589
Total	**88 862**	**56 549**	**83 477**

16.7.7.4 Compressed Air

Plant compressed air from surface will be supplied to the main workshop areas and permanent refuge stations via 50-mm piping routed through the main service decline. Compressed air from surface will only be provided to the main underground workshops and as a source of emergency breathing air to two permanent refuge stations in each complex. There will not be a mine-wide compressed air reticulation system. The underground compressed air requirements from the surface plant are limited to an average of 1.3 m^3/min for each complex.

The development and production drills will be electric-hydraulic and compressed air requirements will be supplied by on-board compressors or portable compressors. Operating equipment requiring compressed air will have fit-for-purpose onboard air compressors or portable compressors.

Underground satellite workshops will have stationary electric air compressor units.

16.7.8 Personnel and Material Movement

All personnel and materials will be transported to/from the underground working via mobile equipment travelling in the main service decline.

16.7.8.1 Personnel Movement

Personnel carriers will be used to move workers to/from underground workplaces at the start and end of each shift. There will be 30-person bus style carriers and smaller 8-person carriers. Workers that operate equipment that travels to surface at the start and end of each shift will not require bussing.

16.7.8.2 Material Movement

Consumable materials, equipment, and maintenance parts will be delivered to designated underground storage locations using cassette trucks and flatbed boom trucks. Service LHDs that can be equipped with forks will be used to move larger pieces of equipment.

16.7.9 Electrical Infrastructure

The underground electrical distribution system and associated substations will originate at the connection to the surface power distribution system at each of the three portals and include distribution to all underground equipment and related services. Mine power distribution riser diagrammes were prepared for each mine.

16.7.9.1 Power Distribution and Redundancy

Portal Substation

The main surface consumer substation will transform 132 kV utility power to 11 kV for distribution to the three portal locations. 11 kV switchgear located near the portals will provide power distribution to underground loads. This is a main-tie-main configuration with circuit breakers for the incoming and tie section and circuit breakers for surface ventilation, refrigeration, portals, and underground feeders in the line-up.

The feeders from the main consumer switchgear to the portal switches and from the portal switches will feature redundant separated routing for the underground services and sized to provide such service for the major ventilation equipment.

Underground Feeders and Tie-Ins

All major feeds on surface and underground are to be N+1 redundant and routed separately. The feeds are sized for the defined loads. All feeds will have coordinated protection schemes suitable for normal and emergency conditions. Each underground feeder will be overload protected, ground-fault monitored, and electrically protected.

From this switchgear, feeders will be routed down the decline to tap boxes (or switches), mine power centres, or switchgear as needed to service the underground loads for conveyors, dewatering pumps, and fixed facility loads. The 11 kV cable power will be routed to the various loads using 11 kV tap boxes, load break fuse switches for interconnecting different areas, and mine load centres. The mine load centres will transform 11 kV to 525 V.

Cables will be isolated by placement on opposite sides of the main decline or one in each decline. Cables will be suspended from the decline backs with messengers and baskets.

For each mine, the redundant feeders (incomers) from the main substation will be connected to a switchgear line up with a tie breaker so that the mine can be completely fed from one feeder or the other.

Under normal development and production mining / operating conditions, both feeders will be operational with their tie breaker open and effectively sharing the underground load. Table 16-72 shows the total loading for each complex.

Table 16-72: Underground Power Usage

Mine Area	Type Load	Connected kW	Connected kVA	Demand kW	Demand kVA
Central Complex	Dewatering	1 879	2 135	1 553	1 764
	Ventilation	10 233	11 628	8 698	9 884
	Material Handling	6 552	7 445	2 619	2 976
	Development	2 693	3 060	463	526
	Infrastructure	652	740	476	541
	Production	1 143	1 298	310	352
	Central Complex Total	**23 152**	**26 309**	**14 121**	**16 046**
North Complex	Dewatering	3 041	3 455	2 562	2 911
	Ventilation	15 998	18 179	13 175	14 971
	Material Handling	8 509	9 669	3 423	3 890
	Development	3 389	3 851	451	512
	Infrastructure	1 115	1 267	892	1 013
	Production	1 902	2 161	332	377
	North Complex Total	**33 954**	**38 584**	**20 837**	**23 678**
South Complex	Dewatering	1 950	2 216	1 594	1 812
	Ventilation	8 353	9 492	7 100	8 068
	Development	2 021	2 296	350	398
	Infrastructure	835	949	647	736
	Material Handling	3 911	4 530	1 579	1 831
	Production	862	979	211	240
	South Complex Total	**17 933**	**20 465**	**11 484**	**13 087**
Total Load		**75 039**	**85 359**	**46 442**	**52 812**

16.7.9.2 Standby Generation

Key loads for underground mine operation in the event of a complete power outage will be provided by standby generators located on the surface at the main consumer substation. For the total standby loading for each complex and the total standby loading for the mine, refer to Table 16-73.

Table 16-73: Standby Loading

Mine Area	Type Load	Connected kW	Connected kVA	Demand kW	Demand kVA
Central Complex	Ventilation	6 600	7 500	5 610	6 375
North Complex	Ventilation	6 600	7 500	5 610	6 375
South Complex	Ventilation	5 500	6 250	4 675	5 312
Total Load		**18 700**	**21 250**	**15 895**	**18 062**

16.7.10 Communications and Automation

Automation and communication systems are interlinked. Automation requires a data backbone to handle data communication and automation needs. The backbone will provide the basis for all communications and enable 24-hour monitoring and control of the surface and underground ventilation fans, refrigeration plant, conveyor system, fire detection / suppression system, water handling system, electrical substations, fueling facilities, refuge stations, mine communications, and other ancillary installations. The mine communication distribution riser diagrammes were prepared for each complex.

16.7.10.1 Communications

Voice and data communication throughout the mine will be provided via leaky feeder radio, with voice over internet protocol (VOIP) telephone as a secondary system. Underground telephones will be installed at all electrical substations, conveyor drives, loading stations, pump stations, refuge stations, workshops, and waiting places.

An emergency warning system will be provided for one-way mine-wide emergency communication from surface to cap lamps equipped with personnel emergency dispatch system pagers.

To provide data communication for fire systems, a fibre-optic cable backbone will be included from the local underground fire alarm panel to the control room.

16.7.10.2 Leaky Feeder

The primary means of underground mine voice and data communication will be a leaky feeder system. The system will be tied to the surface radio system utilizing handheld radios, fixed location, and vehicle radios. The leaky feeder system will be distributed throughout the entire mine and communication devices will be provided to key personnel requiring communication on a frequent basis.

16.7.10.3 Fibre Optic Cable

The backbone for the data communications system is based on a redundant fibre network. Fibre-optic backbone cables will be routed from surface through each conveyor decline to connect various pieces of mechanical and electrical equipment in each mine zone.

Monitoring and control functions will be connected by fibre network to the local control room, office / portal control rooms, and other data acquisition systems on surface.

The fibre-optic back bone system will carry systems, including CCTV, VOIP telephones, power monitoring, and data collection for mine equipment.

16.7.10.4 Control System

The mining control system for surface and underground daily operations will operate locally in the surface office control centre.

Cameras will be installed at each rock breaker, conveyor transfer point, explosive and primer magazines, and pump station.

Fibre will be installed for monitoring the power system and control for conveyors, pumps, and rock breakers.

16.7.10.5 Equipment / Personnel Tracking

A purpose-built real-time tracking system will be used for all vehicles and personnel. The mine will be divided into zones for the purposes of tracking of equipment and personnel.

Overall Stantec believes the mining methods, mine design, and associated infrastructure are at a level that support a DFS.

17 RECOVERY METHODS

The process design for the Concentrator Plant was developed using the metallurgical testwork and assessments discussed in Section 13, as well as previous studies completed for the Waterberg Project. The criteria for the process design are described below and is aligned to the intended mine design.

Based on the outcome of the preceding 2016 PFS, the selected option for the process design was a phased 600 ktpm Concentrator Plant consisting of two modules. The second concentrator module was designed as duplication of the first module, with some exceptions made for shared infrastructure and water treatment. During the course of the FS, the Concentrator Plant design throughput was restated as 4.8 Mtpa. The 4.8 Mtpa Concentrator Plant will be constructed in a single phase. The concentrate produced by the plant will be transported by road to smelters for further processing and the plant tailings will report either to a backfill plant for use as backfill material, or to the TSF.

The Concentrator Plant is targeted to start milling ore in Month 48 of the project ramping up to full production thereafter as ore availability increases from underground.

17.1 Process Design Criteria

The main elements from the process design criteria are summarised in Table 17-1.

Table 17-1: Process Design Criteria Summary

Criteria		Nominal	Design
Mining			
South and Central Ore Makeup (%)	T-South	9.0%	0 - 100%
	F-South	8.3%	0 - 100%
	F-Central	37.4%	0 - 100%
North Ore Makeup (%)	F-Boundary	16.6%	0 - 100%
	F-North	28.6%	0 - 100%
LOM (Yrs)		43	
Production Summary			
Annual ROM Treatment Rate (tpa)			4 800 000
Expected ROM Moisture Content (% m/m)		5	3 - 6
Material Density (t/m^3)	ROM Blend		2.90
	ROM Bulk Density		1.74
	Rougher Concentrate		2.90
	Cleaner Concentrate		3.20
ROM Size Distribution (mm)	F_{100}	450	500
	F_{80}	265	250 - 280
	F_{50}	100	100 - 115
Target Grind (µm)	Primary Mill P_{80}	212	212
	Secondary Mill P_{80}	75	75
Crushing Circuit Operating Schedule			
Operating Days per Annum (d/a)			365
Operating Hours per Day (h/d)			24
Crushing Circuit Utilisation (%)			65%
Crushing Circuit Annual Run Hours (h/a)			5 660
Crushing Circuit Feed Rate (dtph)			848
Milling Circuit Operating Schedule			
Operating Days per Annum (d/a)			365
Operating Hours per Day (h/d)			24
Milling Circuit Running Time (%)			91 %
Milling Circuit Annual Run Hours (h/a)			8 000
Milling Circuit Feed Rate (dtph)			600
Mill Feed Head Grades			
4E (g/t)	T-South	4.05	2.5 - 5.8
	F-South	3.28	2.5 - 5.0
	F-Central	3.08	2.5 - 5.0
	F-Boundary	3.17	2.5 - 5.0
	F-North	3.19	2.5 - 5.0
	ROM	3.23	2.5 - 5.0
Cu (%)	T-South	0.18	0.05 – 0.26
	F-South	0.04	0.04 – 0.25
	F-Central	0.07	0.05 – 0.25
	F-Boundary	0.09	0.05 – 0.25
	F-North	0.10	0.05 – 0.25
	ROM	0.09	0.05 – 0.25
Ni (%)	T-South	0.09	0.08 – 0.15
	F-South	0.13	0.12 – 0.20
	F-Central	0.18	0.12 – 0.20
	F-Boundary	0.21	0.12 – 0.20
	F-North	0.20	0.12 – 0.20
	ROM	0.18	0.12 – 0.20
Concentrate Grades			
Concentrate (g/t 4E)		80	60 – 100
Mass Pull to Final Products			
Concentrate (% of Mill Feed)		3.19	2.4 – 3.8

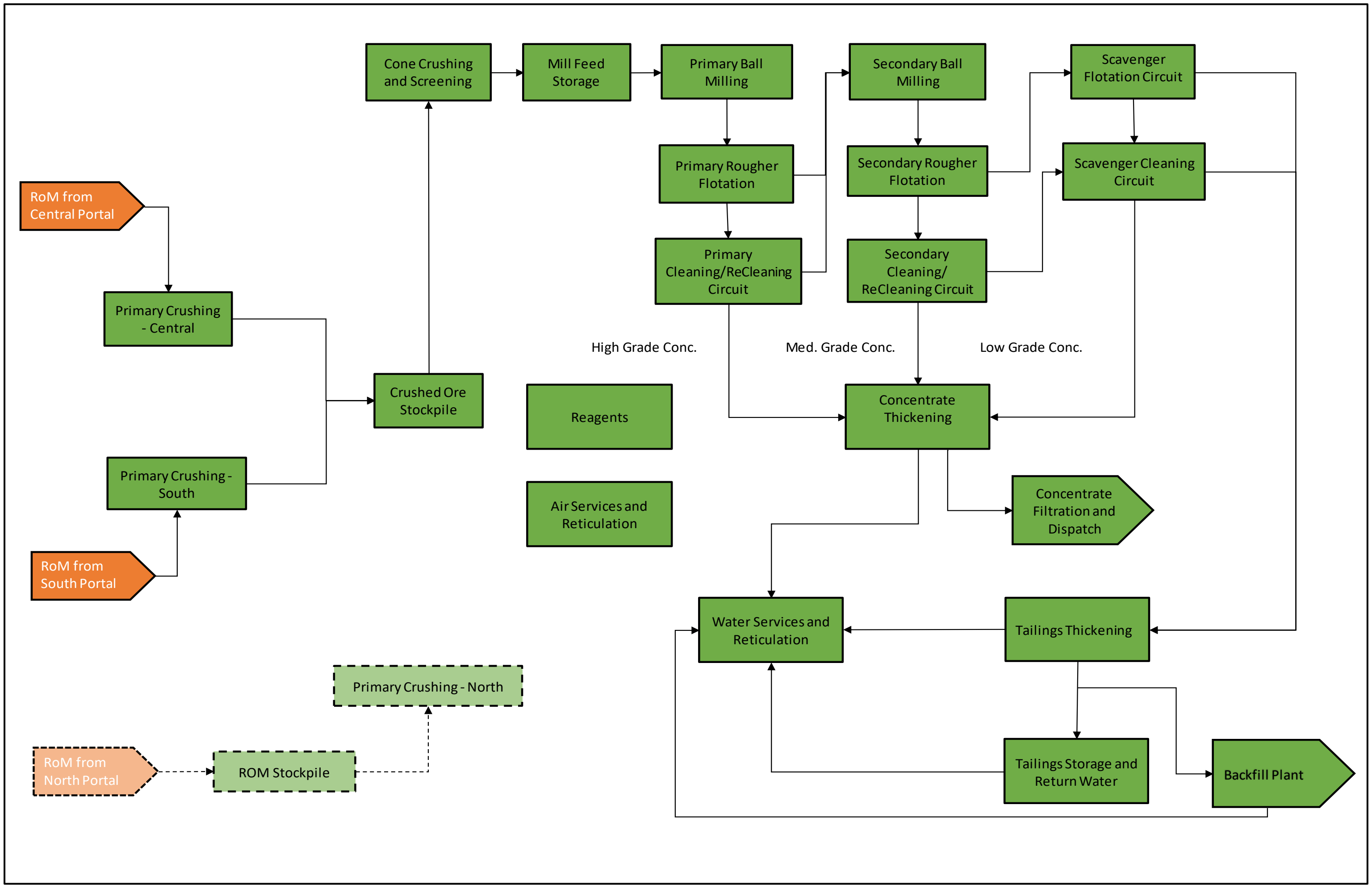
17.2 Process Description

The selected process design makes use the following key unit processes.

- ROM Handling and Storage
- Crushing and Screening
- Milling
- Flotation
- Tailings Disposal
- Concentrate Filtration and Dispatch
- Reagent Makeup and Dosing
- Air and Water Services

Figure 17-1 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 400 ktpm.

Figure 17-1: High-level Block Flow Diagram



17.2.1 Run-of-Mine Ore Storage and Primary Crushing

The ROM from the Central Complex portal, at a top size of 450 mm, will be conveyed to a primary crushing section and crushed to less than 317 mm before being stored on an open stockpile prior to secondary and tertiary crushing. This primary crushing section will include two jaw crushers fed from vibrating grizzly feeders which will allow the undersize material to be conveyed directly to the Central Complex stockpile.

The ROM ore from the Southern portal, at a top-size of 450 mm, will be crushed to less than 317 mm in a single jaw crusher and conveyed overland to the south ROM stockpile (for stockpiling of T-South material), adjacent to the Central Complex stockpile, which will store F-Central material.

The positioning of the Central and South Complexes ROM stockpiles allow for blending of T-South and F-Central material, as required. The ROM will be extracted at a controlled rate from these two stockpiles, in pre-determined ratios and discharged onto the overland conveying system to the secondary and tertiary screening and crushing circuit.

Tramp metal will be removed prior to crushing by means of a tramp metal magnet situated at the conveyor head end. Space provision will be made for future ROM samplers for both portals after primary crushing. Provisions will be made for dust suppression at each of the above primary crushing areas.

Table 17-2 shows the main design parameters for ROM storage and primary crushing.

Table 17-2: Main Design Parameters – Run-of-Mine Storage and Primary Crushing

Description	Value
Central Portal Primary Crushing and Stockpiling	
Central Primary Crushing Circuit Feed (Total) (dtph)	600
Central portal ROM Size Distribution (mm) F_{100}	 450
Crusher Type	Jaw
Number of Crushers	2
Crusher Product Size Distribution (mm) P_{100} P_{80}	 317 169
Central Portal Stockpile (t)	30 000
South Portal Primary Crushing and Stockpiling	
South Portal Primary Crushing Circuit Feed (Total) (dtph)	200
South portal ROM Size Distribution (mm) F_{100}	 450
Crusher Type	Jaw
Number of Crushers	1
Crusher Product Size Distribution (mm) P100 P80	 317 169
South Portal Stockpile (t)	10 000

17.2.2 Screening and Cone Crushing Circuit

The blended primary crushing circuit product from the Central and South Complexes stockpiles will be conveyed to either one of two dual deck, coarse ore screens for classification into three 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 two size fractions.

- The fine ore screens oversize product will report to the tertiary crushing feed conveyor together with the middling 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 minus 13 mm product as feed to the mill feed silo.

Table 17-3 shows the main design parameters for cone crushing and screening.

Table 17-3: Main Design Parameters - Cone Crushing and Screening

Description	Value
Secondary Crusher Type	Cone
Number of Secondary Crushers	2
Coarse Ore Screen Type	Vibrating, Double Deck
Number of Coarse Ore Screens	2
Tertiary Crusher Type	Cone
Number of Tertiary Crushers	2
Fine Ore Screen Type	Vibrating, Double Deck
Number of Fine Ore Screens	2
Crushing Circuit Product Size (mm) P_{100}	13

17.2.3 Mill Feed

The undersize products from the coarse and fine ore screening circuits will report to a dedicated 13 000-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.

Provisions will be made for spillage / scats reloading as well as primary milling grinding media addition to the mill feed belt.

Table 17-4 shows the main design parameters for mill feed storage.

Table 17-4: Main Design Parameters – Mill Feed Storage

Description	Value
Mill Feed Silo Capacity (t)	13 000
Milling Silo Storage (h)	22
Milling Circuit Feed Rate (dtph)	600

17.2.4 Primary Milling and Classification

The primary milling circuit will consist of a 14 MW, 7.21 m × 10.67 m 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.

Table 17-5 shows the main design parameters for the primary milling circuit.

Table 17-5: Main Design Parameters – Primary Milling Circuit

Description	Value
Milling Module Feed Rate (dtph)	600
Mill Feed Size Distribution (mm) F_{100} F_{50}	 13 8
Primary Mill Size (ft) Primary Mill Size (m)	23.65'Ø × 35' EGL 7.21 Ø × 10.67 EGL
Primary mill Size Installed Power (kW)	14 000
Steel Ball Loading (% v/v)	35
Top-up Ball Size (mm)	76
Primary Milling Circuit Product Size P_{80} (µm)	 212

17.2.5 Primary Rougher Flotation

The primary milling classification screen undersize product will gravitate to the 500 m³ primary rougher feed surge tank via a sampling system from where it will be pumped as feed to the primary rougher flotation circuit after the addition of collector.

The primary rougher flotation circuit will consist of a single bank of 5 x 70 m^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 discharge tank at the secondary milling circuit.

Provisions will be made for dosing of frother and depressant to the primary rougher feedbox.

Table 17-6 shows the main design parameters for the primary rougher flotation circuit.

Table 17-6: Main Design Parameters – Primary Rougher Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	600
Flotation Circuit Feed Solids Content (% Solids, w/w)	35
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	5
Flotation Cell Size (m^3)	70
Flotation Bank Residence Time (Minutes)	12.5
Power Input to Cell (kW/m^3)	2.67
Mass Pull to Concentrate (% Mill Feed)	4 - 6

17.2.6 Secondary Milling and Classification

The primary rougher tailings, as well as the primary cleaner tailings, will report to the mill discharge sump from where it will be pumped to the secondary mill classification cyclone.

The secondary milling circuit will consist of a 14 MW, 7.21m Ø × 10.97m EGL, overflow discharge, ball mill operating in reversed closed-circuit configuration with a 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.

Table 17-7 shows the main design parameters for the secondary milling circuit.

Table 17-7: Main Design Parameters – Secondary Milling Circuit

Description	Value
Secondary Mill Size (ft) Secondary Mill Size (m)	23.65'Ø × 37' EGL 7.21 Ø x 10.97 EGL
Secondary Mill Size Installed Power (kW)	14 000
Steel Ball Loading (% v/v)	35
Top-up Ball Size (mm)	32
Primary Milling Circuit Product Size P_{80} (µm)	75

17.2.7 Secondary Rougher Flotation

The secondary milling classification cyclone overflow product will gravitate to the 500 m³ secondary rougher feed surge tank via a sampling system, from where it will be pumped as feed to the secondary rougher flotation circuit, after the addition of collector.

The secondary rougher flotation circuit will consist of a single bank of 7 x 200 m³ 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.

Provisions will be made for dosing of frother and depressant to the secondary rougher feedbox.

Table 17-8 shows the main design parameters for the secondary rougher flotation circuit.

Table 17-8: Main Design Parameters – Secondary Rougher Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	590
Flotation Circuit Feed Solids Content	34
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	7
Flotation Cell Size m³ (m³)	200
Flotation Bank Residence Time (Minutes)	50
Power Input to Cell (kW/m³)	2.33
Mass Pull to Concentrate (% Mill Feed)	4 - 6

17.2.8 Scavenger Flotation

The secondary rougher tailings stream is pumped to the head of the scavenger flotation bank, where collector, depressant, and frother is introduced. The scavenger flotation circuit will consist of a single bank of 8 x 300 m^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.

Provisions will be made for coagulant addition to the scavenger tailings sump upstream of the flocculant dosage at the tailings thickener.

Table 17-9 shows the main design parameters for the scavenger flotation circuit.

Table 17-9: Main Design Parameters – Scavenger Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	559
Flotation Circuit Feed Solids Content (% solids, w/w)	36
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	8
Flotation Cell Size (m^3)	300
Flotation Bank Residence Time (Minutes)	100
Power Input to Cell (kW/m^3)	1.94
Mass Pull to Concentrate (% Mill Feed)	4 - 6

17.2.9 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 4 x 20 m^3 forced air tank cells in series to produce a single concentrate, which will be pumped to the primary re-cleaning circuit.

Table 17-10 shows the main design parameters for the primary cleaner flotation circuit.

The primary re-cleaning circuit will consist of a single bank of 3 x 10 m^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.

Table 17-11 shows the main design parameters for the primary Recleaner flotation circuit.

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 4 x 50 m^3 forced air tank cells in series to produce a single concentrate, which will be pumped to the secondary re-cleaning circuit for upgrading.

Table 17-12 shows the main design parameters for the secondary cleaner flotation circuit.

The secondary re-cleaning circuit will consist of a single bank of 3 x 20 m^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.

Table 17-10: Main Design Parameters – Primary Cleaner Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	37
Flotation Circuit Feed Solids Content (% Solids, w/w)	16
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	4
Flotation Cell Size (m^3)	20
Flotation Bank Residence Time (Minutes)	18
Power Input to Cell (kW/m^3)	3.28
Mass Pull to Concentrate (% Mill Feed)	3

Table 17-11: Main Design Parameters – Primary Recleaner Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	18
Flotation Circuit Feed Solids Content (% Solids, w/w)	17
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	3
Flotation Cell Size (m^3)	10
Flotation Bank Residence Time (Minutes)	10
Power Input to Cell (kW/m^3)	4.52
Mass Pull to Concentrate (% Mill Feed)	1 - 2

Table 17-12: Main Design Parameters – Secondary Cleaner Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	50
Flotation Circuit Feed Solids Content (% Solids, w/w)	15
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	4
Flotation Cell Size (m^3)	50
Flotation Bank Residence Time (Minutes)	25
Power Input to Cell (kW/m^3)	3.02
Mass Pull to Concentrate (% Mill Feed)	2.5

Table 17-13 shows the main design parameters for the secondary recleaner flotation circuit.

Table 17-13: Main Design Parameters – Secondary Recleaner Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	15
Flotation Circuit Feed Solids Content (% Solids, w/w)	14
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	3
Flotation Cell Size (m^3)	20
Flotation Bank Residence Time (Minutes)	25
Power Input to Cell (kW/m^3)	3.28
Mass Pull to Concentrate (% Mill Feed)	0.5 - 1

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 6 x 130 m^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.

Table 17-14 shows the main design parameters for the scavenger cleaner flotation circuit. Provisions will be made for the reagent addition to each of the various cleaning circuits.

Table 17-14: Main Design Parameters – Scavenger Cleaner Flotation Circuit

Description	Value
Flotation Circuit Feed Rate (dtph)	65
Flotation Circuit Feed Solids Content (% Solids, w/w)	14
Flotation Cell Type	Tank Cell, Forced Air Aeration
Number of Flotation Banks	1
Number of Flotation Cells per Bank	6
Flotation Cell Size (m^3)	130
Flotation Bank Residence Time (Minutes)	75
Power Input to Cell (kW/m^3)	3.10
Mass Pull to Concentrate (% Mill Feed)	1 – 1.5

17.2.10 Concentrate Thickening

The three concentrate products (high, medium, and low-grade) from flotation will report to the 33 m diameter high-rate concentrate thickener. Each concentrate product will be sampled individually prior to thickening. Provisions will be made for trash removal via linear screen installations prior to thickening.

The thickened concentrate at 55% solids w/w will be pumped to either one of two concentrate filter feed surge tanks, while the concentrate thickener overflow streams will be re-used for spray water in the flotation circuit. Any excess overflow from the concentrate thickeners will report to the process water circuit for re-use as process water.

Provisions will be made for coagulant addition prior to flocculant addition for each thickener installation.

Table 17-15 shows the main design parameters for the concentrate thickening circuit.

Table 17-15: Main Design Parameters – Concentrate Thickening Circuit

Description	Value
Thickener Circuit Feed Rate (dtph)	23
Thickener Type	High Rate
Thickener Size (m Diameter)	33
Thickener Underflow Density (% w/w)	55%
Unit Area Thickening Rate ($t/h/m^2$)	0.03

17.2.11 Concentrate Filtration

The thickened concentrate will report to the either of two concentrate filter feed surge tanks from where it will be pumped to either of the two 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.

Provisions will be made for final sampling of the final product prior to dispatch. Table 17-16 shows the main design parameters for the concentrate filtration.

Table 17-16: Main Design Parameters – Concentrate Filtration

Description	Value
Filter Type	Horizontal Plate, Pressure Filter
Number of Filters	2
Selected Unit	Larox PF96/120 M60 1 45
Filtration Rate (kg/h/m^2)	120 - 150
Filter Cake Moisture Content (% Moisture, w/w)	12

17.2.12 Tailings Handling and Disposal

The flotation circuit tailings will be pumped to a 47 m diameter H rate thickener 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 either the TSF or the backfill plant. These pipelines will be supplied from a common sump feeding dedicated duty / standby pumping installations consisting of four centrifugal pumps in series (per train) to the TSF and two centrifugal pumps in series (per train) to the backfill plant.

The tailings thickener overflow products will gravitate to the process water circuit.

Flushing water to clear the lines for the transition between the two pipelines is included in the design.

Table 17-17 shows the main design parameters for the tailings disposal.

Table 17-17: Main Design Parameters – Tailings Disposal

Description	Value
Thickener Circuit Feed Rate (dtph)	577
Thickener Type	High Rate
Thickener Size (m Diameter)	47
Thickener Underflow Density (% w/w)	55 - 60
Unit Area Thickening Rate (t/h/m^2)	0.40

17.2.13 Water Services

Raw water makeup will be provided from a balancing dam supplied with water sourced from groundwater services provided from surface drill holes. The raw water 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 centralised services to the processing plant potable water storage tanks from where it will be pumped to the potable water distribution system.

Plant process water will be stored in a process water tank from where it will be distributed to the concentrator via a dedicated pumping system. The process water tank will be fed by the TSF return water, the backfill plant return water, the tailings thickener overflow, excess concentrate thickener overflow product, as well as plant runoff from the dedicated plant pollution control dam. Provision will be made to route the backfill plant return water to the tailings thickener if required, based on water quality.

A clean water system will supply gland service water to the required areas as well as reagent makeup water. A duty / standby pumping system will be provided for the concentrator. The gland service water to the 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 runoff collection.

17.2.14 Air Services

Low-pressure blower air to the flotation circuit will be supplied by a system of multistage, centrifugal air blowers. A common standby unit will be installed.

Plant and instrument air will be supplied by rotary screw compressors. Most 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.

The drying air to each of the 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.

17.2.15 Consumables

17.2.15.1 Collector

The collector will be delivered via bulk road tankers and offloaded into two 30 m^3 storage tanks. The collector will be pumped to a makeup tank where it will be diluted prior to dosing. Dosing to the required points will be done via a dedicated ring main system with a control valve and flowmeter at the dosing points.

Table 17-18 shows the main design parameters for the collector.

Table 17-18: Main Design Parameters – Collector

Description	Value
Reagent Type	Sodium Isobutyl Xanthate (SIBX)
Delivery Form	Liquid
Mixture Strength, as Delivered (% w/v)	40
Mixture Strength, as Dosed (% w/w)	10
Reagent Consumption (g/t)	115
Reagent Consumption (tpm as Delivered)	130

17.2.15.2 Depressant

A carboxy methyl cellulose depressant will be delivered via bulk road tankers and offloaded pneumatically into a 50-t silo. The depressant will be diluted to 1.0% w/v strength prior to dosing. Dosing to the required points will be done via a dedicated ring main system with a control valve and flowmeter at the dosing points.

Table 17-19 shows the main design parameters for the depressant.

Table 17-19: Main Design Parameters – Depressant

Description	Value
Reagent Type	Sendep 30E
Delivery Form	Solid
Mixture Strength, as Delivered (% w/v)	92
Mixture Strength, as Dosed (% w/w)	1
Reagent Consumption (g/t)	416
Reagent Consumption (tpm as Delivered)	181

17.2.15.3 Frother

The frother will be delivered in via bulk road tankers and offloaded into a single 30 m^3 storage tank. The frother will be pumped to a makeup tank where it will be diluted to prior to dosing. Dosing to the required points are done via a dedicated ring main system with a control valve and flowmeter at the dosing points.

Table 17-20 shows the main design parameters for the frother.

Table 17-20: Main Design Parameters – Frother

Description	Value
Reagent Type	Senfroth 522
Delivery Form	Liquid
Mixture Strength, as Delivered (% w/v)	97
Mixture Strength, as Dosed (% w/w)	25
Reagent Consumption (g/t)	175
Reagent Consumption (tpm as Delivered)	72

17.2.15.4 Coagulant

Coagulant will the delivered as liquid in 1-t intermediate bulk containers and made-up to the correct dosing strength. A dedicated dosing pump system will distribute the diluted coagulant to the thickeners.

Table 17-21 shows the main design parameters for the coagulant.

Table 17-21: Main Design Parameters – Coagulant

Description	Value
Reagent Type	Magnafloc 1597
Delivery Form	Liquid
Mixture Strength, as Delivered (% w/v)	100
Mixture Strength, as Dosed (% w/w)	1
Reagent Consumption (g/t)	200
Reagent Consumption (tpm as Delivered)	80

17.2.15.5 Flocculent

Flocculent granules will be delivered in 1-t bags and manually loaded into a single bulk bag bin receiver. The flocculent granules will be transferred to a wetting system via a screw feeder. The flocculent will be made up to 0.2% w/v strength prior to dosing. Dosing to the required points will be done via dedicated dosing pumps to each dosing point.

Table 17-22 shows the main design parameters for the flocculant.

Table 17-22: Main Design Parameters – Flocculent

Description	Value
Reagent Type	Magnafloc 919
Delivery Form	Solid
Mixture Strength, as Delivered (% w/v)	100
Mixture Strength, as Dosed (% w/w)	0.2
Reagent Consumption (g/t)	25 g/t Conc Thickener Feed 25 g/t Tails Thickener Feed
Reagent Consumption (tpm as Delivered)	10

17.2.15.6 Grinding Media

High chrome steel balls will be used as grinding media in the primary and secondary mills.

Table 17-23 shows the main design parameters for the grinding media.

Table 17-23: Main Design Parameters – Grinding Media

Description	Value
Reagent Type	High Chrome Steel
Primary Mill Grinding Media Size (mm)	76
Primary Mill Grinding Media Consumption (g/t)	300
Primary Mill Grinding Media Consumption (tpm)	120
Secondary Mill Grinding Media Size (mm)	32
Secondary Mill Grinding Media Consumption (g/t)	770
Secondary Mill Grinding Media Consumption (tpm)	308

17.3 Sampling and Ancillaries

17.3.1 Process Plant Sampling and Laboratory

Provisions will be made in the Concentrator Plant design for including a sample preparation laboratory to prepare daily samples prior to dispatch to the centralised assay laboratory complex. Required analysis will be conducted on each of the samples at the assay laboratory. The centralised assay laboratory will cater for mining grade-control, processing plant control, concentrate dispatch, and environmental samples (refer to Section 18 for more detail). Provisions will be made in the design for the necessary sampling points and equipment as per Table 17-24.

The primary rougher flotation feed, final tailings, and final concentrate product assays will be used to compile the plant metallurgical balance.

The labour plan used to estimate the process plant operating costs includes operational staff on each shift to cater for sample collection and preparation.

Table 17-24: Process Plant Sampling Summary

Sample Description	Sample Type & Frequency	Analysis Required	Sampling Equipment Provided
Mill Feed Sample	Process Control 1 Composite / Shift	Particle Size Distribution 3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Manual Belt Cut of <13 mm Material
Primary Rougher Feed	Metal Accounting 1 Composite / Shift	6E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Primary Rougher Tails	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Secondary Rougher Feed	Process Control 1 Composite / Shift	Particle Size Distribution 3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Secondary Rougher Tailings	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Scavenger Tailings	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Scavenger Cleaner Tailings	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Primary Cleaner Tails	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Secondary Cleaner Tailings	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in conjunction with a secondary rotary vezin type sampler
Final Tailings	Metal Accounting 1 Composite / Shift	6E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Cross-cut Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Primary Recleaner Concentrate	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Timed Vezin Type Sampler
Secondary Recleaner Concentrate	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Timed Vezin Type Sampler
Scavenger Cleaner Concentrate	Process Control 1 Composite / Shift	3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Timed Vezin Type Sampler
Thickened Concentrate	Process Control 1 Composite / Shift	Particle Size Distribution 3E Fire-assay Cu, Ni, Fe, MgO, SiO_2 via ICP S via Leco	Primary Rotary Vezin Type Sampler in Conjunction with a Secondary Rotary Vezin Type Sampler
Final Concentrate Product	Metal Accounting 1 Composite / Truck	6E Fire-assay Cu, Ni, Fe, Mg, Si via ICP S via Leco	Auger Type Sampler
Reagent Makeup Checks	Process Control 1 Sample / Batch	Various	Manual Sampling Required

17.3.2 Process Control

Provisions will be made in the design for a fully integrated control system to allow for control of the concentrator from a centralised control room.

The concentrator will be equipped with a high level of automation to allow for remote control of major processing equipment by a power-line communication (PLC) and supervisory control and data acquisition (SCADA) system. An integrated SCADA / human-machine interface control system will be used for interfacing with the operational staff.

An appropriate level of access and control will be programmed into the SCADA system during the implementation phase to ensure that only authorized personnel will be able to make changes to the SCADA parameters.

The milling circuit will include automatic feed rate and dilution water control, as well as density and pressure control on the classification circuits. Within the flotation circuit, the slurry feed rate, blower air addition, and cell froth level will be controlled. All reagents will be dosed automatically based on process setpoints linked to the mill feed rate. Human interfacing will be minimised in the reagents make up systems.

The labour 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 analysers were included in the process plant design; however, 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-t trucks will require approximately 15 shipment transfers per day.

17.4 Utility Consumption

17.4.1 Power

Refer to Table 17-25 for a summary of the envisaged power consumption of the concentrator Plant. The power consumption is calculated as 71.0 kW/t ore milled.

Table 17-25: Processing Plant Power Consumption

Item	Installed Power	Absorbed Power
	MW	MW
Concentrator Plant	60.0	41.0
Shared Infrastructure	3.3	1.6
Total	**63.3**	**42.6**

17.4.2 Water

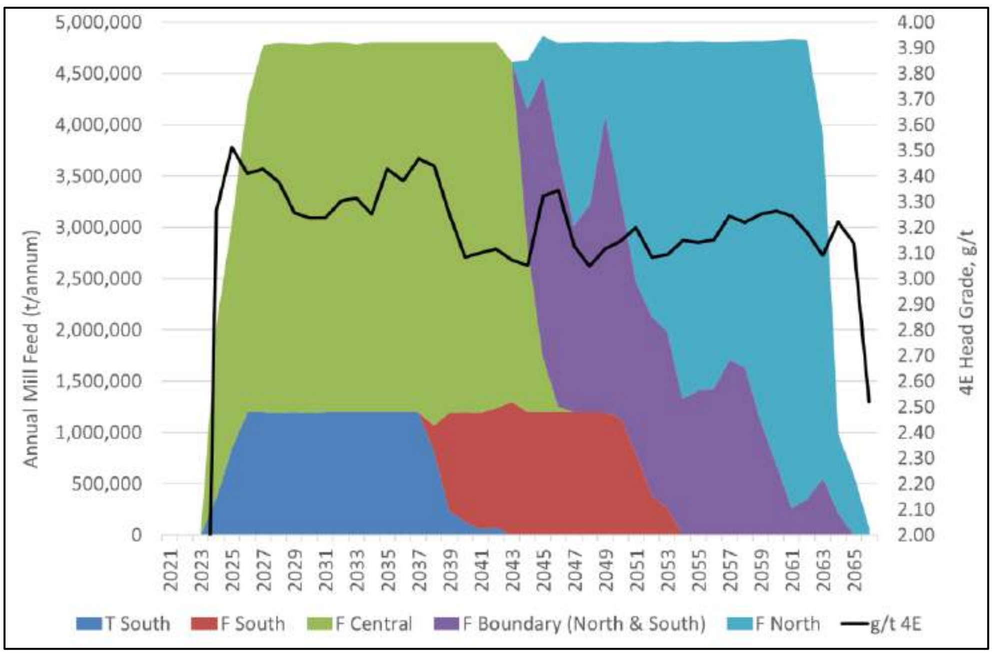
The processing plant raw water requirement is based on the concentrator circuit mass balance and considers the predicted water return from the TSF.

The raw water makeup requirement to the Concentrator Plant is calculated as 264 m^3/h or 0.44 t /t ore milled.

17.5 Production Profile

The milling profile is based on the mining production and is aimed at reducing stockpiling requirements as far as possible while generating revenue as early as possible. Figure 17-2 presents a summary of the annual mill feed profile and associated 4E head grade.

Figure 17-2: Annual Mill Feed Profile Summary



Refer to Figure 17-3 for a summary of the associated annual concentrate tonnage produced and associated mass pulls. The annual base metal and 4E metal production are presented in Figure 17-4.

Figure 17-3: Annual Concentrate Production Summary

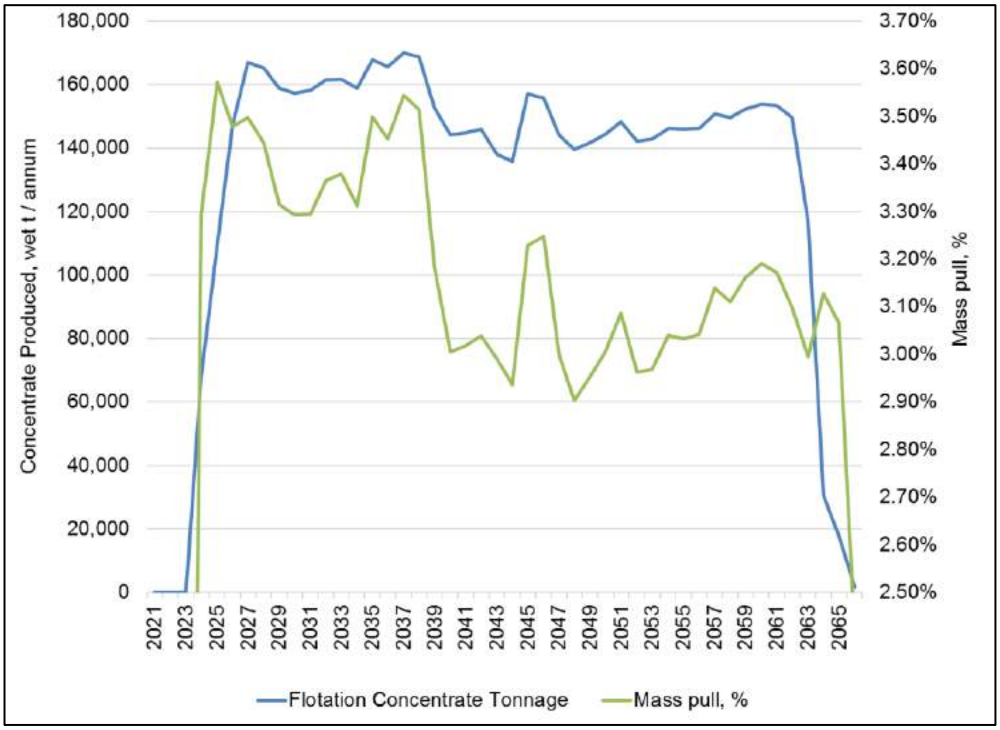
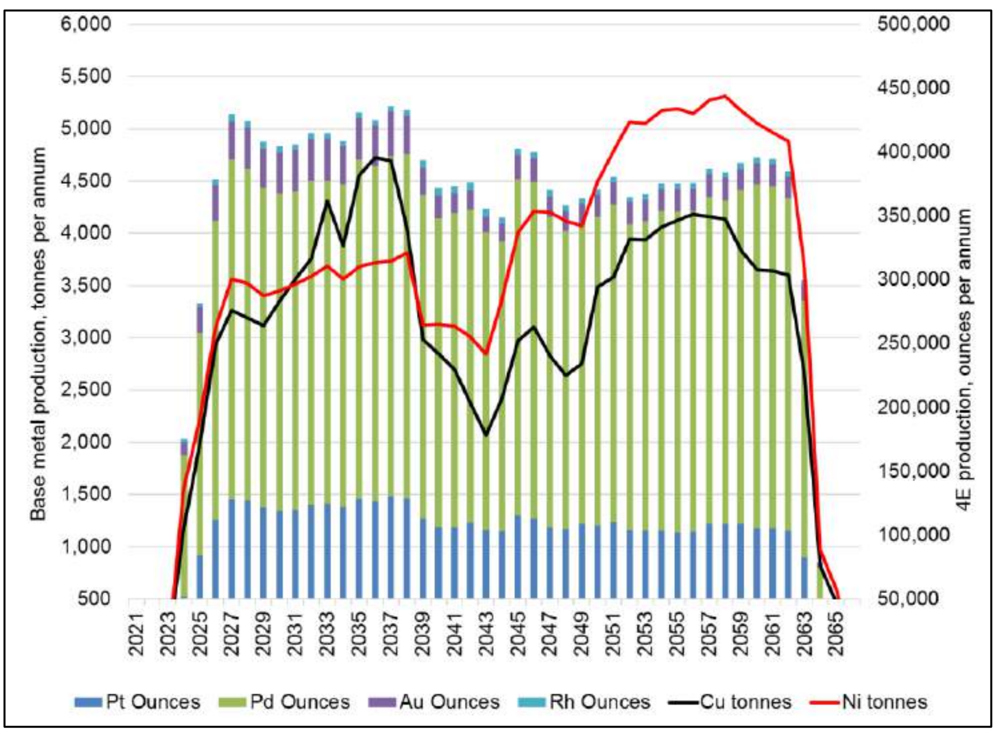


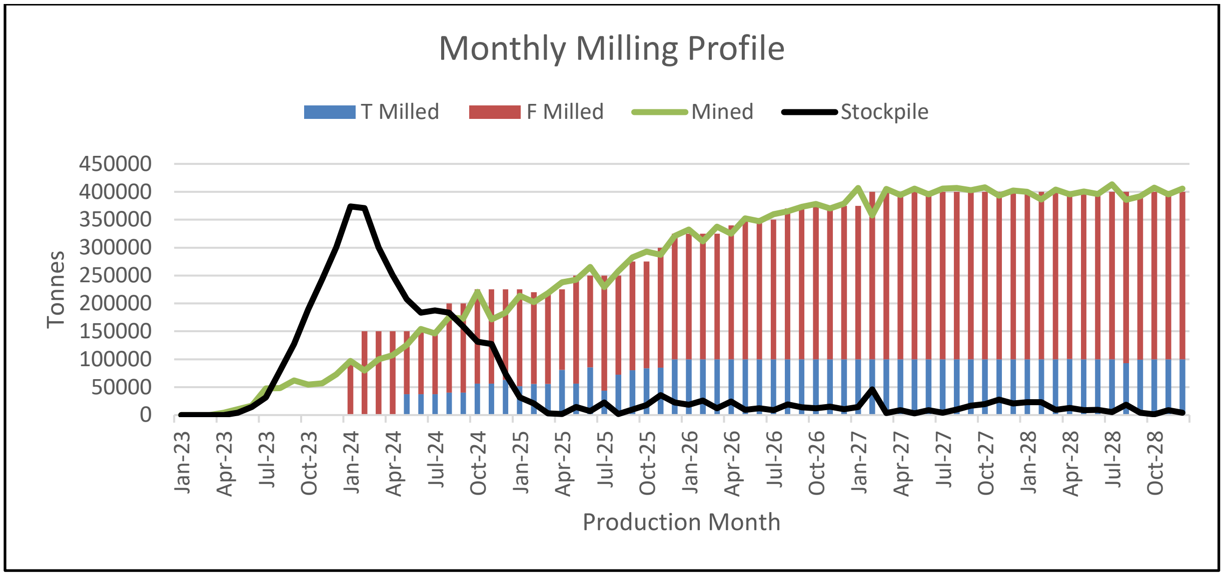
Figure 17-4: Annual Metal Production Summary



Production Ramp Up

The mining operation will deliver first ore in April 2023 and processing will commence in January 2024. A total of 375 kt or ore will be delivered to the stockpile during this period. Figure 17-5 shows the concentrator production ramp-up.

Figure 17-5: Concentrator Production Ramp-up



The monthly treatment rate is ramped up during the first year to consume the stockpile but also to maintain concentrate production for dispatch to the smelter, allowing optimization of the flotation plant to maximise recovery at the desired concentrate grade.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

18.1.1 Overview

The Waterberg Project infrastructure includes both regional, local, and site-specific infrastructure. The existing site infrastructure is basic and intended to support the agricultural activities currently being undertaken in the region plus support for the geological drilling programme that was undertaken during the last number of years for the Waterberg Project.

The existing national road network provides access to the boundary of the site; however, the last 34 km of road to the mine is unpaved.

The existing electrical grid is near capacity and the 22 kV system is inadequate for mine operations; however, it could be used for construction purposes if sufficiently strengthened.

The Waterberg Project will need to construct the following supporting regional infrastructure.

- Bulk Water Supply based on Extracting Water from Drill Holes
- 132 kV Electrical Supply from the ESKOM Power Utility
- Access Roads to and from the Mine
- Telecommunication and Internet Services

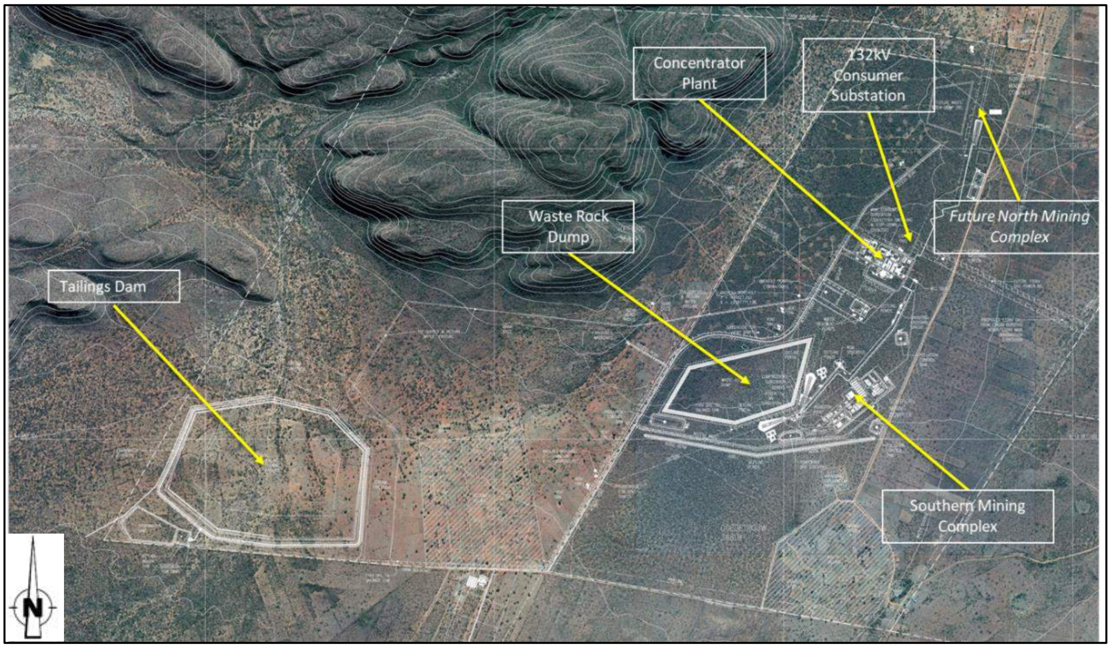
The local surface infrastructure will be constructed on the mine site (Ketting and Goedetrouw farms) and is grouped together in three main areas: South Complex, Concentrator Plant, and the TSF.

A provision was made for the future development of a North Complex on the northern end of the Goedtrouw farm with some ventilation fans being placed on the Early Dawn property to the north. The location of these areas on the property are indicated in Figure 18-1.

Following is the additional infrastructure that will be constructed on surface.

- 132 kV Consumer Substation
- ESKOM Switching Yard
- 11 kV Electric Reticulation
- Ventilation Fans (multiple)
- Backfill Plant
- Explosives Magazine
- Explosive Destruction Area
- Potable Water treatment plants
- Sewerage Treatment plant

Figure 18-1: Site Layout



Following are facilities common to these major areas.

- Substations
- Offices
- Access Control
- Pollution Control Dams
- Service Water Reticulation and Storage Tanks
- Potable Water Reticulation and Storage Tanks
- Waste Handling Facilities
- Fire Water Reticulation, Storage Tanks, and Pumps

First-aid stations are provided in all the major areas of the mine.

18.1.2 South Complex

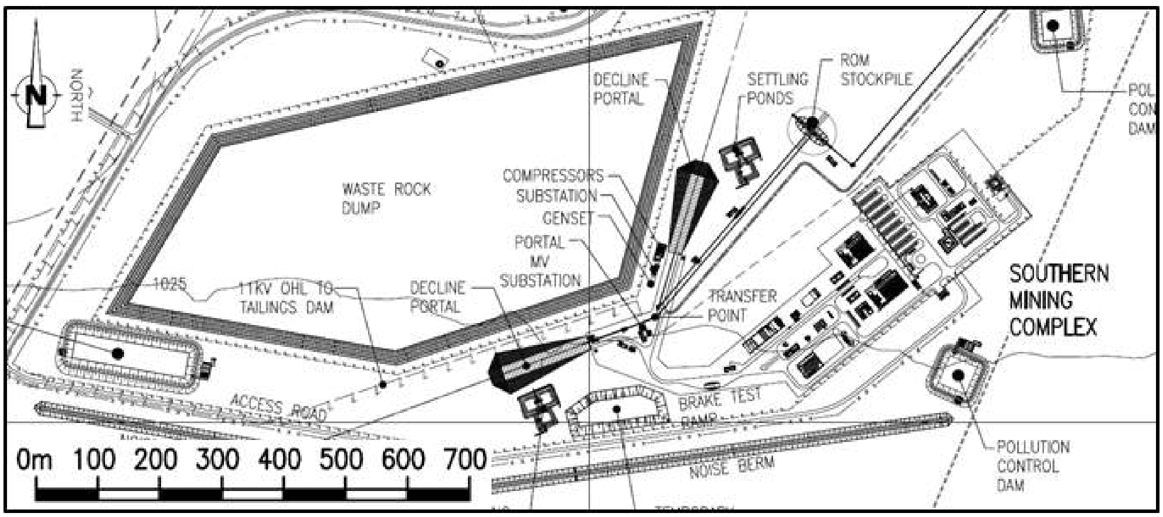
Built in close proximity to the underground access portals, facilities included in the South Complex to support mining operations are listed below.

- Change House
- Lamp Room
- Control Room
- First-aid Station
- Compressor House
- Emulsion Storage Silos

- Trackless Mobile Machinery Workshop
- Wash Bay for Underground Vehicles
- Brake Test Ramps
- Temporary Ore Stockpile Facility
- Waste Rock Dump
- Central Workshop
- General Store
- Bulk Fuel Storage and Dispensing
- Compressor House

The layout of the South Complex shown in Figure 18-2.

Figure 18-2: Surface Layout: South Complex

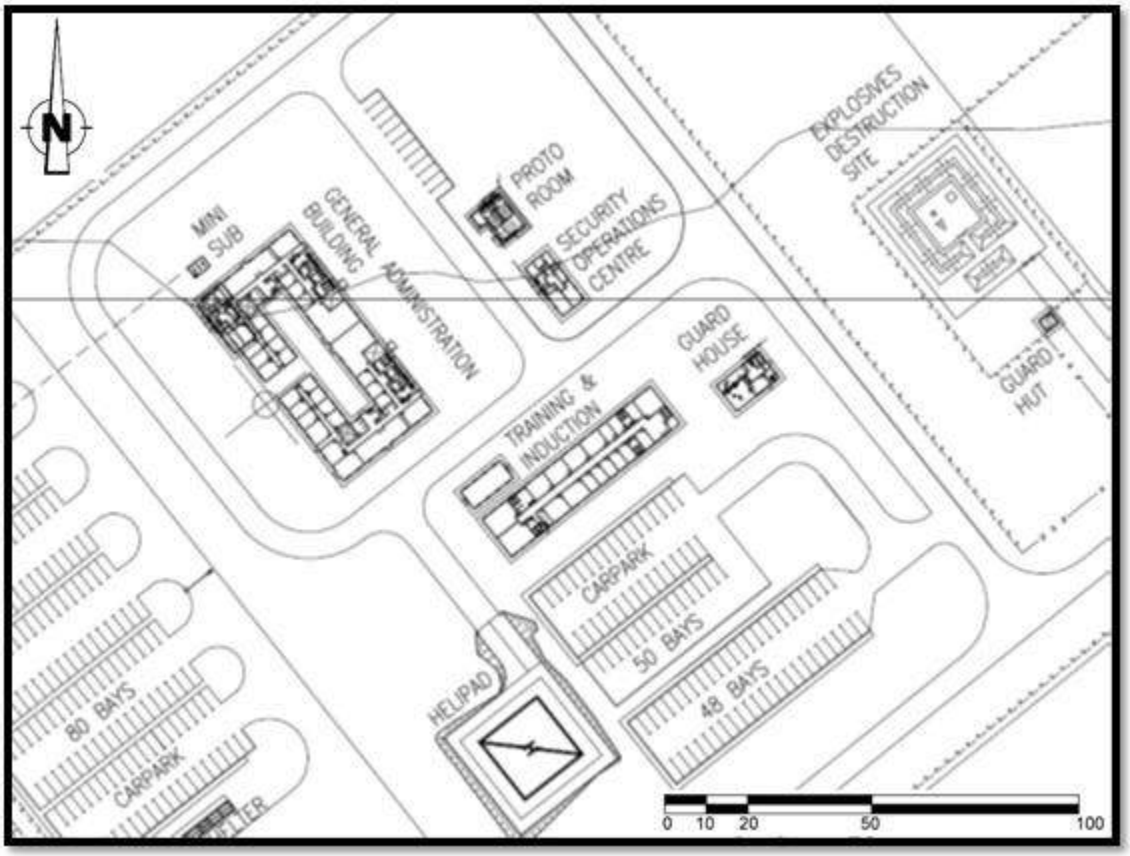


18.1.3 Shared Services

Adjoining the South Complex is the shared services area with the following structures as shown in Figure 18-3.

- Administrative Offices
- Training / Induction Centre
- Proto Room
- Security Operations Centre
- Guardhouse and Access Control to Area
- Helipad
- Explosives Destruction Site

Figure 18-3: Surface Layout: Shared Services



Mine operations are further supported by the following facilities.

- Potable Water Treatment Plant and Storage Tanks
- Sewerage Treatment Plant
- Bulk Water Distribution and Buffer Dam (Balancing Dam)
- Water Diversion Canals using Repurposed Topsoils

18.1.4 Plant Infrastructure

The Concentrator Plant operation is supported by the following facilities.

- Assay Laboratory (Section 18.8)
- Workshop
- Store
- Change House
- Administrative Office

- Control Room
- Weighbridge

18.1.5 Ventilation

Mine ventilation fans and BAC plants are positioned on surface as described above in Section 16.

18.2 Site Layout and Access Roads

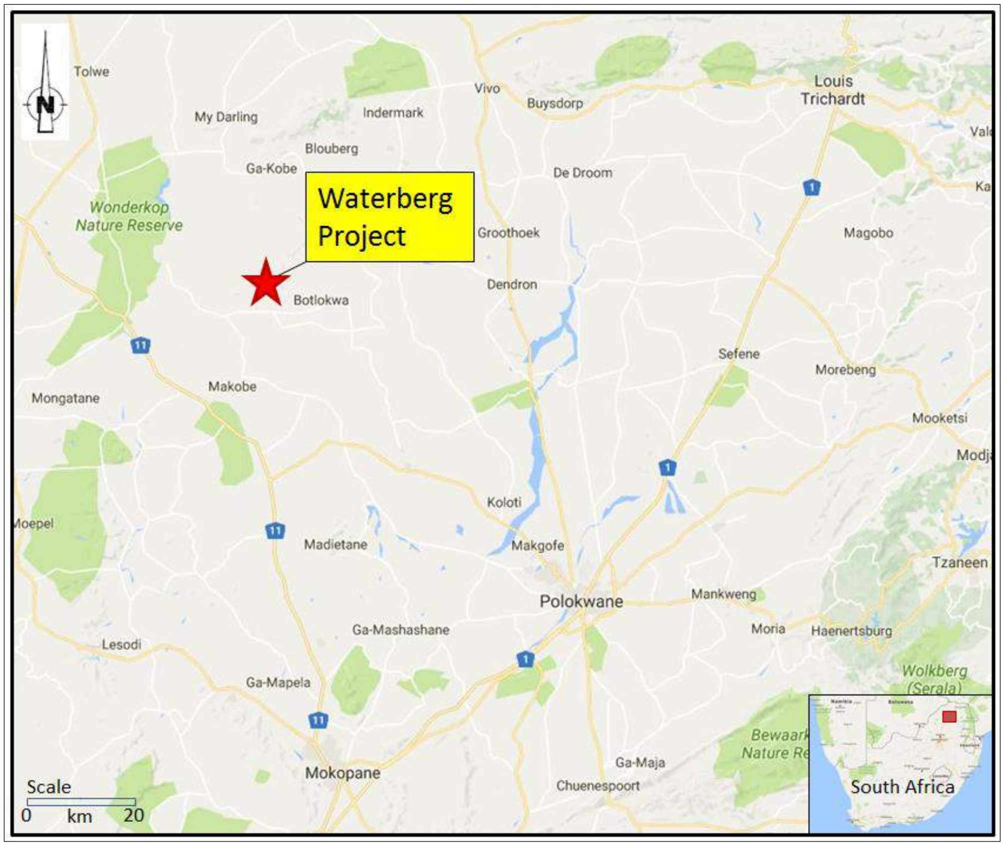
The Waterberg Project is situated some 34 km from the N11 national road that links Mokopane with the Groblers Bridge border post to Botswana. Access to the Waterberg Project area is from the existing national road network. The towns of Mokopane (112 km) and Polokwane (94 km) are the closest major urban centres and can be reached on existing roads however the 34 km of roads local to the mine are unpaved. The Waterberg Project location is shown in Figure 18-4.

Although the bulk of the roads surrounding the site are provincial roads under the jurisdiction of the Roads Agency Limpopo, some of the minor roads are the responsibility of either the Capricorn District Municipality or Waterberg District Municipality.

The Waterberg Project intends to upgrade and surface the 34 km road from the mine to the village of Steilloop by creating a paved road link, which will connect the mine to the N11 national road. Further upgrading of 9.4 km of unpaved road to the town of Bochum will also be completed to facilitate the transport of staff that might be based there.

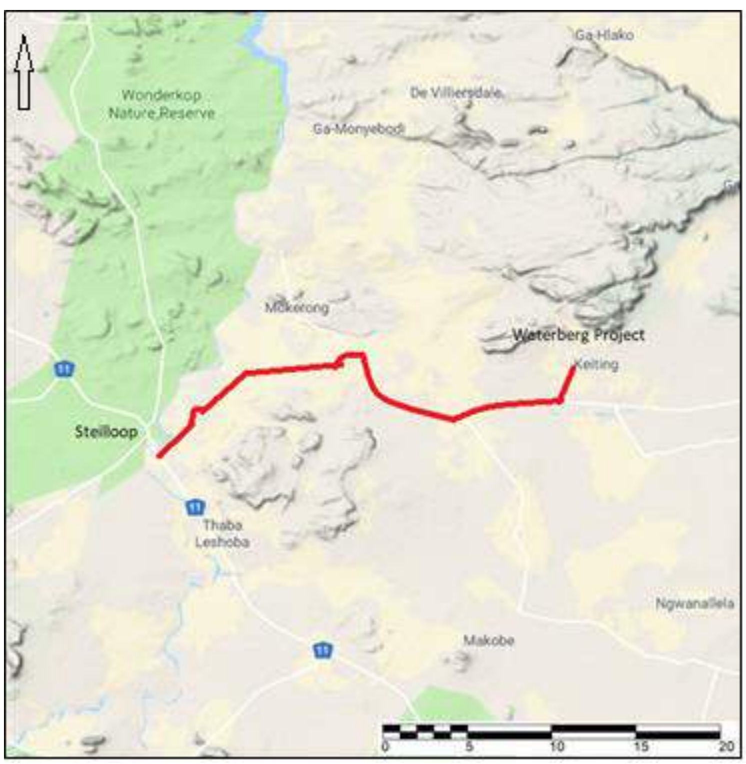
A geotechnical investigation was completed for the selected route and a typical road cross section was designed. The road design is also aligned with current provincial road standards. The selected route to the N11 is indicated by the red line in Figure 18-5.

Figure 18-4: Location of Waterberg Project



Source: Google Maps

Figure 18-5: Route from Project Site to N11



18.3 Water General Infrastructure

South Africa is a country of relatively low rainfall especially in the Limpopo Province where the Waterberg Project is situated. The project is located in the Mogalakwena River Catchment area, which is semi-arid with a mean average rainfall of less than 400 mm per annum and runoff is limited.

Previous studies investigated various sources of water and the use of groundwater from drill holes was selected as the go-forward case and is included in this study.

Water security for mining and concentrate production activities was identified as a risk. To mitigate this, an extensive hydrological investigation was undertaken as part of the study. This study modelled the infiltration of fissure water into the mine and pump tests on the identified drill holes were conducted. The impact on the surrounding communities was also modelled to understand the impact of the operations on the supply of water to the surrounding area.

A site-wide water balance was developed to understand the water requirements of the project and mining operation and take account of the impact on the communities. The water balance considers all operational activities related to mining, the Concentrator Plant, TSF, and the backfill plant. Water treatment plants are included in the design to meet the potable water requirements of the operation.

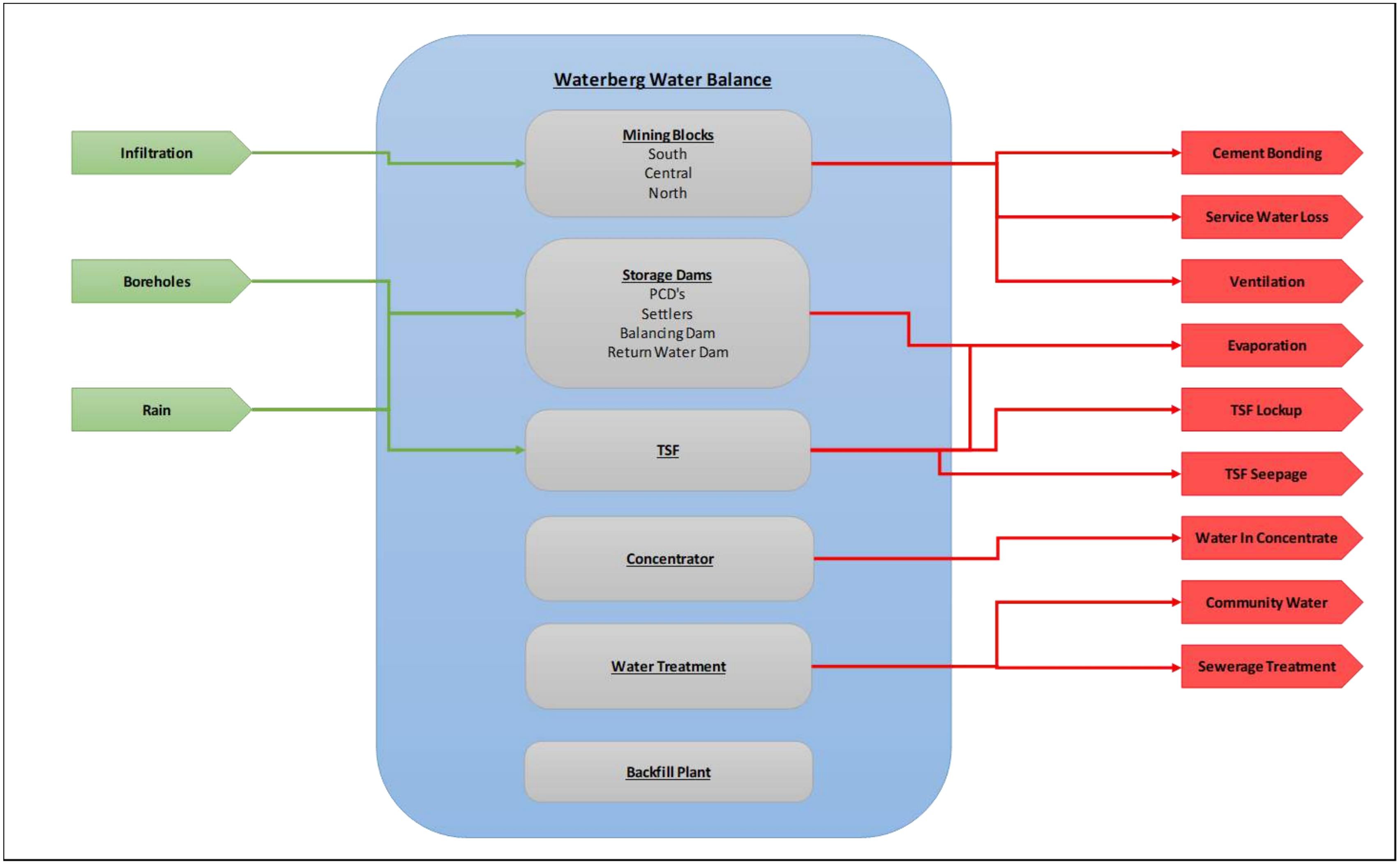
The estimated water demand for the Waterberg Project is calculated to be 6.2 Ml/d.

18.3.1 Water Balance

A simplified view of the overall water balance indicates that the mine will have access to three sources of water, including infiltration of fissure water as a result of mining activities, intermittent rainfall in catchment areas, and water supplied by drill holes in the vicinity of the mine. Figure 18-6 shows an overview of the water balance.

All processes within the balance interact with one another via intermediate recycle streams, which are not shown in Figure 18-6, but are accounted for in the detailed water balance.

Figure 18-6: Simplified Waterberg Water Balance



Water consumption is related to the following items.

- Water losses in the mining blocks through the ventilation services and service water consumption.
- Water contained in the mining blocks through the cement bonding of the backfill.
- Evaporation on water storage dams such as settlers, pollution control dams, return water dam (RWD), stormwater dam (SWD), and the TSF.
- Water losses due to tailings storage.
- Water entrainment in the concentrate production.
- Sewage treatment.
- Supply of water to the surrounding communities.

The water supply for the mine from the drill holes was determined, excluding the positive effect of rainfall. Due to the variable nature associated with rainfall and the arid region, various rainfall scenarios were investigated and during operation captured runoff will be utilized as process water. The outcomes of the scenario showing water demand and supply without rainfall is indicated in Table 18-1.

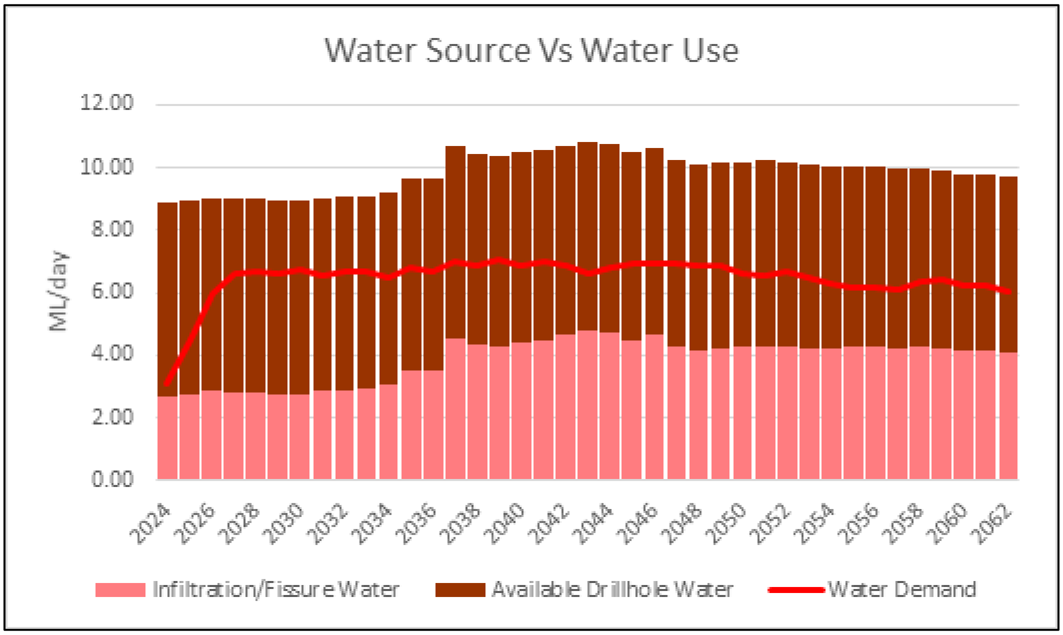
Table 18-1: Water Source versus Water Use for No Rain Scenario

No Rain Scenario			
Water Inflows		Water Outflows	
Water Source	ML/day	Water Use	ML/day
Infiltration / Fissure Water	3.9	Evaporation	0.6
Available Drill Holes	6.2	Underground Losses	
Rain	0.0	Cement Bonding	2.6
		Service Water Losses	0.1
		Ventilation Losses	0.7
		TSF	1.8
		TSF Seepage	0.2
		Water in Concentrate	0.1
		Community Water Supply	0.3
		Sewerage Treatment	0.0
Total	**10.1**	**Total**	**6.5**
Surplus	**3.6**		

It was concluded that the water supplied by the drill holes and the infiltration is sufficient to support the necessary mining and processing operations over the LOM. The capture and use of rainfall water will allow for a reduce demand on the groundwater during the rainy season.

The water requirements and usage were also modelled over the LOM and results are demonstrated graphically in Figure 18-7.

Figure 18-7: Water Source versus Water Use for No Rain Scenario over Life of Mine



18.3.2 Bulk Water Sources

Groundwater abstraction schemes in the area were also developed mainly for domestic consumption at the rural villages. Potable water can be abstracted from the drill holes, some of the other drill holes have low-quality water due to high salts and nitrates in some areas rendering it unsuitable for human consumption. However, it is suitable for use as plant process water (subject to final confirmation with future testwork) and can be treated on site to provide potable water for the project. The project is also able to return, following treatment of the water, high-quality potable water to the surrounding communities affected by the mine dewatering activities.

Following investigations to ascertain the security of the water supply, Table 18-2 indicates the drill holes identified for the Waterberg Project and tested to determine the sustainable yield of the well field.

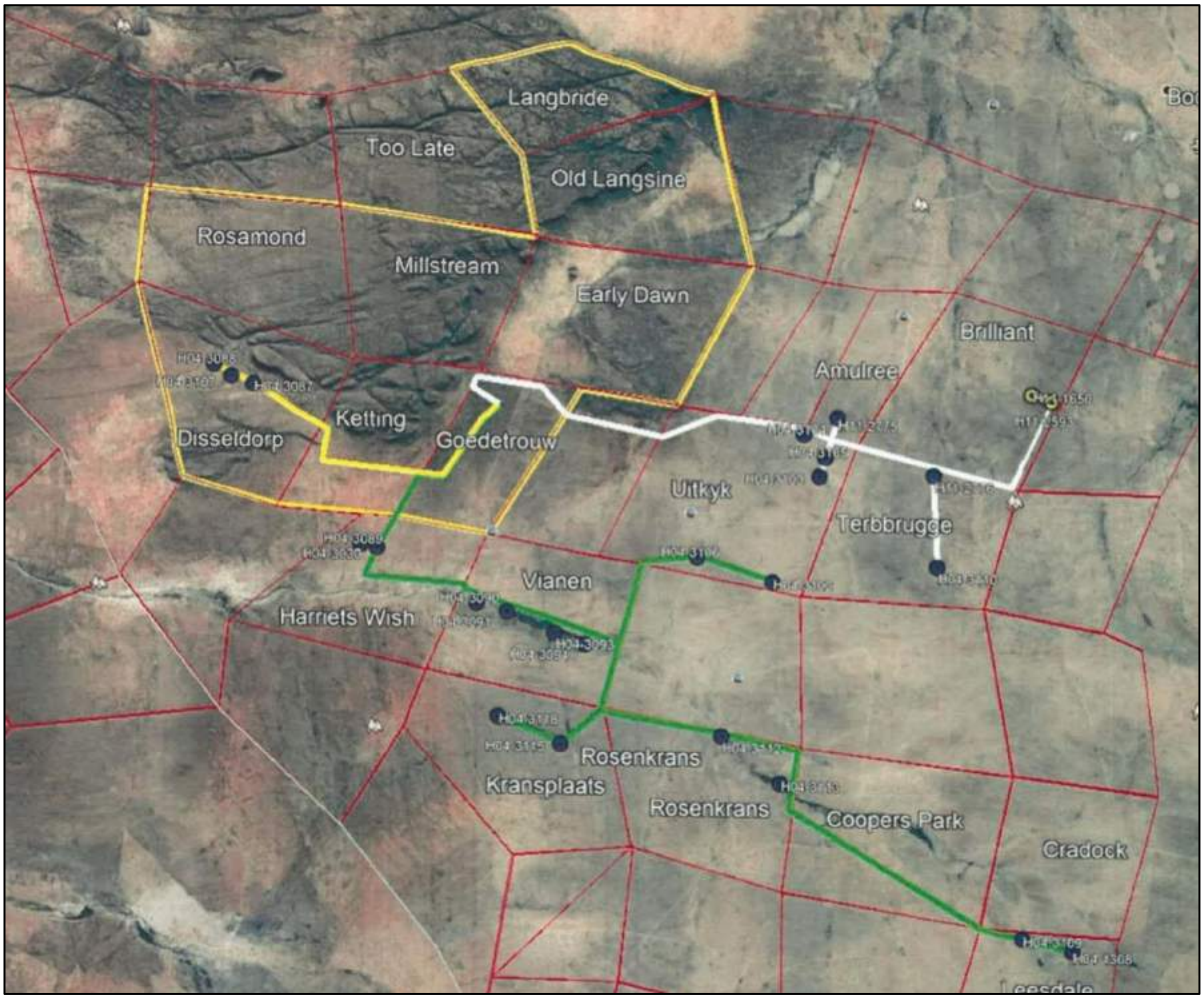
Water from the drill holes will be pumped into surface storage tanks. From these tanks, water will be pumped via buried pipelines of varying sizes to the project site balancing dam from where water will be distributed to various areas as required.

Figure 18-8 indicates the location of drill holes and storage tanks.

Table 18-2: Proposed Production Drill Holes

Drill Hole No.	Longitude	Latitude	Farm	Depth Drilled (m)	Model Recommended Pumping Rate (m^3/day)	Model Equipped Pumping Rate 12 Hours Per Day (l/s)	Recommended Use
H04-3087	28.83792	-23.35960	Disseldorp	189	300	6.5	Production Drill Hole
H04-308	28.82558	-23.35423	Disseldorp	108	200	2.8	Production Drill Hole
H04-3089	28.87165	-23.40543	Vianen	83	350	7.8	Production Drill Hole
H04-3030	28.87675	-23.40622	Vianen	138	150	2.5	Production Drill Hole
H04-3090	28.90841	-23.42173	Vianen	80	300	4.0	Production Drill Hole
H04-3091	28.91775	-23.42436	Vianen	36	400	7.0	Production Drill Hole
H04-3093	28.93264	-23.43073	Vianen	80	200	3.1	Production Drill Hole
H04-3094	28.94199	-23.43340	Vianen	61	350	6.0	Production Drill Hole
H11-1650	29.08128	-23.36005	Briliant	64	350	6.0	Production Drill Hole
H11-2593	29.08748	-23.36184	Briliant	84	400	15	Production Drill Hole
H04-3102	29.0008	-23.41485	Uitkyk	79	200	3.0	Production Drill Hole
H04-3103	29.01525	-23.38426	Uitkyk	109	200	3.2	Production Drill Hole
H04-3104	29.01029	-23.3723	Uitkyk	90	200	3.0	Production Drill Hole
H04-3105	29.01704	-23.37881	Uitkyk	84	300	6.0	Production Drill Hole
H04-3106	28.97719	-23.40799	Uitkyk	84	250	4.7	Production Drill Hole
H11-2776	29.05096	-23.38354	Terbrugge	70	300	5.0	Production Drill Hole
H11-2775	29.02499	-23.36119	Amulree	67	350	7.1	Production Drill Hole
H04-3110	29.05212	-23.40994	Terbrugge	79	200	3.4	Production Drill Hole
H04-3112	28.98516	-23.45945	Rosenkrans	92	250	4.0	Production Drill Hole
H04-3113	29.00362	-23.47268	Rosenkran	65	300	5.1	Production Drill Hole
H04-3115	28.93472	-23.46212	Kransplaats	72	150	2.2	Production Drill Hole
H04-3108	29.09511	-23.51944	Leesdale	85	200	3.0	Production Drill Hole
H04-3109	29.07953	-23.52174	Leesdale	100	300	5.0	Production Drill Hole
Total					**6 200**		

Figure 18-8: Drill Hole and Storage Tank Location



Source: Google Earth

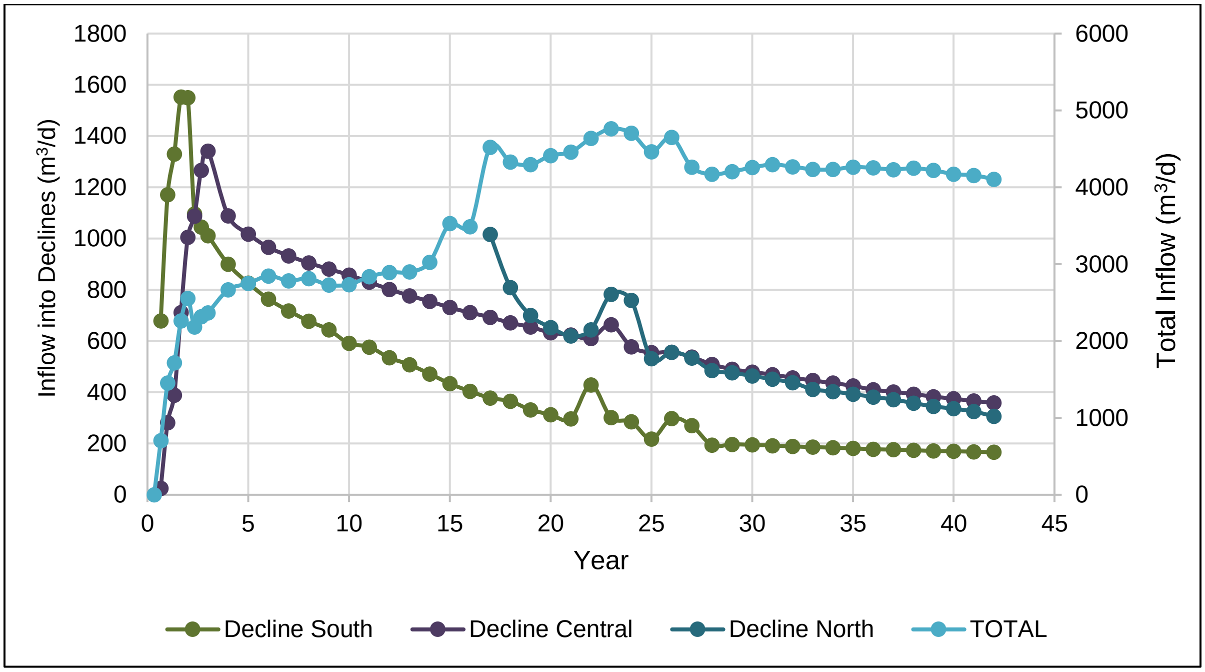
Fissure Water

Figure 18-9 indicates the expected infiltration of fissure water into underground workings over the time period of the Waterberg Project.

Inflows are 2 800 m^3/d when only the Southern and Central Complexes are in operation. When the North Complex comes online, it is assumed that the Central and South Complexes will continue to be dewatered. Inflows will increase to 4 700 m^3/d, before declining to 4 200 m^3/d. Total inflows over LOM amount to 60 729 430 m^3.

Water from underground, including fissure water and reclaimed mining service water will be pumped to surface and stored in settling dams on surface. Water from the settling dams will be returned underground as service water, with surplus water being sent to the process plant. Solids accumulating in the settling dams and filters will be removed mechanically and processed as required to allow storage on the TSF.

Figure 18-9: Expected Infiltration of Ground Water into Underground Workings



18.3.3 Stormwater and Containment

Stormwater falling within the mine footprint, TSF, and plants will be collected in pollution control dams and fed into the process plant to be used as process water. Stormwater falling outside of these areas is directed away from the mining area using cut-off berms to divert runoff upstream of the mining area for discharge downstream of the mining area.

Water captured within the mining operations is designed to remain within the closed-loop water balance internal to the mining area. This includes rainwater falling within the mining footprint, spillage water, or fissure water.

The internal water management measures include the following features.

- Runoff drains local to the process plant and portal areas to collect all polluted water.
- Site-wide runoff concrete-lined drains to collect polluted water from other areas in the mining area and deposit it to the HDPE-lined pollution control dams.
- Dedicated contaminated water drainage systems around the stockpile and waste rock dump areas.
- Silt traps to collect water from runoff drains and remove silt before discharge into the pollution control dam.

- Four HDPE-lined pollution control dams are included in the project: waste rock dump, South Complex, plant, and future north portal pollution control dams.
- These dams are sized based on the defined catchment areas, to contain a 1:50 year flood event with a duration of one day, and 800 mm freeboard.

In accordance with the overall water balance, water will be pumped out from the pollution control dams back into the water circuit for industrial use. All contaminated and stormwater systems are estimated in accordance with the expected requirements of the EMP and integrated WUL.

18.4 Electrical General Infrastructure

18.4.1 Predicted Electrical Load

The Waterberg Project will receive power from the National grid at 132 kV. The design described in this Technical Report includes for the distribution of this power from the 11 kV consumer substation to the point of use.

The predicted electrical load based on connected load and the use of power factor correction resulted in the steady-state electrical load as described in Table 18-3.

Table 18-3: Predicted Electrical Load

	Installed Power (MW)	Run Power (MW)	Estimated Maximum Demand (MVA)
South Complex	19.0	9.9	10.8
Central Complex	26.2	12.4	13.0
Bulk-air Cooling Plants	13.8	11.8	14.5
Plant	61.4	39.1	43.0
Backfill Plant	6.8	4.3	4.6
Total	**112.6**	**77. 5**	**85.9**

The main Consumer Substation is divided in four bus sections, each with an incomer from a 40 MVA transformer. A power factor correction bank will be installed for each bus section.

The future requirement for the North Portal is estimated at 23 MVA for full production similar to the Central Complex it replaces. It is noted that the Central and North Complexes are not planned to be in production at the same time and the North Portal loads are not included in Table 18-3.

18.4.2 Bulk Electricity Supply

The bulk electricity supply to the Waterberg Project will comprise a permanent grid-based supply by Eskom from its 132 kV electrical network. The Waterberg Project will be supplied at 132 kV, and the mine-owned infrastructure will include a 132/11 kV step-down substation.

Eskom confirmed the availability of a supply capacity of 140 MVA. The sustainable capacity of the proposed Eskom bulk supply infrastructure is 108 MVA at 132 kV, which compares to the planned mine electrical load at 11 kV of 86 MVA as detailed in Section 18.4.1 and provides a capacity reserve margin of over 20%.

It is forecasted that the reserve margin will be temporarily reduced during the period when the Central Complex mining activities are ramping down and the North Complex mining activities are ramping up.

The bulk electricity supply infrastructure will include the following items.

- Eskom-owned infrastructure.
  - One new 132 kV line feeder bay in the existing Eskom Burotho 400/132 kV Main Transmission Station.
  - A new Eskom 132 kV switching station to be located on or near the Goedetrouw property.
  - One 132 kV overhead line approximately 74 km in length, from the existing Eskom Burotho 400/132 kV Main Transmission Station to the new Eskom 132 kV switching station to be located on or near the Goedetrouw property.
- Mine-owned infrastructure.
  - A new 132/11 kV step-down substation comprising 4 x 40 MVA 132/11 kV step-down transformers.
  - A short 132 kV overhead line approximately 3.5 km in length from the 132/11 kV step-down substation to the Eskom 132 kV switching station.

Figure 18-10 shows the planned route for the 74 km long 132 kV overhead line from the Burotho 400/132 kV Main Transmission Station to the new Mine 132/11 kV substation, via the Eskom 132 kV switching station.

Eskom confirmed the availability of the required capacity from its 132 kV network at Burotho Main Transmission Station. Eskom also prescribed the proposed 132 kV network expansion plan, although the capacity of these expansions is currently being revised downwards to account for the lower notified demand load of 90 MVA at 132 kV (compared to previous PFS estimates).

The development of the Eskom 132 kV infrastructure is being done in terms of a self-build process with most of the development work completed under Eskom supervision.

Environmental impact studies are currently underway to obtain EAs for some of the above-mentioned 132 kV infrastructure, and to amend portions for which EAs were previously issued. Negotiations with landowners to acquire servitudes for the 132 kV overhead lines are in advanced stages.

Figure 18-10: Bulk 132 kV Infrastructure and 132 kV Overhead Line Route

Source: Nel, H.H. 2019. TDx Power. *Internal planning report*.

18.4.3 Temporary Electricity Supply

The permanent bulk electricity infrastructure is scheduled for construction during the mine construction process, with a completion date prior to beginning mine commissioning. The electricity supply to the mine during its construction period will be from a temporary supply to be sourced from the local Eskom 22 kV network.

Planning processes are underway for the development of this temporary electricity supply with a capacity of approximately 3 to 5 MVA.

18.4.4 Emergency Power Generation

Eight 2.5 MVA light fuel emergency power generator sets will be installed and connected to the 11 kV consumer substation. Emergency power is reticulated to downstream substations at 11 kV using the same infrastructure as the normal supply. The ventilation fans will be eight 1.5 MW units. The 20 MVA emergency supply will be sufficient to supply the ventilation fans and other loads as distributed to the MCCs.

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 off loading and storage facilities will be provided at the South Complex and are adequately sized to cater for three days of operation during steady state. The storage comprises two 80 000 m^3 tanks for diesel. These facilities will be suitably isolated from nearby infrastructure and adequately ventilated. The storage containers will be self-bunded to prevent environmental contamination. Fire protection is provided as described below.

18.5.2 Fire Protection Facilities

The fire-water system (supply, storage, and distribution) will be designed in accordance with A.S.I.B – 11th Edition Codes of practice, SANS 62 & 719 – Galvanised and or Painted Carbon Steel piping and fittings and NFPA 15 – Standard for Water Spray Fixed Systems for Fire Protection.

A surface fire water 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.

The Concentrator Plant and surface conveyor fire mains will be carbon steel and painted as required, the buried pipelines will be constructed of HDPE. The underground workings will be supplied from the main at the 200 mm flanged connection at the entrance to the respective portals. 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 the fire protection systems installed.

The surface fire main system will be dedicated solely for the purposes of firefighting and no other off-take will be allowed to be taken off the fire main system for process or domestic water purposes.

Three fire pump stations will be constructed one as part the South Complex, one as part of the plant area and one as part of the future North Portal. The pump stations will store potable water equipment with a pressure maintenance (Jockey) pump, primary electrical pump, and secondary diesel pump if power is not available.

Fire hydrants and hose reels will be connected to the ring mains. Every hydrant will have a designated fire hose cabinet containing two 30 m length of hoses with an instantaneous coupling and a nozzle. Portable fire extinguishers will be positioned at each building as required.

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 Key Surface Buildings

18.5.3.1 Compressor House

Compressor houses will be constructed for both the mining and plant areas and will house the compressors that provide the compressed air requirements for both plant and underground operations independently. The mining compressor house located close to the portals will serve both South and Central decline shafts and related underground workings, the North Portal will be supplied with air from a similar structure. The plant compressor house is located in close proximity to the reagents and concentrate handling areas.

18.5.3.2 Change Houses

Two change-house buildings are proposed for the Waterberg Project located at the South Complex and at the plant. They are sized for 940 (mining) and 172 (process) personnel, respectively. The buildings include laundry facilities, pre-shift briefing area, stores, and administrative offices. Provisions are made for both male and female workers.

A third change house at the North Complex to cater for 746 personnel will be constructed when that portal is developed.

18.5.3.3 Administration Offices

Office buildings will be provided at the plant and mining complexes. There is also a general administration building, which will serve as the centre for the mine administration functions and house the various department heads.

18.5.3.4 Control Room - South Complex, Future North Portal, and Plant

Control rooms are located within the South Complex and plant. A control room is planned for the future North Complex. Each building comprises an engineering, PLC, storeroom, manned control work area, kitchen, and ablution facility. The internal environment is airlocked and will be mechanically ventilated to suit equipment specifications.

18.5.3.5 Access Control

Guard houses are located at the entrance to each of the mining complexes, plant, and shared services area. The guard house building's function is to ensure access control for the mining complexes and processing plant facility for personnel and vehicular flow in and out of these areas.

The guard house is comprised of a covered on-off shift personnel thoroughfare area with double full height turnstiles in each direction for staff traffic. A male / female search room is included for inspections and an enquiry room. Boomed vehicular access control is located externally on the roadside. Time and attendance for surface employees is logged at point.

18.5.3.6 Lamp Room (South and North Complex)

The lamp room is located close to the change house. The building will include lamp racks to cater to 1 050 underground lamps and rescue packs, personal protective equipment issue and storerooms, lamp repairs and store area, kitchen, office, and a room for gas detection instruments and testing. Time and attendance for underground employees is logged at point on collection and return of the equipment.

18.5.3.7 Trackless Mobile Machinery Workshop

The workshop is an open drive-through workshop and was sized in terms of number of workshop bays required for the fleet. The trackless mobile machinery workshop will be utilized during the mine development phase. Once the underground workshops are constructed, repairs to most mobile machinery will mainly be done underground. The workshop includes seven repairs bays and four refueling bays.

18.5.3.8 General Stores

Two stores buildings are planned for the Waterberg Project. The larger one will be located at the South Complex and the second at the Concentrator Plant.

18.5.3.9 Plant Workshop

The main plant workshop area is 550 m^2 and a 5-t overhead traveling crane with a provision for an additional crane.

18.5.3.10 Combined Surface Workshop

The combined surface workshop is a large facility catering for plant and vehicle repairs, including the mining fleet, and services wash bays. The structure is located on the mining complex and has a footprint of 2 688 m^2. All major repairs will take place at this workshop once the mine and plant are in full operation.

18.5.3.11 Explosives Accessories Magazine

The accessories magazine is a building structure utilized for storing detonator cartridges and related consumables. The magazine has a minimum safe radius of 400 m off the mine access road and any other existing or planned surface building.

18.5.3.12 Temporary Construction Camp

A temporary construction camp will be established on Harriet's Wish, the property just south of Ketting where the mine is located. Specific areas are allowed for contractors of different trades.

During the initial stages of construction, the earthworks contractor will expand the area around the current geological camp to provide space for camp expansion. The camp facilities will be increased over time to accommodate the full contingent of the construction personnel.

The temporary contractor accommodation facilities will be used to house the mining and construction contractors only during the construction period.

18.5.3.13 Communications

The surface communications will consist of the following networks.

- Telecommunications Network
- Information Technology Network
- Control Network
- Radio Network

The telecommunications network will consist of an external supplier providing a data link to site. Telephone communication will be via microwave data connection substituted with cellular. The on-site data network will be fibre interconnecting all infrastructure and underground operations.

The backbone for the control system communication is based on a redundant ring fibre-optic network. This communication will be used to support all critical control system data communication requirements for the Waterberg Project. A radio network will also be available for site communications and operational staff.

18.6 Waste Facility

18.6.1 General Waste Facilities

Operational and domestic waste handling facilities will be provided at the South Complex and Concentrator Plant.

The following waste handling areas will be provided.

- Salvage yards at the plant and South Complex for salvaging mine equipment and scrap.
- General domestic waste produced by the offices will be separated into organics and recyclables (metals, plastics, glass, paper, etc.).
- Hazardous storage areas for hazardous waste materials such as batteries, lubricants, and other hazardous substances. Hazardous materials will be disposed of by an accredited service provider.
- Medical waste disposal facilities will be provided for the South Complex and plant first-aid stations. Medical waste will be disposed of by an accredited service provider.
- A waste skip area outside the plant and mining security area will be provided from where the waste contractor will collect the waste.

18.6.2 Waste Rock Dump

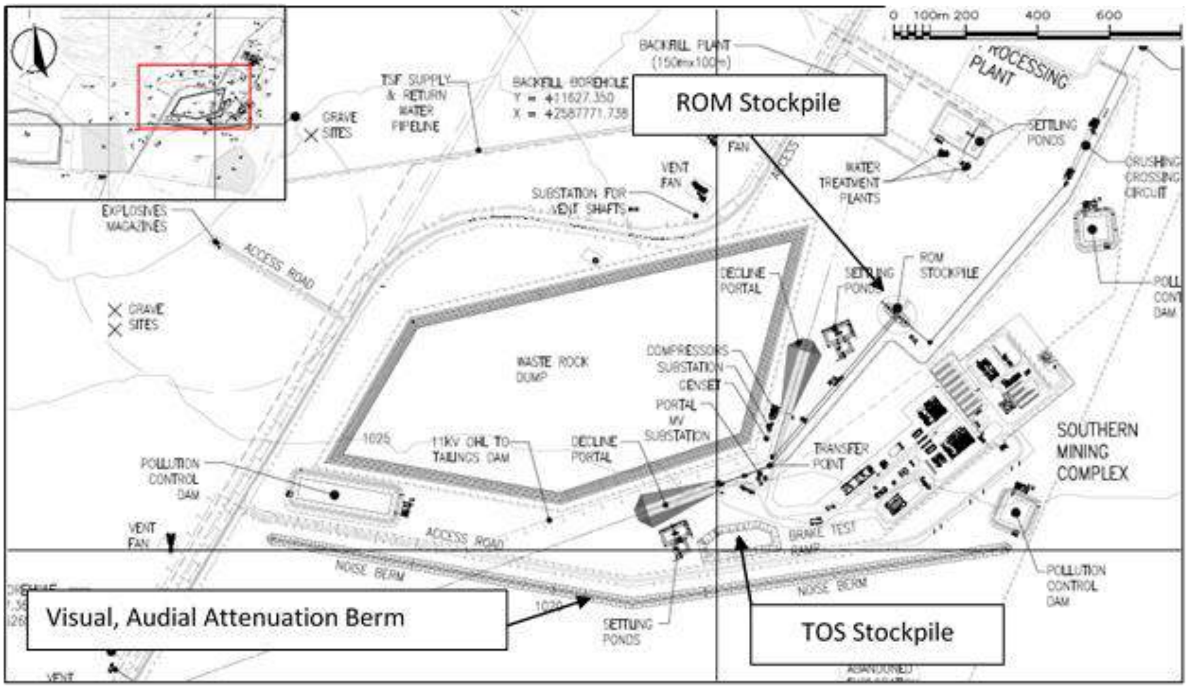
Waste rock resulting from the underground development activities will be placed on a single waste rock dump near to the South Complex until the North Complex is in operation. Based on Act No 59 of 2008 Waste Act, the waste stream generated from waste rock is classified as a Type 4 Waste with the following definition: "*Excavated earth material not containing hazardous waste or hazardous chemicals*." This waste stream classification must be disposed of at a Class D Landfill. The containment barrier design associated with Class D Landfill is 150 mm thick base preparation requiring minimal earthworks.

Rainfall in this area is classified as dirty water and will be reticulated through a series of concrete-lined dirty water channels into silt traps and into a dedicated pollution control dam.

Waste rock material will be used to construct a visual / audial attenuation berm between the mining complex and that of the local Kgatu village community, which will assist in buffering sound and visual pollution to the village occupants.

A waste rock dump for the future North Portal will be developed to the same specification. Figure 18-11 shows a general view of the stockpiles.

Figure 18-11: Stockpiling and Reclamation Areas – South Complex



18.7 Stockpile Reclamation

18.7.1 Crushed Ore Stockpile

ROM conveyed from underground to surface will be fed into primary crushers on surface before being conveyed to a crushed ore stockpile. One stockpile will be created for the Southern Portal and one for the Central Portal. The stockpiles will be served by a common tunnel that will allow for the withdrawal of the material using vibrating feeders and a conveyor system.

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 are required to be processed as a controlled blend in the process plants to maximize process plant recovery.

18.7.2 Temporary Ore Stockpile

Ore from underground operations brought to surface during the mining development phase (without conveying infrastructure) and prior to the completion of the processing plant will initially be trucked to surface and deposited on the temporary ore stockpile where it will be stored until the commissioning of the plant.

Ore of equivalent metallurgical characteristics will be stockpiled together. Once the surface overland conveyors are operational this stockpiled, material will be introduced to the crushing system by means of front-end loaders tipping into the reclaim hoppers to feed a primary crusher.

Based on the current mine production schedule, the stockpile was designed to store up to 505 000 tonnes of ore at a 20 m height prior to the start of each of the process plants. During the initial months of plant operation, the plant will be fed from a combination of ore mined and ore reclaimed from the stockpiles.

18.7.3 Topsoil Stockpiles

The construction of surface infrastructure at the South Complex, future North Portal, and processing plant will necessitate a 200 mm topsoil strip prior to earthworks and construction activities. The topsoil material will be stockpiled for reuse, as directed, for clean stormwater diversion berms and replacement purposes, when required.

18.8 Central Assay Laboratory

The Waterberg Project design allows for a centralised laboratory to be designed and operated by a third-party supplier. The Waterberg Project will supply the laboratory building and the associated equipment. The current allowance is for a 100% manual preparation system; however, the opportunity exists to change to a robotic, or a semi-automated preparation system, which will reduce the number of personnel, but increase initial capital requirements.

18.8.1 Laboratory Scope and Analytical Methods

The laboratory scope is summarised in Table 18-4.

Table 18-4: Waterberg Laboratory Scope Summary

Sample Type	Sample Size (kg)	Samples per month	Turnaround Time (h)	Analytical Method
Mine Grade Control	10	3 040	7 days	Fire assay (4E) ICP (Ni, Cu, MgO, SiO_2, Fe) Leco (S)
Geological	10	1 520	7 days	
Laboratory Testwork	2	150	12 – 24	
Process Control	10	2 430	4 for ICP – 24 for 3E	Fire assay (3E) ICP (Ni, Cu)
Metal Accounting	10	1 050	24 – 48	Fire assay (6E) ICP (Ni, Cu, MgO, SiO_2, Fe) Leco (S)
Environmental	2 L	480	24 – 48	Water Analysis

18.8.2 Laboratory Human Resources

The laboratory will operate 24 hours per day, 7 days per week, 365 days per year with 43 staff members working 12-hour shifts per day, 7 days a week, on a 4-shift panel rotation. The laboratory resource plan is presented in Table 18-5.

Table 18-5: Waterberg Laboratory Resource Plan

Sample Type	Total Staff	Crew 1	Crew 2	Crew 3	Crew 4
Total	43	12	10	11	10
Lab Manager	1	1			
HSE Representative	1			1	
Shift Chemist	4	1	1	1	1
Weighers	4	1	1	1	1
Wet Technician	8	2	2	2	2
Fire Assayers	4	1	1	1	1
Fire Assay Technician	4	1	1	1	1
Sample Prep Technician	16	4	4	4	4
Cleaner	1	1			

18.8.3 Laboratory Information Management System

Provisions were made in the costing to install a laboratory information management system into the Waterberg Project laboratory, which will allow the processing of samples and handling of all analytical data efficiently in a controlled and secured database environment, along with the necessary QA/QC requirements.

18.9 Tailings Storage Facility

Epoch was appointed by DRA to complete the FS design of the TSF and its associated infrastructure.

18.9.1 Tailings Storage Facility Design Criteria

The LOM production of concentrator tailings will amount to 93M tonnes over 45 years, delivered to the TSF after backfill requirement – it is noted that the backfill plant will prepare full plant tailings without any form of classification being implemented. DRA determined that the particle SG of the tailings was 2.96. The design criteria are summarized in Table 18-6.

Table 18-6: Design Criteria

Item	Criteria	Value	Source
1	Ore Type	Pt	DRA
2	Design Life of Facility	45 years	DRA
3	Average Tailings Deposition Rate	2 330 957 tpa	DRA
4	Total Tailings	93 036 911 tonnes	DRA
5	Particle SG	2.96	DRA
6	Average Particle Size Distribution	80% passing 75 µm	DRA
7	% Solids to Water Ratio (by Mass)	50	DRA
8	Delivery Method	Hydraulically Pumped	DRA
9	Maximum Rate of Rise	2 m/year	Epoch

18.9.2 Site Selection and Key Components

A site selection study was undertaken to locate an appropriate site for the TSF. Five sites were identified during the study. A risk-based evaluation of each site was undertaken to determine the lowest risk option by assigning a risk rating to each predetermined risk category (e.g. environmental damage, loss of life, etc.).

Following is a summary of the main characteristics of each site.

- Ketting ranked first on the weighted site selection ranking as a result of its safety and environmental ratings.
- Goedetrouw South ranked third due to scoring well in a number of categories, particularly, safety and public health; however, the site would require relocation of a community and so it was not considered further.
- Goedetrouw North ranked last due to its safety and environmental ranking as a result of its close proximity to human settlements and water resources. The site may also encroach on mining portal positions.
- Norma ranked second, even considering the large starter wall volume and proximity to a number of houses.
- Early Dawns scored fourth on the ranking due its low score for the safety and environmental category. Further drawbacks to the site are that it is possibly in an environmentally sensitive area and upstream of a community.

It was determined that the site on Ketting farm would be the most cost-effective option. With few people residing downstream of the site, it was found to be the lowest risk option.

The TSF was designed to store a total of 93M dry tonnes of tailings over a period of 45 years. The total footprint area of the TSF will be 287 Ha. The TSF comprises the following facilities.

- A tailings dam (TD) with a footprint area of 171 Ha and a maximum height of 65 m from the lowest contour.
- A 34 500 m^3 RWD.
- A 256 000 m^3 SWD.
- Associated infrastructure (i.e. solution trench, catchment paddocks, toe drains, etc.).

18.9.3 Geochemical Classification of the Tailings

The geochemical properties of the tailings were tested in 2017 to determine the lining requirements in terms of South Africa's legislation [National Environmental Management Waste Act, Act No. 59 of 2008, National Norms and Standards for the Assessment of Waste for Landfill Disposal (Regulation 36784)].

Two tailings samples (Central-F and South-T ore zones) were assessed by identifying the chemical substances present in the waste by analysing the total concentrations and leachable concentrations of the elements that have been identified in the tailings and comparing that to the threshold limits specified in Section 6 of the National Norms and Standards, Regulation 635.

Tailings are classified into 4 categories of waste, Waste Type 0 to Waste Type 4, where Waste Type 0 is considered extremely hazardous and Waste Type 4 is considered inert.

The testwork was undertaken by GCS Environment Engineering (Pty) Ltd (GCS). GCS classified the tailings as Waste Type 3, as they found 4 elements [Cu, Ni, selenium (Se), and Sb] in the total concentrations test in excess of the limits for Waste Type 4 but less than Waste Type 2. The leachable concentrations test resulted in no concentrations applicable to Waste Type 4.

Each waste type has a corresponding liner specification in terms of the Act, such that Waste Type 3 requires a Class C liner.

18.9.4 Class C Liner

A Class C liner comprises the following items.

- 1.5 mm HDPE Geomembrane
- 300 mm of Compacted Clay [or a Geosynthetic Clay Liner (GCL)]
- A leakage Detection System

A GCL was selected to replace the compacted clay as no available clay source nearby has been identified. Aquatan (Pty) Ltd provided a cost to supply and install the GCL and the HDPE liners.

18.9.5 Geotechnical Investigation

A geotechnical investigation of the TSF site was completed by Inroads Consulting (Pty) Ltd (Inroads). This included excavation, drilling, profiling of test pits and drill holes, sampling of soils, and the laboratory test work performed on the samples.

The soils encountered at the TSF are characterised by transported soil of mixed origin, but mainly of aeolian provenance comprising silty sands of loose to medium dense and occasionally dense to very dense consistencies. It generally exhibits a pinholed structure suggesting that it has the potential to undergo additional collapse settlement if loaded and subsequently wet. The sand overlies talus and nodular ferricrete and occasional calcrete nodules or, where the latter are absent, it extends to the bottom of the pits at an average depth of 2.8 m in the range of 0.4 to 5.8 m.

No groundwater was noted in any of the drill holes or test pits; however, the investigation was completed at the end of the dry season.

18.9.6 Seepage and Stability Assessment

The stability of the TD was assessed under various seepage conditions. The results show that the TD is stable with a factor of safety well above the required minimum of 1.5 under normal operating conditions. Abnormal operating conditions such as a large pool of water or a damaged liner emphasize that water should not be stored on the TD as it was not designed to store large quantities of water and the factor of safety would reduce to below the minimum levels. Abnormal conditions should be avoided through proper quality controls and supervision during construction and especially operations.

18.9.7 Depositional Methodology

Results obtained from tests on the expected Waterberg tailings depict a material with a large quantity of fines and clay minerals. The tailings are thus expected to exhibit shrinkage cracks and form flat beaches. The method of deposition should aim to minimise the shrinkage capabilities of the tailings and thus minimise the risks associated with internal erosion and piping.

It is concluded that the Waterberg Project TSF should make use of the hybrid paddock-spigotting method of deposition to ensure a dense outer wall, sufficient freeboard, and adequate drainage. The initial stages of deposition behind the starter wall will be used to complete trail paddocks that will provide additional knowledge on the behaviour of the material. After sufficient field data is available, the tailings operator may choose to implement a more optimal deposition method such as a spigotting-only operation.

Piezometers are to be installed during start up to measure the level of the phreatic surface through the TD. Occasional piezocone probing may be required during operations to assess the densities and the consolidation characteristics of the tailings, as well as to verify the stability of the facility throughout the LOM.

18.9.8 Water Balance

An actual daily water balance was developed to determine the average potential volume of return water from the TSF. The water balance model comprised inflows and outflows from/into the TSF.

The inflows are comprised of the following items.

- Daily Rainfall Records (onto TSF, RWD, and SWD)
- Tailings Slurry Water

The outflows are comprised of the following items.

- Daily Evaporation Records (from TSF, RWD, and SWD)
- Interstitial Lockup (water held in the voids between tailings particles)
- Water Returned to the Process Plant

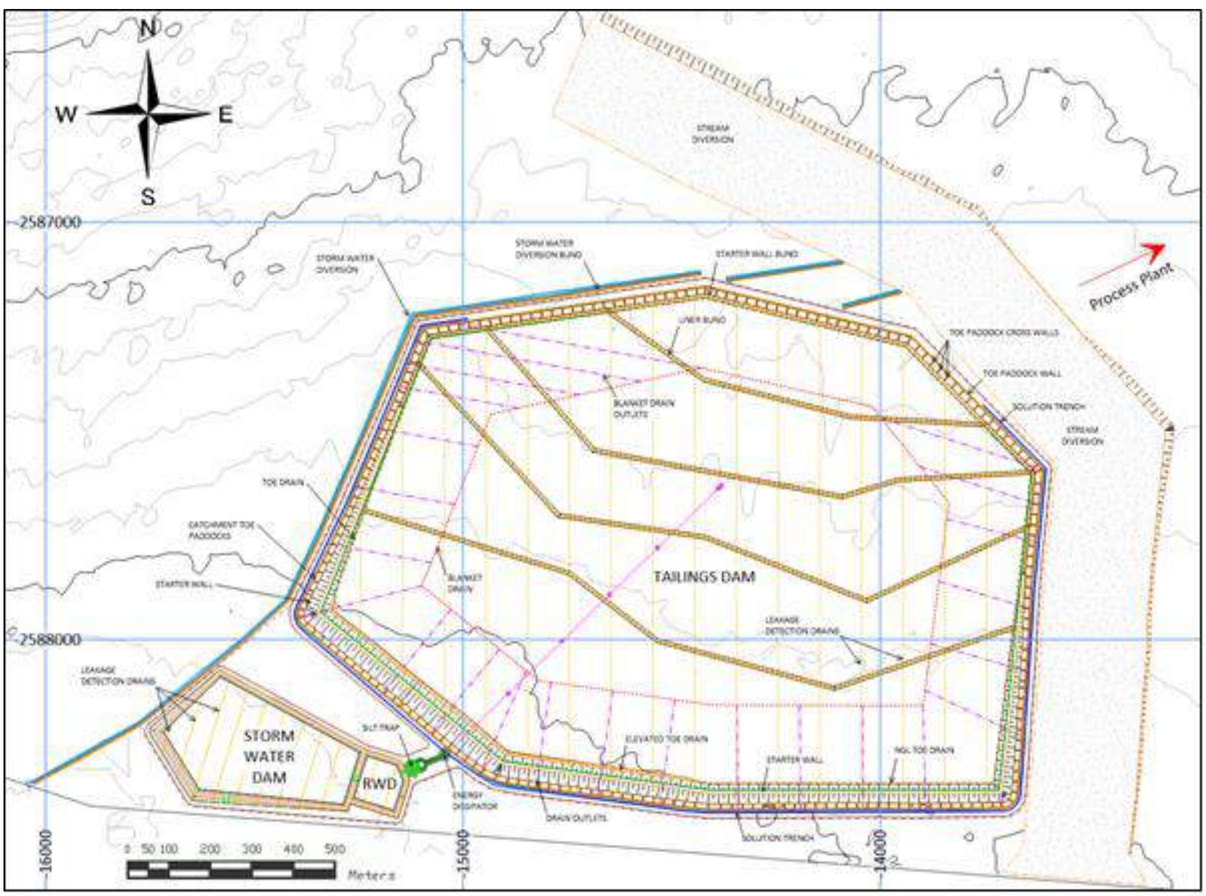
The average volume of water returned to the process plant was determined to be 117 719 m^3/month or 54% of the water sent to the TSF. This includes the seasonal variations of rainfall and evaporation; therefore, reduced returns can be expected between April and September and higher returns between October and March.

18.9.9 Key Design Features

The layout of the TSF is shown in Figure 18-12 and the key design features of the facility are listed below.

- A TD constructed by upstream, self-raised, hybrid-spigotting deposition method complete with the following items.
  - An engineered, earth-filled starter wall.
  - A concrete penstock and pipeline decant system.
  - Toe and blanket drain seepage collection system (to reduce phreatic level).
  - Catchment paddocks at the downstream toe of the TSF.

Figure 18-12: Tailings Storage Facility Layout



Source: (August 2019, Epoch Resources, BFS Design Drawings)

  - A Class C liner.
  - Stormwater diversion trenches.
- A lined SWD and RWD complete with the following items.
  - A Class C liner.
  - Spillway for rainfall events above the 1 in 100-year storm event.
- A stream diversion to divert runoff around the TSF.

18.9.10 Risk Identification

Following are the summarised risks associated with the TSF.

- Desiccation / shrinkage cracking may result in ratholing and tailing spills. Utmost care must be taken during operations to ensure that the cracking is minimised through optimization of the cycle times and deposition into the hybrid paddocks.
- The extent of the collapsibility of the soils need to be investigated further through impact roller tests and additional sampling to ensure differential settlement is minimised. If this is not managed correctly, shearing of subsoil pipes or the liner could occur.
- During major storm events, water must be removed as soon as possible. As the facility is a self-raised facility, it does not have capacity for storing water. It is critical that this water be removed quickly to prevent overtopping and eventual erosion.

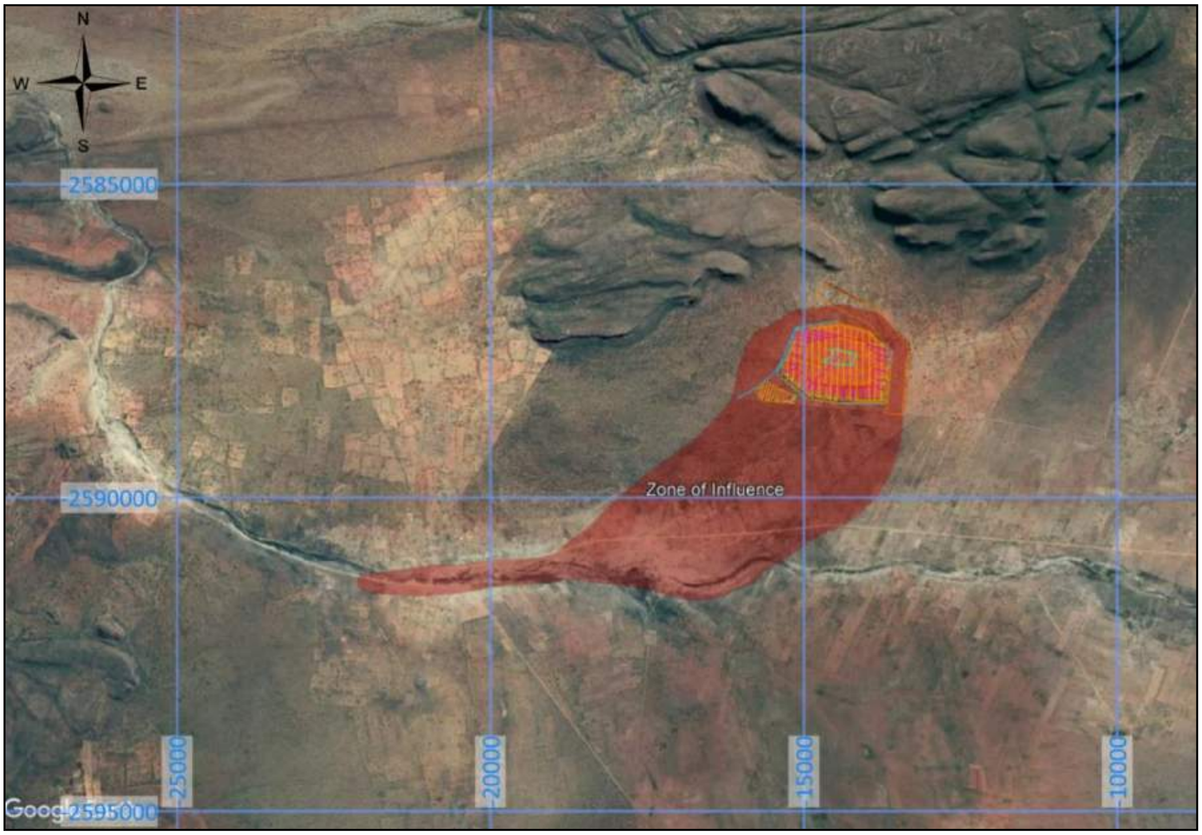
18.9.11 Safety Classification

The TSF was classified according to the South African National Standards, Code of Practice for Mine Tailings (SANS 0286:1998). This classification provides the basis for the implementation of safety management practices for specified stages of the life cycle of a TD. The code prescribes the aims, principles, and minimum requirements that apply to the classification procedure. The classification in turn gives rise to minimum requirements for investigation, design, construction, operation, and decommissioning.

The safety classification serves to differentiate between high, medium, and low hazard based on the potential to cause harm to life and/or property. The facility is classified as high hazard due to the presence of some small farms downstream and inside the zone of influence of the TSF.

The zone of influence, as shown in Figure 18-13, may be described as the extent of the area around the TSF that may be affected with time, taking into consideration the possible impacts that may arise from the TSF (e.g. flow slide, surface and groundwater contamination, sterilization of arable land, etc.).

Figure 18-13: Zone of Influence for the TSF



Source: (August 2019, Epoch Resources Zone of Influence Determination and Google Earth Background)

18.9.12 Conclusions

From the engineering studies completed, the following conclusions were reached.

- A suitable site was identified in the site selection study, Ketting, which will accommodate the specified quantity tailings.
- Seepage and stability modelling indicate that the facility will be stable under the design conditions with factors of safety well above 1.5. Abnormal conditions (i.e. a large pool, damaged liner, and/or damaged drains) will affect the stability of the facility and must be avoided through application of quality controls and supervision of construction and operations.

18.9.13 Recommendations

The following recommendations are provided for the TSF detailed design phase.

- Confirm design criteria and site selection.
- Further analysis and design of the stream diversion.

- Further optimization of the capital and operating cost estimate, where possible, by completing the following tasks.
  - Developing a tender enquiry on the detailed design to acquire final construction rates.
  - Further optimization of earth and civil works, where possible.
  - Finalising the responsibilities of the operator by incorporating input from all parties (contractor, client, and consultants).
- Further evaluation of geochemical risk in terms of liner requirements / details.
- Confirmation of survey data accuracy. It is recommended to complete survey points of the site to confirm elevation.
- Further geotechnical assessments of the collapsible soils, including impact roller testing to determine its effectiveness.
- Continued monitoring of the risks relating to the following items.
  - Collapsible soils.
  - Severe desiccation cracking.

18.10 Surface Paste Backfill Plant

18.10.1 Backfill Product

The mining methods include Longitudinal and Transverse Sublevel Stoping with backfill as support medium. Tailings from the Concentrator Plant will be dewatered and blended with binder to produce a cemented paste backfill.

18.10.2 Key Assumptions and Design Criteria

The paste backfill DFS is based on the following key assumptions.

- When not backfilling, the concentrator tailings will be diverted to the TSF through the concentrator's discharge system developed by DRA.
- When backfilling is taking place, the entire tailings feed stream is fed to the backfill plant (578 t/h) and utilised for backfilling.
- Paste backfill will always require binder for placement underground. For secondary stopes, there is a minimum amount of binder required to mitigate liquefaction.
- The binder estimates and requirements are based on annual mined volumes determined by Stantec.
- Tailings from the South Complex are 75% from Central Complex tailings and 25% from T Zone tailings.
- Tailings from the North Complex are 50% from north Super F tailings and 50% from boundary tailings.

The design is further based on the results of the testwork completed during the course of the study.

18.10.3 Testwork

Thickening and filtration tests were completed during the PFS in 2016. The dewatering test results from the campaign completed in 2016 were applied to size the backfill plant dewatering equipment.

SSBS undertook rotational viscometer tests to determine the rheological flow behaviour properties as well as UCS tests to determine the strength gain for different cement contents and curing periods.

Rotational viscometer tests were undertaken on uncemented as well as a cemented South and North complex tailings.

Following from the results of a trade-off Study on different cement types, Minova Fillcem Cement (CEM III A 42.5N) was used for the cemented tests at a cement content of 8%.

Cement mortar compressive strength tests were carried out in accordance with the SANS 50196-1 standard to confirm that the cement comply with the minimum strength requirements specified by SANS 50197-1 prior to testing.

18.10.3.1 Unconfined Compressive Strength Tests

Cement Mortar Compressive Strength Tests

Cement mortar compressive strength tests were completed in accordance with the SANS 50196-1 standard to confirm that the cement comply with the minimum strength requirements specified by SANS 50197-1 prior to testing.

Unconfined Compressive Strength Tests

The UCS test results of the backfill material are used to determine the cement dosage rate to achieve the minimum required backfill strengths. The UCS tests were conducted for cement contents of 4%, 8%, 12%, and 16%.

The UCS achieved for various water to cement ratios for the North and South Complex tailings are shown in Figure 18-14 and Figure 18-15, respectively.

The UCS test results for the North and South Complex tailings are summarised in Table 18-7 and Figure 18-16.

The results show that the North Complex tailings produced a higher strength for the same cement content and mass concentration than the South Complex tailings.

Figure 18-14: Water Cement Ratio versus Uniaxial Compressive Strength for the North Complex Tailings

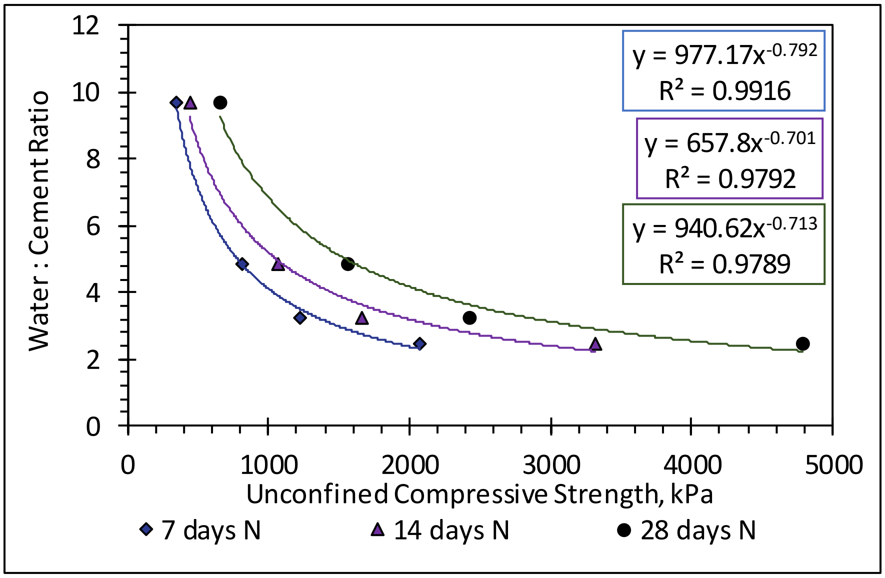


Figure 18-15: Water Cement Ratio versus Uniaxial Compressive Strength for the South Complex Tailings

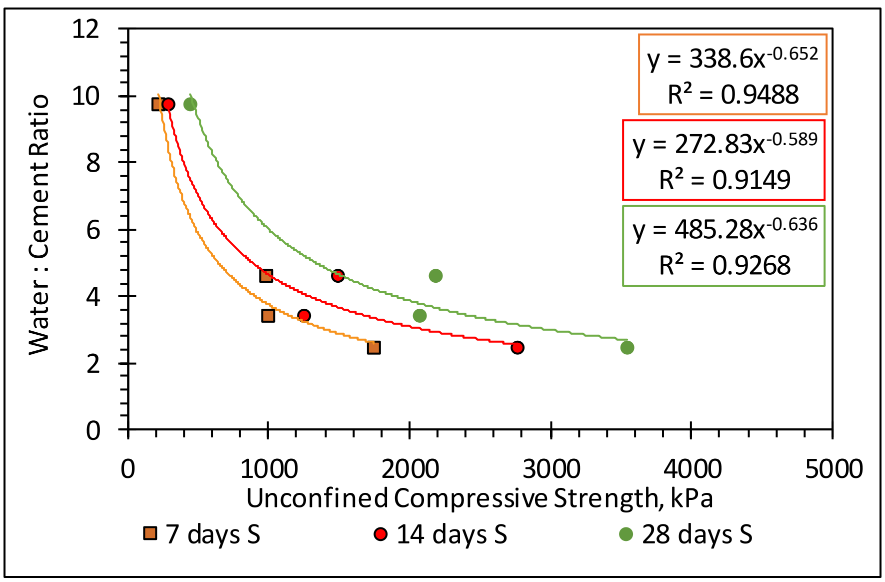
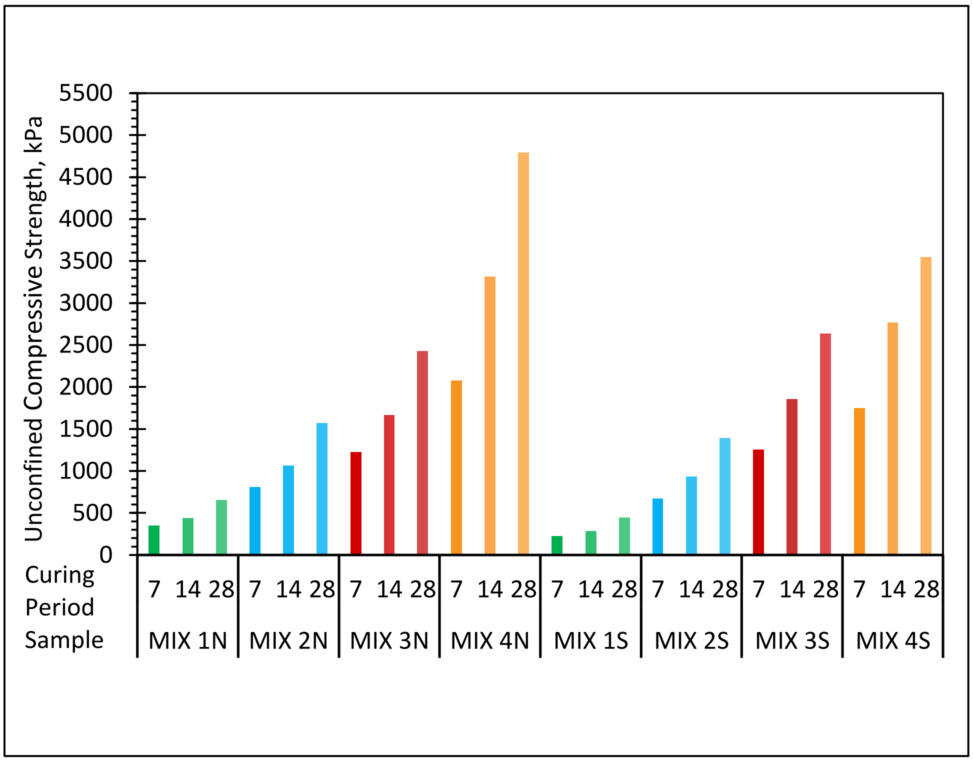


Table 18-7: Unconfined Compressive Strength Test Results

Sample	Admixture	UCS (kPa)		
Curing Period		7 Day	14 Day	28 Day
Mix 1N	North 4% @ Cw =72%	347	438	654
Mix 2N	North 8% @ Cw = 72%	807	1 062	1 566
Mix 3N	North 12% @ Cw = 72%	1 225	1 663	2 423
Mix 4N	North 16% @ Cw = 72%	2 075	3 317	4 790
Mix 1S	South 4% @ Cw = 72%	220	285	446
Mix 2S	South 8% @ Cw = 72%	671	933	1 392
Mix 3S	South 12% @ Cw = 72%	1 250	1 856	2 632
Mix 4S	South 16% @ Cw = 72%	1 748	2 768	3 544

Figure 18-16: Tailings Only Unconfined Compressive Strength versus Curing Period



18.10.4 Backfill Plant Capacity

Table 18-8 presents the operating parameters applied to determine the backfill plant capacity.

Table 18-8: Operating Parameters

Item	Operating Parameters		
Complex	Central / South		North
	Central	South	
Operating Days per Annum	353 Days		353 Days
Number of Backfill Shifts / Day	2 Shifts / Day		2 Shifts / Day
Shift Duration	10.5 Hours / Shift		10.5 Hours /Shift
Backfill Face Time per Shift	7.0 to 7.5 Hours / Shift		7.5 Hours / Shift
Backfill Plant Availability	90%		90%
Backfill Hours (annual)	4 448 Hours / Year		4 766 Hours / Year
Backfill Hours (monthly)	371 Hours / Month		397 Hours / Month
Head Feed Ratio	75%	25%	100%
Head Feed	300 000 t / Month	100 000 t / Month	400 000 t / Month
Void Volume*	100 000 m^3 / Month	33 333 m^3 / Month	133 333 m^3 / Month
Shrinkage Allowance	7.5%	7.5%	7.5%
Overbreak Allowance	10%	10%	10%
Monthly Backfill Design Volume	118 250 m^3 / Month	39 417 m^3 / Month	157 667 m^3 / Month
Backfill Density (Cw)	71% to 72% Cw		67% to 71% Cw
Hourly Tonnage Rate**	3 x 144 t / Hour	144 t / Hour	4 x 144 t / Hour

Notes:

- * Void volume calculated based on a rock density of 3.00 t/m^3
- ** Dry mass tailings (excluding binder)

18.10.5 Process Overview

Tailings are received at the paste backfill plant from the process plant via the tailings pipeline. When the backfill plant is in operation, tailings are fed to the backfill plant, otherwise tailings are diverted to the TD.

Tailings are received in two agitated tailings tanks and pumped to four disc filters at the top of the backfill structure. The tailings are dewatered to a mass solids concentration of 77% m without thickeners. The filter cake is conveyed to four continuous twin shaft mixers located underneath the filters via four belt conveyors.

Cement from the supplier is received by bulk road tankers and discharge into in 8 cement bulk silos with a capacity of 300 m^3 per silo. Cement blowers are used to transfer cement from the bulk silos to four active silos with a capacity of 60 m^3 per silo. Four screw conveyors are used to transfer the cement to the continuous twin shaft mixers.

The four continuous twin shaft mixers receive feed from the filters in the form of filter cake, cement from the active silos, and a percentage of tailings feed used to dilute filter cake to obtain an appropriate consistency when mixed to prepare the backfill material.

Backfill material containing a 6% cement content (or as required) from the continuous mixers discharges into four backfill tanks from where the material discharges into four pipelines. Three backfill pipelines are dedicated to backfilling the three drill holes at the Central Complex. There is one dedicated backfill overland pipeline to the South Complex. A positive displacement pump is used to transport the backfill overland and pump it underground via the drill hole at the South Complex.

The water requirements of the plant are supplied from the potable water treatment plant with raw and process water supplied from the Concentrator Plant.

18.10.6 Further Backfill Work and Studies

The backfill plant capacity is based on receiving tailings at a 100% feed rate from the Concentrator Plant with two operational shifts per day. After each shift, the backfill plant will stop, be flushed, and prepared for the next shift.

The opportunity exists to increase the backfill plant operating hours by considering a “hot” change over between shifts and operate the backfill plant on a continuous basis.

19 MARKET STUDIES AND CONTRACTS

A formal marketing study was not completed as part of this DFS; however, as one of the JV partners is a PGE producer (including mining, concentrating, smelting, base metal refiner, and precious metal refiner), the marketing aspects are understood 'in house.' Table 19-1 shows economic PGEs and base metals for the Waterberg Project in order of economic value at 3 year trailing and 04 September 2019 spot prices for the first 13 years of production with the ratio of T-zone to F-zone will be approximately 25:75 and for the life of the mine.

Table 19-1: Economic PGEs and Base Metals for first 13 Years and Life of Mine

Metal	Approximate Percent of Revenue (3-year trailing to 04 Sept 2019)		Approximate Percent of Revenue (04 Sept 2019 Spot Price)	
	First 13 years	LOM	First 13 years	LOM
Pd	54.3%	55.8%	59.4%	60.6%
Pt	23.2%	22.1%	18.2%	17.2%
Au	8.3%	6.1%	7.3%	5.3%
Ni	8.7%	10.5%	9.5%	11.3%
Cu	4.1%	4.0%	2.7%	2.6%
Rh	1.5%	1.5%	2.9%	3.0%

Waterberg Project will be a PGE flotation concentrate producer and there was significant growth of 'independent' concentrate producers in South Africa during the last 15 to 20 years. As such, toll treatment of flotation concentrates or purchase agreements are common within the PGE industry with the major producers, including the JV partner. Waterberg may be one of the future 'independent' concentrate producers and initially, a concentrate sales agreement will be required to treat the production from the mine.

No other PGE smelter operators were formally approached to express interest in the toll treatment of the Waterberg concentrate. Informally, there is significant interest in processing this flotation concentrate, especially with the JV partner.

No formal contracts were entered into for the Waterberg Project implementation apart from with the JV partners (JOGMEC, Hanwa, and IMPLATS).

19.1 PGM and Base Metal Market Review

The market and prices for Pt and Pd have diverged since the completion of the Waterberg PFS in 2016. The price of Pt was negatively impacted by the decrease in demand for diesel automotive powertrains, particularly in Western Europe, stemming from the Volkswagen "dieselgate" scandal. One of Pt's primary uses is for pollution control (autocatalysis) in diesel vehicles. The general sentiment for diesel automotive adoption continues to be negative and Pt prices are expected to be weak going forward. Perceived oversupply of Pt from South African producers also weighed on sentiment for the metal with analysts predicting a significant surplus of available metal going forward and a “lower for longer” price environment. Several large Pt mines in South Africa are in the process of being closed or restructured, which could result in upside risk to the Pt price should supply be significantly curtailed. A burgeoning market for fuel cell technology that uses Pt may create a new demand segment over the medium to longer-term horizon. In general, there is an expectation for Pt prices to remain subdued in the near term.

The main beneficiary of Pt’s slide is its sister metal Pd. While diesel automotive sales have been weak, gasoline car sales have been very strong, usurping sales from a declining diesel market. Pd’s primary use is for pollution control in gasoline vehicles. Autocatalyst demand for Pd hit record levels in 2018 as stricter vehicle testing procedures lifted loadings on European cars. The introduction of the European Real Driving Emissions test is expected to increase PGM loadings for both gasoline and diesel cars significantly. Stringent new emissions legislation in China is scheduled to take effect in 2020. The China 6 standards represent a step-change in Pd loadings, which will put continued pressure on metal availability. The Pd market experienced multiple years of significant deficits as strong demand and limited supply response led to successive years of price increases. Supply from recycling, investment liquidation, and sales of pipelines stocks from major producers filled the supply void in the interim, although, any growth in supply from these sources is unlikely. There is some discussion of autocatalyst manufacturers potentially substituting Pd with cheaper Pt, although there is no evidence that this is currently occurring. Any effort at substitution would likely require a wider price differential between the two metals and take several years to implement. Industry analysts expect Pd prices to remain strong going forward.

Minor PGM elements and base metals contribute to the overall Waterberg revenue basket. Pricing and demand for Rh has been particularly strong. Ni and Cu prices have been volatile with future performance predicated on global growth and industrial demand.

Table 19-2 and Table 19-3 present the actual and forecasted Pd and Pt supply and demand, respectively.

Table 19-2: Palladium Supply and Demand ('000 oz)

Supply	2017	2018	2019
South Africa	2 547	2 543	2 744
Russia	2 452	2 976	2 792
Others	1 409	1 458	1 460
Total Supply	**6 408**	**6 977**	**6 996**
Gross Demand			
Autocatalyst	8 532	8 721	9 496
Jewelry	173	157	156
Industrial	1 827	1 918	1 812
Investment	-386	-574	-310
Total Gross Demand	**10 146**	**10 222**	**11 154**
Recycling	-2 863	-3 124	-3 349
Total Net Demand	**7 283**	**7 098**	**7 805**
Movements in Stocks	-875	-121	-809

Source – 'Johnson Matthey Market Report' May 2019

Table 19-3: Platinum Supply and Demand ('000 oz)

Supply	2017	2018	2019
South Africa	4 450	4 467	4 565
Russia	720	687	668
Others	953	959	956
Total Supply	**6 123**	**6 113**	**6 189**
Gross Demand			
Autocatalyst	3 248	3 051	3 128
Jewelry	2 400	2 269	2 227
Industrial	2 117	2 459	2 322
Investment	361	67	858
Total Gross Demand	**8 126**	**7 846**	**8 535**
Recycling	-2 047	-2 105	-2 219
Total Net Demand	**6 079**	**5 741**	**6 316**
Movements in Stocks	44	372	-127

Source – 'Johnson Matthey Market Report' May 2019

19.2 PGM and Base Metal Prices

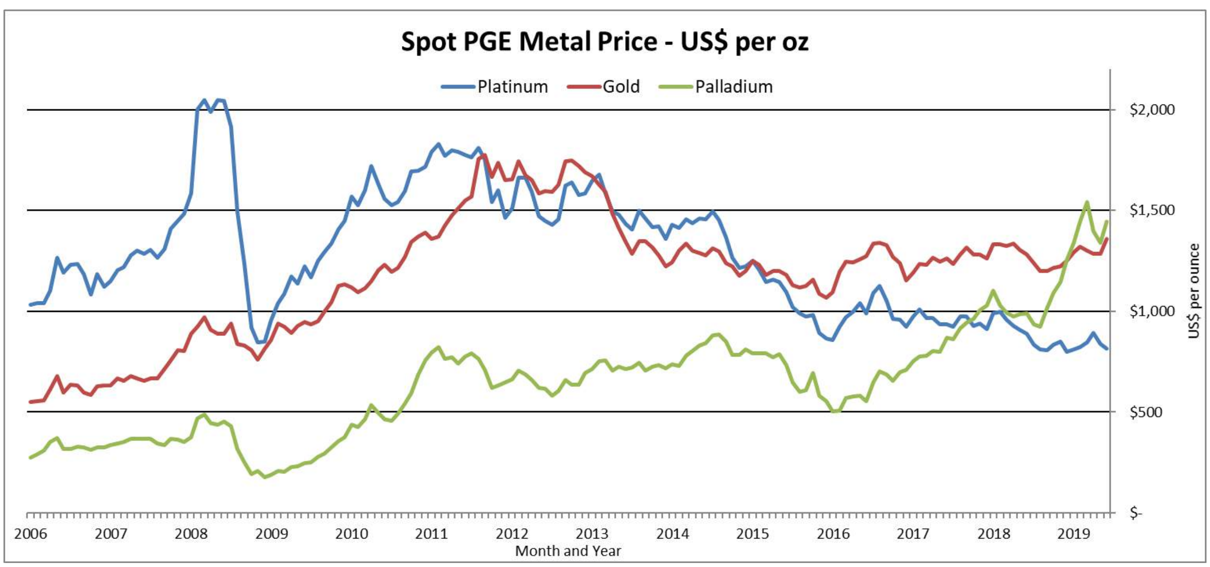
The spot prices of the metals of economic interest to the Waterberg (Pd, Pt, Au, Ni, Cu, and Rh) were reviewed for the last number of years on a average basis with three-year, two-year, and one-year (rolling average) and monthly spot prices being determined as of 04 September 2019. The company is listed on the NYSE-American exchange in the United States and requires that economic studies consider trailing average metal prices over a three-year period. Spot and other metal prices will be evaluated in the financial sensitivity analysis.

These price decks (adjusted to 01 July 2019 value) were used in the financial evaluation to determine the economic viability of the project. The effective date for the price decks used is 01 July 2019 and the details are available in Section 21. The Waterberg Project is located within South Africa and a large proportion of the capital and operating costs will be generated in ZAR terms. The currency exchange rate to the major international currencies (US$, EUR, JPY, GBP) is also evaluated in addition to the metal prices.

19.2.1 Palladium, Platinum, and Gold Pricing

Pd prices have been rising during the last number of years with the increase in demand while Pt prices have been falling during the same period with the decrease in demand as shown in Figure 19-1. Au prices were stagnant during the last number of years but with a recent rally due to global uncertainty as shown in Figure 19-1. The Waterberg Project financial evaluation will be based upon the three-year trailing average metal price and associated averages and spot prices for sensitivities. These study prices are indicated in Table 19-4 and are the arithmetic average metal prices to show the trends over the recent periods.

Figure 19-1: Metal Pricing – Historical



Source – 'Johnson Matthey Metal Prices'

Table 19-4: Pricing for all Economic Metals

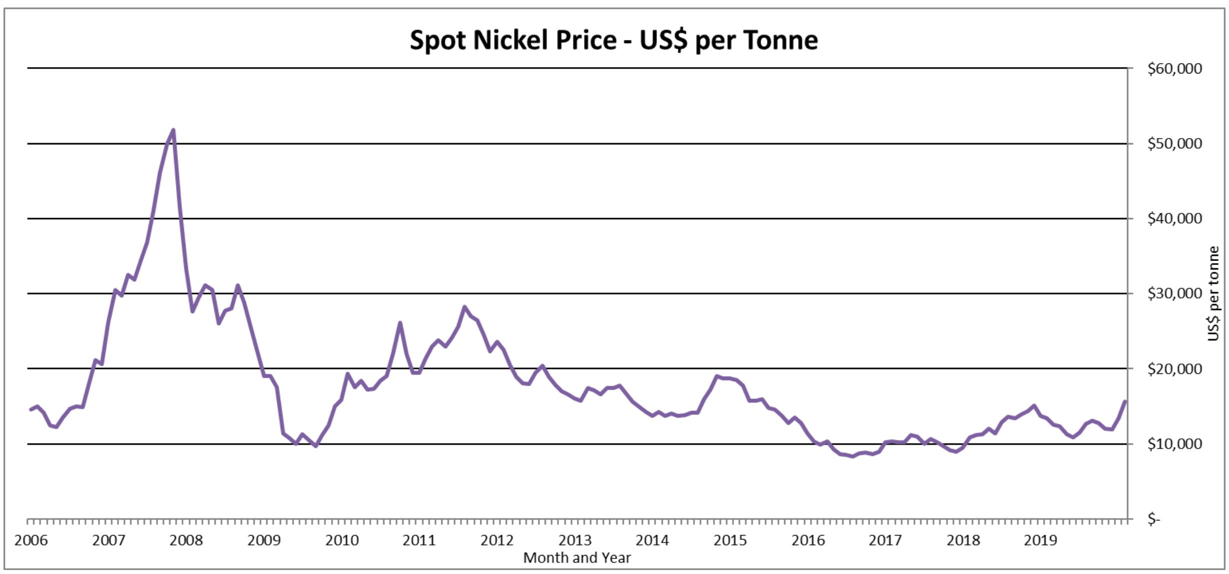
Period	Pd US$/oz	Pt US$/oz	Au US$/oz	Ni US$/tonne	Cu US$/tonne	Rh US$/oz
Three-year Trailing	$ 1 055	$ 931	$ 1 318	$ 12 248	$ 6 333	$ 1 930
Two-year Trailing	$ 1 174	$ 891	$ 1 322	$ 13 034	$ 6 530	$ 2 427
One-year Trailing	$ 1 338	$ 841	$ 1 318	$ 12 666	$ 6 146	$ 2 942
04 September 2019 Spot	$ 1 546	$ 980	$ 1 548	$ 17 855	$ 5 646	$ 5 036

Source – 'Johnson Matthey Metal Prices' & London Metal Exchange - Monthly Average

19.2.2 Nickel Pricing

Ni prices have been stagnant during the last number of years, as shown in Figure 19-2 with the decrease in demand due to global economic conditions. The Waterberg Project financial evaluation will be based on the three-year trailing average metal price and associated averages and spot prices for sensitivities. These study prices are indicated in Table 19-4.

Figure 19-2: Nickel Pricing – Historical

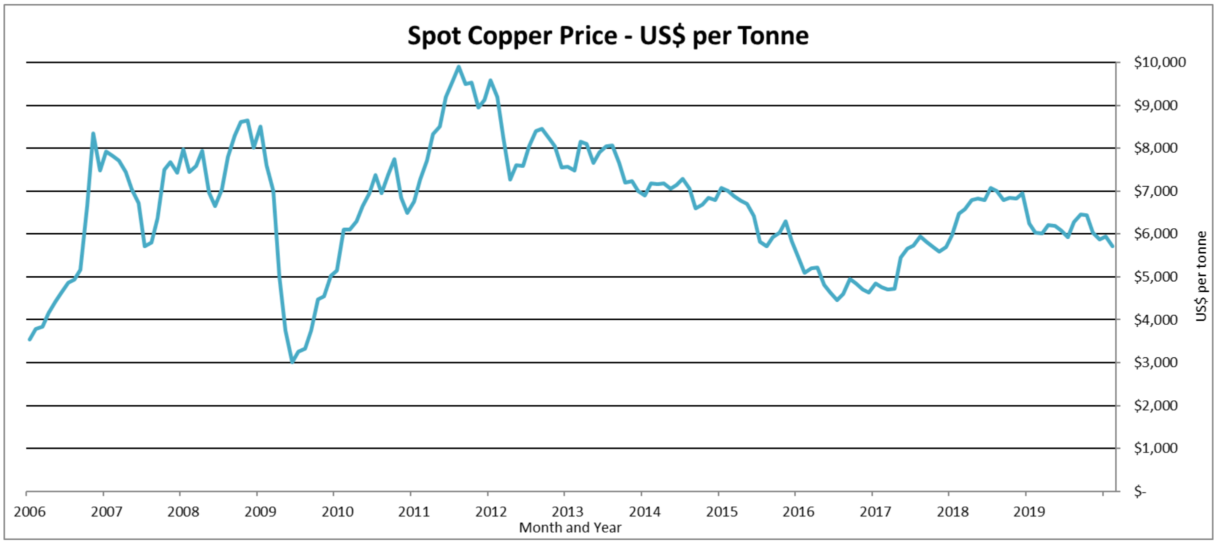


Source – 'London Metal Exchange - Metal Prices'

19.2.3 Copper Pricing

Cu prices have been falling during the last number of years with the decrease in demand due to the global economic crisis as shown in Figure 19-3. The Waterberg Project financial evaluation will be based on the three-year trailing average metal price and associated averages and spot prices for sensitivities. These study prices are indicated in Table 19-4.

Figure 19-3: Copper Pricing – Historical

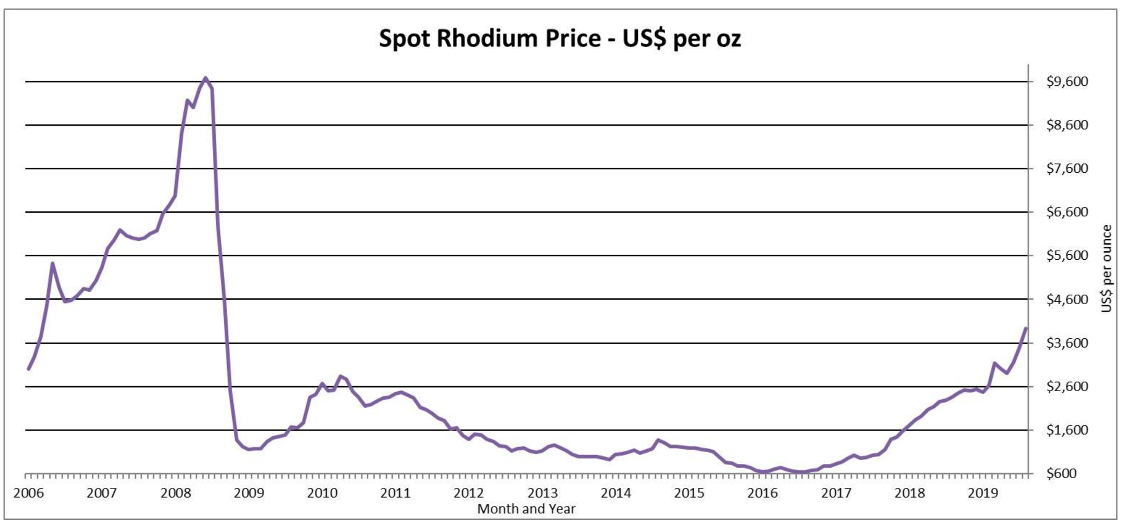


Source – 'London Metal Exchange - Metal Prices'

19.2.4 Rhodium Pricing

Rh prices have been rising during the last number of years from the lows during 2016 and the extreme highs of 2008, as shown in Figure 19-4 with the change in demand pattern. The Waterberg Project financial evaluation will be based on the three-year trailing average metal price and associated averages and spot prices for sensitivities. These study prices are indicated in Table 19-4.

Figure 19-4: Rhodium Pricing – Historical



Source – 'Johnson Matthey Metal Prices'

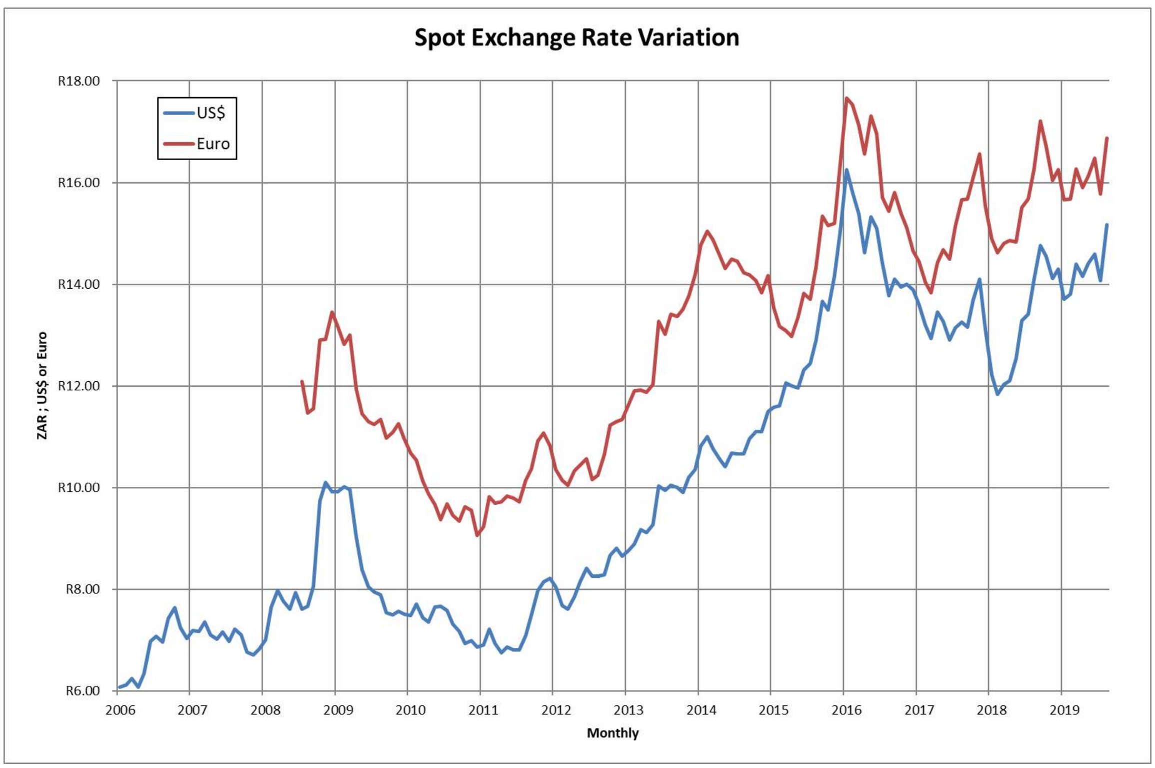
19.2.5 Metal Price Comparison

The average metal prices that are applicable to this DFS are shown in Table 19-4 for comparison purposes for the Base Case and different possible sensitivity periods that may be considered in the financial model.

19.2.6 Exchange Rate Evaluation

The exchange rate between the ZAR and the US$ and other major currencies was volatile during the last number of years, as shown in Figure 19-5 with the changing sentiment towards South Africa. The major currencies that may impact the project are US$, Euro, JPY, and GBP with the US$ having the highest impact due to metal prices being quoted in US$ as the norm. The Waterberg Project financial evaluation will be based on the estimated rate of exchange, namely R15.00: US$1.00, which is comparable to the August 2019 rate of R15.17 and is compared with the three-year trailing average rate of exchange and associated averages better understanding for sensitivity purposes. These historical exchange rates are indicated in Table 19-5.

Figure 19-5: ZAR to US$ and Euro Exchange Rate – Historical



Source – 'OANDA - Forex Prices'

Table 19-5: ZAR to Major Currencies Exchange Rate – Average and Spot

Rate of Exchange – ZAR				
Period	**US$**	**Euro**	**JPY**	**GBP**
Three-year Trailing	R13.59	R15.50	R0.123	R17.64
Two-year Trailing	R13.65	R15.85	R0.124	R17.92
One-year Trailing	R14.34	R16.25	R0.130	R18.39
04 September Spot	R14.92	R16.42	R0.140	R18.16

Source – 'OANDA - Forex Prices'

The Waterberg Project has accepted that the rate of exchange will be R15.00 per US$ and R16.35 per Euro for project costing. The Bloomberg forecast is also considered as detailed in Section 21 of this Technical Report.

19.3 PGM and Base Metal Contribution to Revenue

Based on the project revenue calculations and the Base Case metals pricing, the contribution from the 'pay metals' is indicated in Table 19-6. This is based on the 'prill splits' for the two major geological zones to be mined and this is independent of the production profile. The table clearly indicates that the PGEs are the major revenue contributor at more than 87%.

Table 19-6: Revenue Contribution to Concentrate

Metal	3-year Trailing		04 September 2019 Spot	
	T Zone	F Zone	T Zone	F Zone
Pt	22.4%	23.5%	18.3%	18.3%
Pd	42.8%	56.8%	48.8%	61.6%
Au	22.9%	5.5%	20.9%	4.8%
Rh	1.6%	1.6%	3.2%	3.1%
4E's	**89.7%**	**87.4%**	**91.3%**	**87.9%**
Cu	6.7%	3.7%	4.6%	2.5%
Ni	3.6%	8.9%	4.1%	9.6%
Total	**100.0%**	**100.0%**	**100.0%**	**100.0%**

Base metals (Cu and Ni) are financially important in terms of overall project return with the other precious metals [iridium (Ir) and ruthenium (Ru)] being of no economic value to the project. As with all industrial commodities prices continue to be volatile. Ni and Cu markets are closely linked to Chinese demand which continues to be difficult to predict.

19.4 Concentrate Production and Quality

The Waterberg Project will be producing a flotation concentrate, which will be sold, or toll treated so that the Waterberg 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 13 000 tonnes of concentrate per month at steady-state production or in excess of 155 000 tpa.

The quality of this concentrate was evaluated during the metallurgical testwork programme conducted at Mintek, Johannesburg. While this is a 'snapshot' based on a few samples from drill core, Table 19-7 indicates the anticipated production to be treated in the subsequent recovery process in terms of economic metals and elements of interest.

Table 19-7: Concentrate Quality – Major Elements

Concentrate Contents				
Element	**Units**	**Individual**	**Minimum**	**Maximum**
Pt	(g/t)	23	9	35
Pd	(g/t)	52	18	69
Rh	(g/t)	1	1	2
Ru	(g/t)	<1.0	ND	ND
Ir	(g/t)	<0.5	ND	ND
Au	(g/t)	5	2	27
4E	(g/t)	80	30	108
Cu	(%)	2.3	1.0	9.2
Ni	(%)	2.7	1.1	5.0
Fe	(%)	14.5	11	22
SiO_2	(%)	41.3	23	43
MgO	(%)	16.0	6	24
S	(%)	6.5	3	19

Minor elements that were evaluated during the testwork programme during the PFS and the FS and are indicated in Table 19-8 and show the potential for deleterious elements being fed into the subsequent recovery process as evaluated during the PFS and the FS. There are no expected deleterious elements indicated in the flotation concentrate.

Table 19-8: Concentrate Quality – Minor Elements

Waterberg Concentrate Minor Elements (Nominal)							
Element	**Unit**	**PFS**	**FS**	**Element**	**Unit**	**PFS**	**FS**
Ca	%	1.6	3.0	Rb	ppm	6.5	2.5
Al	%	1.6	2.6	Ge	ppm	<0.05	2.0
Ti	%	<0.05	0.1	Cd	ppm	<0.05	1.8
Mn	%	0.1	0.1	Nb	ppm	2.5	1.8
Cr	%	0.1	0.0	La	ppm	<12	1.3
V	%	<0.05	0.0	Sb	ppm	<0.05	1.2
K	%	0.0	<0.1	Ta	ppm	712.1	1.0
Chlorine	%	0.0	ND	Th	ppm	11.6	1.0
Co	ppm	711.8	1 262.8	Tl	ppm	3.8	0.6
Zn	ppm	678.6	462.7	Cs	ppm	<5	0.5
As	ppm	<0.05	89.3	U	ppm	5.0	0.5
Sr	ppm	36.1	51.2	Li	ppm	ND	<10.0
Pb	ppm	66.0	49.3	In	ppm	5.7	<0.2
Ba	ppm	36.3	29.6	Se	ppm	28.1	ND
Mo	ppm	9.8	10.1	Bromine	ppm	3.1	ND
Bismuth (Bi)	ppm	<0.5	8.2	Y	ppm	4.4	ND
Sn	ppm	<0.05	6.8	Zirconium	ppm	6.3	ND
Ag	ppm	8.4	6.7	Hafnium	ppm	<2.0	ND
Ga	ppm	<0.05	4.3	Mercury	ppm	2.0	ND
Ce	ppm	<2.6	2.7	Tellurium (Te)	ppm	4.5	ND
W	ppm	<1.2	2.7	Iodine	ppm	<0.07	ND

Additional economic metals that may be considered for the project include Ir, Ru, Co, and Ag, although the revenue stream generated from these metals will be insignificant.

The mineralogical composition of the concentrate is as detailed in Table 19-9.

Table 19-9: Concentrate Mineralogical Composition

Mineral	Primary Cleaner Concentrate	Secondary & Tertiary 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 on 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.

19.5 Concentrate Treatment Options

Marketing work for the project advanced since the completion of the PFS in 2016. The JV commissioned a study in 2017 for a specialist consulting firm to analyse and study potential off-take options and estimated commercial terms. As part of IMPLATS US$30M investment in the project, for a 15% stake, they acquired a right of first refusal for future smelting and refining of concentrate. Hanwa Co. Ltd. maintains the marketing right to solely purchase all the metals from the project at market prices, having acquired this right from JOGMEC. A concentrate sales agreement will need to be formalised to treat the production from the mine.

No smelter operators were formally approached to express interest in the toll treatment of the Waterberg Project concentrate to date. Based on work to date it is estimated that an appropriate amount of capacity is available.

19.6 Capacity Available Locally

Four PGE producers 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 installed additional smelter capacity during the last few years.

The Waterberg Project will produce a low-chromitite concentrates which can be blended with the high-chromitite UG2 concentrates produced by most of the Bushveld Complex mines and will assist in managing the negative impacts of the higher Sr and Fe content to the benefit of the Waterberg Project and the smelter operator using conventional smelter technology.

Outside South Africa, there is limited smelting capacity in Zimbabwe and Botswana, which could be considered; however, this would require statutory approval and is expected to be a short-term solution only during the ramp-up phase of the Waterberg Project. Export of concentrate would also have significant cost implications. It is estimated that there is adequate available smelter capacity for the Waterberg Project; however, steady-state production could place a significant strain on this capacity. Additional smelting capacity may need to be constructed in the industry to be able to treat the flotation concentrate from Waterberg and the other potential Platreef miners. Conversely, the closure of existing mines in the Rustenburg area could open fresh capacity.

Alternative hydrometallurgical treatment options exist, which could be considered applicable to the Waterberg concentrates; unfortunately, none of these are proven on a commercial scale. Significant developmental testwork would be required before any of these processes could be considered for treating the concentrate.

19.7 Smelting and Refining Contracts

IMPLATS retains a right of first refusal for future concentrate production; however, no formal smelting or refining contracts are in place for the Waterberg Project.

19.8 Metal Payability or Treatment Terms

Typical economic metal recoveries for the conventional smelting and refining route are between 96% and 98%.

Several tolling agreements are in place between the different smelter operators and can be summarised into the following 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, such as with Impala Refinery Services.

It is proposed for this DFS that the financial evaluation be based upon a fixed payability percentage with an average of 85% for all 4E metals, 73% for Cu and 68% for Ni. These are regarded as fair and reasonable although negotiations may change these terms based on the desirability of the concentrate. These payabilities were confirmed by the JV partner to be acceptable for DFS purposes as an ‘arms-length’ transaction.

The concentrate could be transported by Waterberg project to the Rustenburg smelter complex of Impala Refinery Services within South Africa.

Three smelting hubs exist within South Africa in relation to the Waterberg mine site: Polokwane (109 km southeast), Northam (312 km southwest), and Rustenburg (417 km south-southwest). Since the JV partner has a smelter complex at Rustenburg, it is anticipated that this will be the destination for the concentrate shipments. The transport distance from the Waterberg Project to the smelter gate in Rustenburg is 417 km.

A budget proposal was received with an estimated cost of R400-450 per wet tonne transported 417 km. The average transport cost for concentrate based on this proposal is R1.08 per wet concentrate tonne per km. The concentrate moisture will be about 12% resulting in the cost per dry tonne delivered being R476, which is based on transport rate and moisture content reduction, delivered to the Rustenburg area.

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 Project concentrate will be smelted and refined more quickly than any other concentrate being treated at a toll smelting facility.

Each of the payable metals (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 12 weeks for all metals.

In terms of payment, there may be mechanisms that can be adopted for the Waterberg Project whereby an upfront payment for 85% of the contained metals is available during the month of delivery, subject to an interest charge but this has not been included in the financial model. The balance of 15% of the payment will then be available after the full 'release' period of 12 weeks.

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 4E grade, potentially high chromitite content, and possible other deleterious elements such as Fe, As, Bi Se, Te, MgO, and SiO_2.

The concentrate from Waterberg will have negligible chromitite but the other elements could cause penalties to be applied for deleterious elements, but this is unlikely.

The concentrate is expected to be a desirable product in the PGE industry as the low chromitite level with the expected high level of S and base metals, allow blending with the forecast increasing UG2 concentrate production (high chromitite content) within South Africa, thus improving the feed composition into the smelting furnaces.

19.11 Pure Metal Sale Agreements

The metal pricing applied to the delivered concentrate for any month is to be based on the arithmetic average 4E 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 4E and base metals. Base metal discounts of US$200 per tonne of Cu and US$100 per tonne of Ni will be applied.

19.12 Material Contracts

No material contracts are in place for the Waterberg Project apart from those related to the JV agreement with Hanwa, JOGMEC, IMPLATS, and PTM.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Waterberg mining right application area, over 20 482 ha in extent, is found at an elevation of approximately 1 000 MASL, is flat lying and sloping gently towards the perennial Mogalakwena River and the non-perennial Seepabana River to the south. The north-south lying Makgabeng Plateau rises up to an elevation in excess of 1 200 MASL through the Waterberg Project area. Climate is temperate to warm with summer rainfall averaging 350 to 400 mm per year allowing for planning for all-round mining without special considerations to allow for weather conditions.

Bushveld vegetation, flora, and fauna predominates with a distinction between the flatlands and the rocky mountainous area. The primary wind direction is from north-northwest.

Settlement pattern is rural, typical of those found in the Limpopo Province. Primary agricultural practices are subsistence farming and grazing for family-owned cattle herds in the flatter lying areas. In consultation with the community, the mine footprint was planned to exclude areas significant to the community including prime grazing areas.

The mineralised rocks dip towards the west at a 34° to 38°. From an environmental perspective, the greatest impacts from mining are anticipated in the eastern (plant footprint) and south-east-central sections where surface infrastructure is planned as this is the shallowest access for underground mining and is topographically relatively flat. The central and western sections, while considered equally by the EIA, should be less significantly impacted. This allows for a number of the assessed potentially significant environmental impacts to be avoided, leaving the primary recommended mitigations for the eastern and central-south-eastern sections by applying appropriate impact management and reduction to reduce the risks.

It is noted, for purposes of clarity, that the application process for environmental permission requires that alternative positions are considered for activities and both the original Scoping Study position of the mine footprint, PFS designs, and the final DFS position were assessed. Amongst other advantages, the newer DFS position negates the relocation of homesteads. In addition, environmental impacts of alternative mining and tailings disposal methods were investigated resulting in the decision to incorporate backfill (using a cemented paste made from tailings) into the mining method. Advantages of this method are improved safety and a reduction in the size of the TSF with the resulting reduction in risk.

20.1 Environmental Issues that could Materially Impact Issuers Ability to Extract Mineral Resources or Mineral Reserves

Waterberg JV Resources has submitted the following key environmental and social licenses and permit applications for the Waterberg Project.

- Mining Right, which includes a SLP.
- EA, which includes the initial environmental scoping study, EIA, and EMPr, environmental financial provision for rehabilitation, and closure plans.
- Integrated WML.
- WUL.
- ‘Consent to development’ from the SAHRA.

The EIA & EMPr in support of the EA and WML application linked to the Mining Right application was submitted to the Competent Authority (i.e. the DMR).

Future applications for EA amendments may need to be submitted for approval by the authorities, due to changes in the nature of the Waterberg Project, approved activities and/or the position of significant activities such as relocating access portals in the mine plan.

In terms of the MPRDA (Act 28 of 2002 as amended), the Minister must grant a prospecting or mining right if, among others, the mining *“will not result in unacceptable pollution, ecological degradation or damage to the environment”*.

The findings of the Environmental Assessment Practitioner and specialists’ assessments completed have shown that the Waterberg Project may result in both negative and positive impacts to the environment, however, adequate mitigation measures are included into the EMPr to reduce the significance of the identified negative impacts. Most negative impacts (classified as minor or moderate) can be reduced through the implementation of mitigation measures.

Following is a list of identified environmental and socio-economic impacts.

- Surface and Groundwater Contamination
- Depletion of Groundwater Reserves
- Alteration of Hydrological Regimes
- Impact on Sensitive Heritage Features, including Graves and Historical Buildings
- Removal of Natural Vegetation and Fragmentation of Habitats
- Faunal Displacement and Mortality
- Dust Emissions
- Soil Contamination and Loss of Soil Resources
- Loss of Agricultural Land

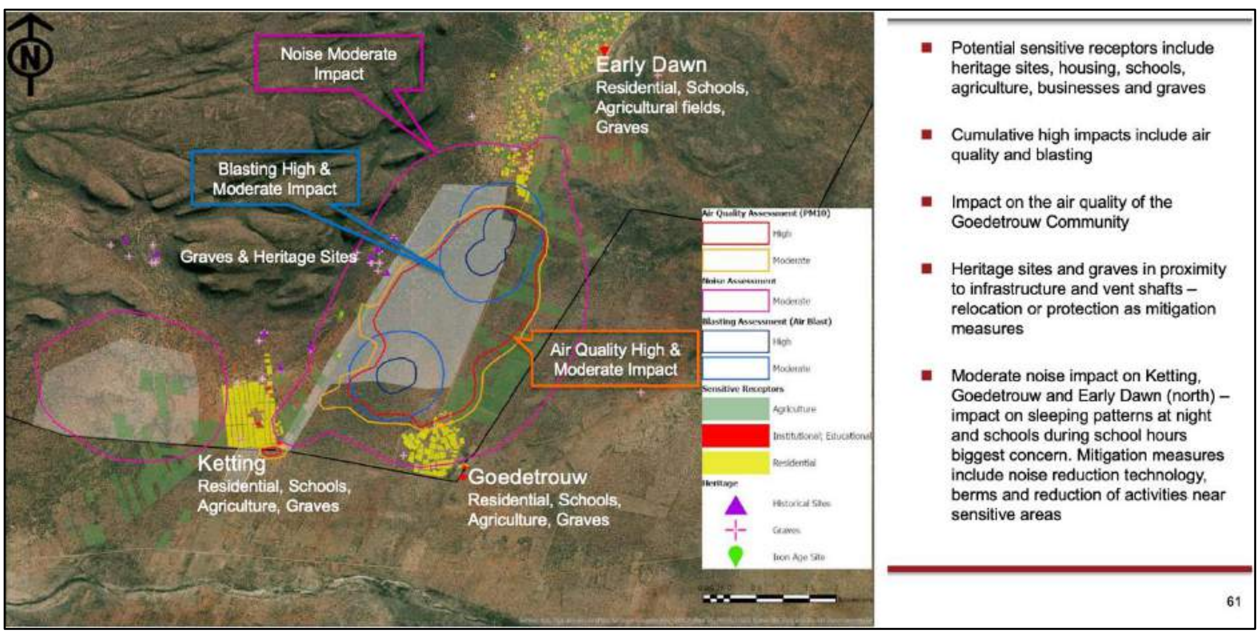
- Land use Conversion (agricultural to mining)
- Noise, Light, and Vibration Nuisance
- Direct and Indirect Job Creation
- Economic Stimulation and Growth
- Community-based Projects, which will Benefit the Local Communities
- Increased Traffic Volumes

The main potential social impacts associated with the Waterberg Project include some economic displacement due to a loss of access to cultivated land or other livelihood resources; influx in job seeking, which, combined with the additional workforce, will place considerable pressure on local infrastructure and services; negative perceptions of project impacts; and increased traffic volumes on roads in the vicinity of the local project area. There are social risks due to the social environment under which the Waterberg Project operates as well as stakeholder fatigue resulting from ongoing mining and exploration activities within the area. Community unrest poses the risk of striking, property destruction, and interruptions of operation schedules. Stakeholder engagement is an ongoing process and a grievance mechanism will be developed to manage stakeholder concerns.

Figure 20-1 provides a visual representation of some of the potential sensitive receptors and impacts relative to the planned mine footprint overlain on a Google Earth image. Figure 20-2 shows the potential impact to groundwater level.

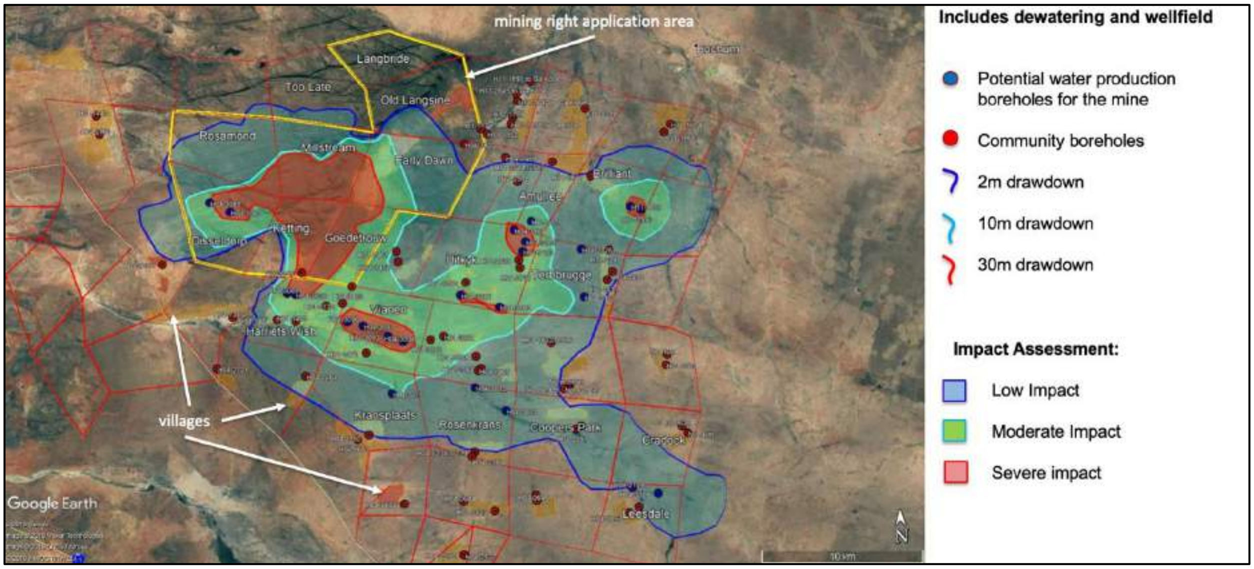
Separate EA applications are also being sought by the project for power and water servitudes.

Figure 20-1: Results of Air Quality, Heritage, Noise and Blasting Studies



Source: Bateleur Environmental, 2019

Figure 20-2: Assessment on Potential Impacts to Groundwater Level



20.2 Requirements and Plans for Waste and Residue Disposal, Site Monitoring, and Water Management, both during Operations and Post Mine Closure

As discussed, applications for a WML (waste) and WUL (water) have been and are to be submitted respectively to the competent authorities. Many of the requirements, including specialist assessments, overlap with the requirements for a mining right and EA and the plans are coalesced. Site monitoring, as well as waste and water management during operations are addressed by the conditions of the EA, WML, and WUL and compliance audits mentioned above.

The EMPr in conjunction with financial provision regulations require that plans for rehabilitation, closure, and latent and residual risk assessments for ongoing impacts post-closure (typically waste and water related) are updated on an annual basis. Financial provisioning will be required to cover these impacts and have been included in the project financial model.

20.3 Project Permitting Requirements

Prior to construction and operation of a mine, the following local legislative authorisations are required.

- A mining right, granted by the Minister of Mineral Resources in terms of Section 23 of the MPRDA, 2002 (Act No. 28 of 2002 as amended) by the DMR is the basic requirement, which must be accompanied by an EA.

- An EA in terms of the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEM Act) read together with the EIA Regulations, 2014 (as amended) and the Financial Provisioning Regulations, 2015 (as amended) from the DMR under the auspices of the Department of Environmental Affairs.
- A WUL in terms of Section 21 of the National Water Act, 1998 (Act No. 36 of 1998) from the Department of Water and Sanitation.
- A WML for categorised waste activities in terms of the National Environmental Management Waste Act, 2008 (Act No. 59 of 2008) from the Competent Authority (i.e. DMR).
- Consent from SAHRA for a new development in terms of the National Heritage Resource Act, 1999 (Act No 25 of 1999).

It is anticipated that the submission of the WUL application will be imminent pending finalisation of water-servitude agreements. Applications for the remaining above-mentioned licenses and permits were submitted to the authorities as shown in Table 20-1.

Table 20-1: Table of Environmental Licenses and Permits for the Waterberg Project

License / Permit Application	Authority	Reference Number
Mining Right	DMR	LP 30/5/1/2/2 /2/10161MR
EA	DMR on behalf of the DEA	LP 30/5/1/2/2 /2/10161EM
WML	DMR on behalf of the DEA	LP 30/5/1/2/2 /2/10161MR
WUL	DWA	Awaiting
Heritage Resources	SAHRA	LP 30/5/1/2/2 /2/10161MR - 12878

The procedure for the EA application is to submit a series of documents in a stage-gated approach. The final stage, the submission of the EIA and EMPr was on 15 August 2019.

An amount for the initial rehabilitation Financial Provision proposed for the trust fund as part of the Mining Right grant process was recommended by the EAP as part of the EA application and amounts to R110 million in July 2019 money terms. This amount is pending agreement by the authorities. The amount is to be revised annually as part of compliance with the mining right. There are a number of approved methods of financing the rehabilitation fund and these are discussed in more detail in Section 22 of this Technical Report.

20.4 Social or Community Related Requirements and Plans

In terms of the provisions of the MPRDA, mineral resources are the common heritage of all the peoples of South Africa hence the Minister of Mineral Resources (the Minister) must ensure the sustainable development of South African's mineral resources whilst promoting economic and social development. The economic and social development requirements are guided by the

Mining Charter / Mining Charter III, which sets out the framework, targets, and timetables for transformation by affecting the entry of HDSA's into the industry and allows South Africans, especially the mine community, to benefit from the exploitation of mining and mineral resources.

The Project's "social license to operate" in South Africa is guided by the Mining Charter / Mining Charter III and regulated by the SLP, which was compiled and submitted as part of the Mining Right Application in August 2018. The SLP is currently being evaluated by the South African regulatory authorities. This process involves negotiation on the finer points of the proposed plan. Legally, the approved document forms part of a granted mining right.

An SLP addresses four required areas for which Waterberg JV have complied.

- Mine Community Development
- Human Resources Development
- Procurement of Goods and Services
- Downscaling and Retrenchments

The SLP is a "living document" and is revised every few years. The final requirement attains greater significance at the end of the mine life. The third requires an ethical undertaking to preferentially use South African and locally acquired goods and services to support and benefit the community. The first two have monetary undertakings, which are included in the DFS financial model.

The SLP is a commitment to sustainable social development and incorporates plans for human resources (skills) development, employment equity, mine community development (including local economic development) housing and living conditions, and eventual downscaling. It seeks to uplift and create opportunities for the community within which the mine operates.

Following are the Waterberg Project's proposed local economic development projects to be approved by the DMR.

- Provision of Infrastructure and Educational Support to Local Schools
- Mine and Community Bulk Water Supply and Reticulation
- Extension and Equipping of Existing Clinic / Health Facility
- Construction of a Creche and Pre-school
- Support to Local Small, Medium and Micro-Sized Enterprises
- Road Construction

The Waterberg Project could represent an alternative economic environment for the community currently deriving a livelihood from subsistence farming in an area with low rainfall or having to travel to find skilled work.

20.5 Status of Negotiations or Agreements with Local Communities

Local landowners, land users, and communities were consulted and updated with respect to the Waterberg Project from the prospecting stage and are aware of the project plans. Land use agreements are currently being concluded with the Goedetrouw Community, the Ketting Community, and individual property owners on the farms traversed by the proposed water pipeline and powerlines.

20.6 House Strategy for Employees

The housing strategy was compiled by Waterberg Project to give effect to Section 100 (1) (a) of the MPRDA; Sections 26(1), (2), and (3), and 27(1),(2), and (3) of the Constitution; the National Housing Act, 1997 (Act No. 107 of 1997); the National Housing Code of 2009 and other related policies and legislation by ensuring that adequate housing, healthcare services, balanced nutrition, and water are adequately provided to mine employees in South Africa.

The purpose of the housing strategy seeks to provide guidelines to the Waterberg Project during operations with regards to the facilitation of suitable housing, accommodation, and related matters to enhance employee well-being, and through this process, to contribute towards the achievement of the overall business objectives of Waterberg Project. The strategy aims to achieve the following goals.

- Achieve a collaborative relationship with government to accelerate housing delivery among Waterberg Project’s labour sending areas.
- Identify and support employees to access low-cost housing rental stock.
- Promote and facilitate home ownership.
- Promote other forms of tenure for employees and contractors who do not wish to own homes in neighbouring communities.
- Introduce debt consolidation as a catalyst to home ownership for credit defaulters who have shown keen interest in our programme.
- Address infrastructure deficiencies collaboratively with government.
- Secure additional land and funding options.

Following are key principles identified in guiding this strategy during LOM.

- The Waterberg Project’s core business should remain that of mining / processing and not the provision of accommodation.
- The strategy is to assist its employees in becoming homeowners.

- The strategy will be aligned with its recruitment, remuneration, and local economic development programmes at the operational level to ensure a holistic approach to this issue during the life of the various operations and facilitate sustainable solutions beyond the mine's life.
- To recognize the Waterberg Project's business plan and the projected workforce requirements in enough time for effective planning mechanisms to be implemented.
- The strategy will endeavour to facilitate the prevention of informal settlement in the areas of operation.

20.7 Training Analysis and Strategy

20.7.1 Labour and Education Level

Local communities will benefit from a portable skills development and training strategic approach. This has a long-term effect in increasing employee's marketability providing for increased sustained employability, which creates opportunity to enhance economic spinoffs in the communities. The training analysis completed by NORCAT for the Waterberg Project allows the operation to focus on the specific skills required to meet production targets. The training strategy incorporates a staged approach to employment and skills training through operational-specific learning pathways, with accredited qualifications and programmes by recognized training providers under the Mining Qualifications Authority (MQA) Sector Education Training Authority.

The Waterberg Project is located in the southern portion of the Blouberg Municipality of the Capricorn District Municipality, Limpopo province. According to the most recent census, the Blouberg Local Municipality has a population of 172 601, of which 10 231 are unemployed and 5 198 are discouraged work seekers (Census 2011. Statistics South Africa. 2012). There are 186 primary schools, 84 secondary schools, and 1 institute of higher education (the Senwabarwana campus of the Capricorn FET College) (Thutse, 2019). Table 20-2 shows the Blouberg Municipality education levels.

Table 20-2: Blouberg Municipality Education Levels

Education	Male	Female	Total
Completed Primary or Less	2 742	2 979	**5 721**
Some Secondary	7 636	9 077	**16 713**
Grade 12	3 286	4 793	**8 079**
Higher Education	618	960	**1 578**

Labour within the Waterberg Project will fall into three categories: contractor trained local workers, national workers, and expatriate workers. A contractual obligation will be established for the contractor to hire a predetermined number of locally-sourced entry-level miners and facilitate integration within the construction activities, after which they will transfer into operations during ramp-up and commissioning. This will reduce the training time and investment of long-term operations. In addition, specifying the manufacturer and model for mining equipment during the construction phase to ensure alignment with steady-state Owner equipment requirements will result in direct transferable skills and a smooth transition during handover.

Assumptions were made around workforce composition – ratio of experienced operators (nationals) to mine trainees from the local talent pool. The following ratios will be applied within the Waterberg Project.

- Low-skilled roles – 4 locals and 1 national.
- High-skilled roles – 1 local and 4 nationals.
- Specialized roles – highly skilled South African workers or interim international expatriates will be used for specialized roles such as Lead Miner – Jumbo Operator, ITH – Longhole Driller, Lead Miner – Bolter Operator.

20.7.2 Human Capital Strategy

The mechanized mining approach to the Waterberg Project and shift to automated processes and solutions will translate into new employment opportunities, enabling women to enter and remain in the workforce. Specific emphasis on the mechanized mining skills of employees will be needed to build capacity and support a mechanized mining learning culture. It will be crucial to have champions with mechanized mining experience in critical roles, specifically mining equipment maintainers, development and production drill operators to drive the process and mentor trainees.

Recognizing that mechanized mining in the region is in a transitional state, NORCAT has exercised its extensive experience and expertise to develop sophisticated learning pathways to ensure that training results in productive, effective, and safe workers, while aligning the completion of qualifications with MQA standards in an efficient manner. Applying this line of progression system is an important element of the Waterberg Project’s human capital strategy, as it will not only foster workforce development, but also enhance retention and cultivate an effective workforce capable of achieving future growth and success.

Career development for novice miners and maintainers in the operations phase will be centred primarily around e-Learning, classroom, simulation training, and on-the-job experiential learning through a structured progression plan for mine operations roles. Leveraging the right mix of training methods and technology will benefit the Waterberg Project in maximizing the transfer of skills during ramp-up training, but also ongoing operator and maintainer upskill training and proficiency building into steady-state operations. A simulator with key training cabs, specifically LHD, haul truck, jumbo drill, and mechanized rock bolter cabs were identified in the training strategy and budget.

Operators and maintainers brought in at an entry-level will be given practical training and work alongside more experienced operators and maintainers to build competency and confidence in specific work areas and equipment.

20.7.3 Operational Readiness and Ramp-up

As a result of the analysis of human capital data and activities, an integrative, adaptive and strategic training tool was developed that includes a training inventory and training matrix indicating the training units that will be required by the various roles within the Waterberg Project.

Role-specific strategies have been developed to ensure operational readiness. This includes cross-functional strategies for increasing equipment availability and improving advancement, resulting in significant production benefits. A modular approach will be used for curriculum design which will simplify training development and cross-functional implementation as each module develops and builds skills, familiarization and knowledge.

During commissioning, stationary and mobile equipment suppliers will deliver training to core operators and maintainers on the full range of operating parameters under all normal and emergency scenarios. From ramp up and into steady-state operations, optimization activities will be captured and training curriculum updated for training of subsequent operators and maintainers.

20.7.4 Estimated Training Schedule

During the construction phase, job readiness programming in general education will be provided to approximately 500 local candidates over a 4-year period to supply a pipeline for local recruitment. Skills training will then take place over a 2-year period from the construction phase into ramp-up and commissioning. A total of 12 months of training time per trainee is allocated for skills training of approximately 347 local trainees, and 3 months of training time per trainee is allocated for skills training of approximately 644 national trainees.

20.8 Mine Closure Requirements and Costs

Closure and rehabilitation are a continuous series of activities that begin with planning prior to the project's design and construction and ends with achievement of long-term site stability that creates a safe, physically stable rehabilitated landscape that limits long-term erosion potential and environmental degradation and restores the land to pre-mining conditions as far as possible.

As the Waterberg Project is an underground mine, there will be no concurrent rehabilitation apart from a provision for vegetating or cladding the TSF. Final rehabilitation will be carried out once the Waterberg Project goes into its closure phase. This final rehabilitation will be completed within the context of the closure plan. Structures will be removed or repurposed for community use, mine access declines will be safely closed off and the TSF is anticipated to remain and be rehabilitated.

Closure cost estimates for LOM are built into the financial model. Included within this estimate is a financial provision amount, discussed in Section 20.3, which is paid to the authorities as part of the EA and mining right.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimate was prepared with an accuracy range of -10% to +15% (Class 2 estimate as defined by the American Association of Cost Engineers). The estimate is expressed in ZAR. Where applicable, costs obtained in other currencies were converted to ZAR using a fixed rate of exchange based on R15 to US$1 and other approved exchange rates as applicable.

The following cost classifications were applied to the Waterberg Project.

21.1.1 Project Capital Costs

Project Capital Costs are from the start of the project in January 2020 until 70% of planned steady-state underground production is achieved in December 2025, including the operating costs which will be capitalized during this period. On surface, this includes all off-site and on-site infrastructure and equipment, including the processing plant. For underground this includes excavations, infrastructure, equipment, and initial stoping during production ramp-up.

21.1.2 Sustaining Capital Costs

After the Project Capital Cost period, Sustaining Capital Costs start in January 2026 and end in 2063. For both surface and underground, Sustaining Capital Costs include infrastructure extension and mobile and fixed plant equipment rebuilds and/or replacement required to maintain steady-state production.

21.1.3 Operating Costs

After the Project Capital Cost period, Operating Costs start in January 2026 and continue to the end of the mine life in 2066. For surface, Operating Costs include all on-site costs, including the processing plant. For underground, Operating Costs include excavations from the footwall infrastructure to access the stopes and all stoping activities (including backfilling).

21.1.4 Definition – Project, Sustaining, and Operating Cost

The split between Project Capital and Sustaining Capital Costs is shown in Figure 21-1.

Figure 21-1: Project Definitions

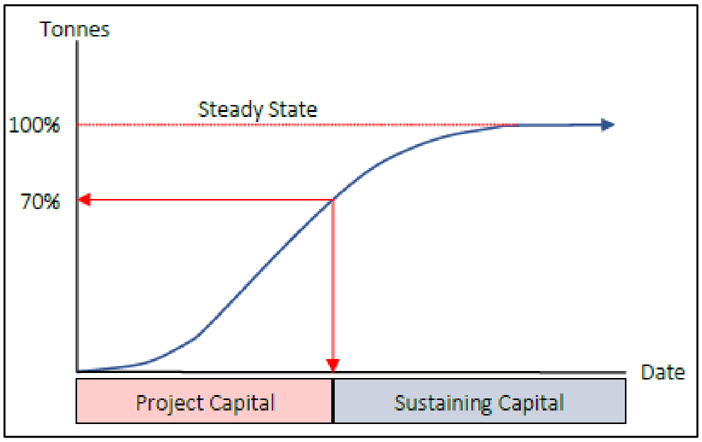
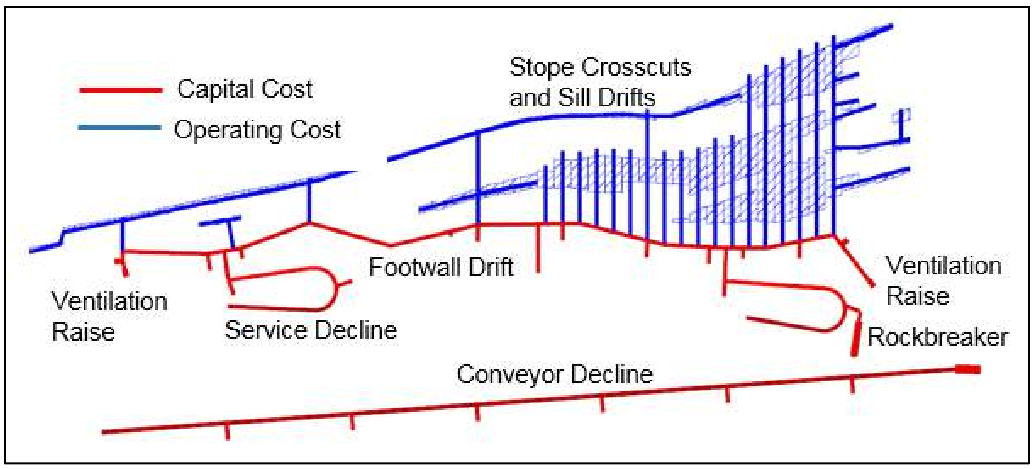


Figure 21-2 provides a visual representation of the definitions applied to typical underground sublevel infrastructure, which details the split between sustaining capital development and operating cost production.

Figure 21-2: Underground Development Capital and Operating Cost Footprint



21.2 Capital Cost Estimate Summary

21.2.1 Capital Costs

The capital costs total R38 176 M over the LOM, including R16 559 M of Project Capital and R21 617 M of Sustaining Capital. Capital costs are stated in real terms base dated 01 July 2019 without escalation. The capital cost breakdown is presented in Table 21-1.

Table 21-1: Capital Cost Breakdown

Facility Description	Project Capital (ZAR M)	Sustaining Capital (ZAR M)	Project Capital (US$ M)	Sustaining Capital (US$ M)
Central and South Complex	6 280	10 072	419	671
North Complex	0	10 286	0	686
Concentrator Plant	3 060	846	204	56
TSF	315	165	21	11
Backfill Plant	448	0	30	0
Shared Services	424	53	28	4
Access Roads	195	0	13	0
132 kV Supply	380	40	25	3
Bulk Water Supply	196	0	13	0
Preproduction Costs	125	47	8	3
Owners Team Cost	384	0	22	0
Subtotal	**11 807**	**21 510**	**784**	**1 434**
Contingency	1 298	42	87	3
Other Capitalised Costs	3 453	65	234	4
Total	**16 559**	**21 617**	**1 104**	**1 441**

21.2.2 Basis of Capital Estimate

The capital costs include the expenditure required for the following activities.

- Engineering and design.
- Procurement.
- Underground development.
- Fabrication, delivery, and erection on site of equipment and supporting steelwork and civil work.
- Commissioning.

The estimate also includes the following indirect costs.

- Owners’ team.
- Insurance.
- Social and labour development.
- Training.
- Engineering, procurement, and construction management (EPCM).

- Funding for rehabilitation.
- Contingency.

The resultant scope of this estimate covers the total cost of mine development, bulk earthworks, civil works, mechanical work, structural steelwork, piping, electrical work, control and instrumentation, reimbursable costs for professional services, Owner's cost and other project overhead costs.

21.2.3 Scope of Capital Costs

The following activities define the scope of the Waterberg Project capital cost estimate.

- Development of three box cuts with twin declines and underground workings to access the Central Complex, South Complex, and North Complex.
- Construction and commissioning of the ventilation and mine air refrigeration infrastructure.
- Underground mobile and fixed equipment
- Construction of workshops, stores, offices, stormwater management, and other infrastructure to support the mining operation grouped in the South Complex as described in Section 18.
- Construction and commissioning of a 400 ktpm Concentrator Plant as described in Section 17.
- Construction and commissioning of a backfill plant as described in Section 18.
- Construction and commissioning of a TSF as described in Section 18.
- Construction and commissioning of local and regional infrastructure, including the bulk earthworks, 132 kV electrical supply, 11 kV electrical reticulation, bulk water supply, on site water distribution, and road upgrades as described in Section 18.
- Provision for preproduction costs including surface vehicles, spares and initial fills of lubricants, reagents, and grinding media.
- Other capitalised costs that include operating costs incurred during the project period.
- Owner team costs, including Owner's management team, insurances, site security, and SLP commitments.

21.2.4 Sustaining Capital Costs

Sustaining capital costs include the following.

- Additional capital equipment required to ramp up to full production.
- Ongoing capital development into new production areas to sustain production and extension of mine infrastructure and services to the new production areas.
- Capital rebuild and replacement of equipment required to sustain full production.

The sustaining capital includes for rebuild and replacement cost for the mobile equipment fleet. Fleet rebuild and replacement costs were calculated based on when operating hours reach the specified intervals.

Fleet refurbishments and replacement philosophies as well as utilizations and availabilities to derive operating hours were provided by the engineering team. Quotes were supplied by various original equipment manufacturers.

Sustaining capital for the plant and surface infrastructure was determined as a factor of the mechanical and electrical equipment costs.

All capital development, equipment purchases, and infrastructure construction (surface and underground) costs required to access and develop the North Complex is included in sustaining capital.

21.2.5 Capitalised Operating Cost

Capitalised operating costs are derived similarly to OpEx, which is detailed in Section 21.10.1. Capitalised OpEx is defined as operating costs that occur during the project capital period (ending December 2025) and processing 5.14 Mt of ore until 70% of steady-state production is achieved on a monthly basis. The revenue generated during this period is not capitalised but is included in the financial model.

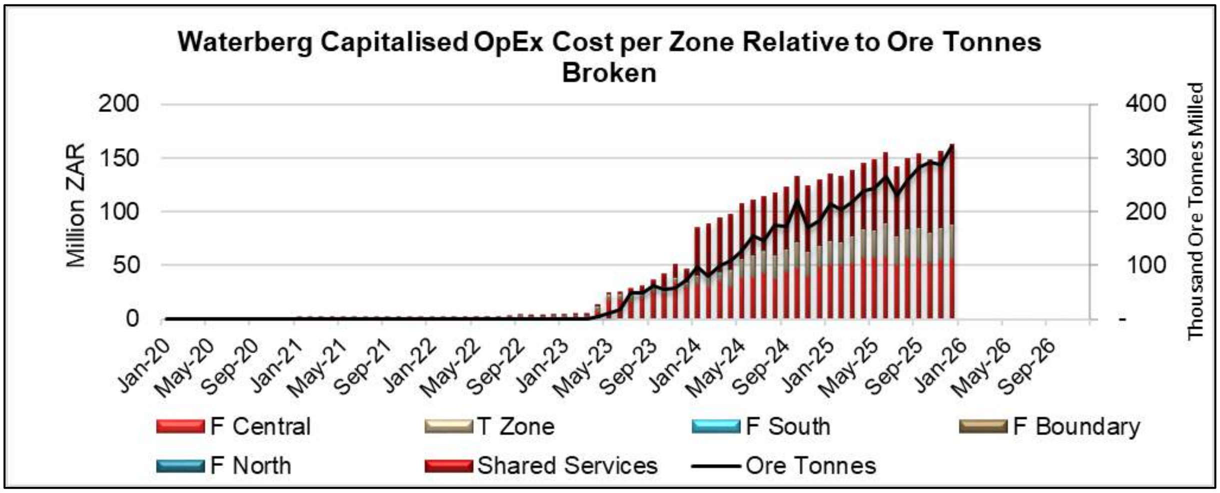
The total capitalised operating cost for the Waterberg DFS is estimated at R3.453 billion (R671.86 per tonne milled) and are detailed per area in Table 21-2.

Figure 21-3 presents the capitalised operating costs over the period with the ore tonnage profile and the cost profile closely follows that of production. The cost increase observed in October 2024 and June 2025 is directly related to production tonnage increases during the respective periods.

Table 21-2: Capitalised Operating Cost to December 2025

Area	Cost Category	LOM ZAR (M)	LoM ZAR / Ore Tonne Milled
Mining	Materials and Supplies	R936	R182.13
Mining	Labour	R496	R96.54
Mining	Fixed Overheads	R-	R-
Mining	External Services	R-	R-
Mining	Utilities	R249	R48.50
Engineering and Infrastructure	Materials and Supplies	R372	R72.29
Engineering and Infrastructure	Labour	R92	R17.84
Engineering and Infrastructure	Fixed Overheads	R-	R-
Engineering and Infrastructure	External Services	R114	R22.28
Engineering and Infrastructure	Utilities	R277	R53.89
G&A	Materials and Supplies	R8	R1.54
G&A	Labour	R67	R12.97
G&A	Fixed Overheads	R-	R-
G&A	External Services	R4	R0.87
G&A	Utilities	R8	R3.58
Process	Materials and Supplies	R425	R82.68
Process	Labour	R125	R24.33
Process	Fixed Overheads	R-	R-
Process	External Services	R-	R-
Process	Utilities	R269	R52.41
Total		**R3 453**	**R671.86**

Figure 21-3: Capitalised Operating Cost per Zone to end December 2025



The cost observed starting in June 2020 is related to camp and construction utilities while all power costs are costed in the operating cost model.

Figure 21-4 provides a graphical presentation of the cost breakdown per area in Table 21-2.

Figure 21-4: Average R/t Capitalised Operating Cost Breakdown per Area

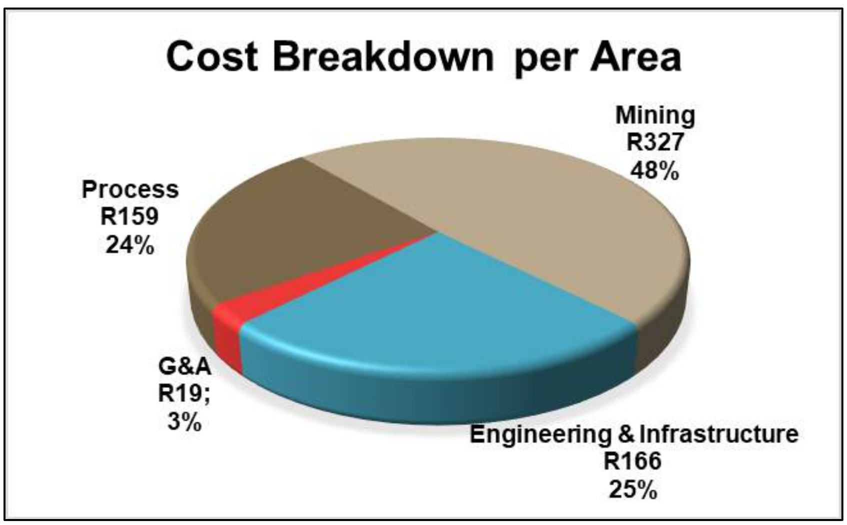
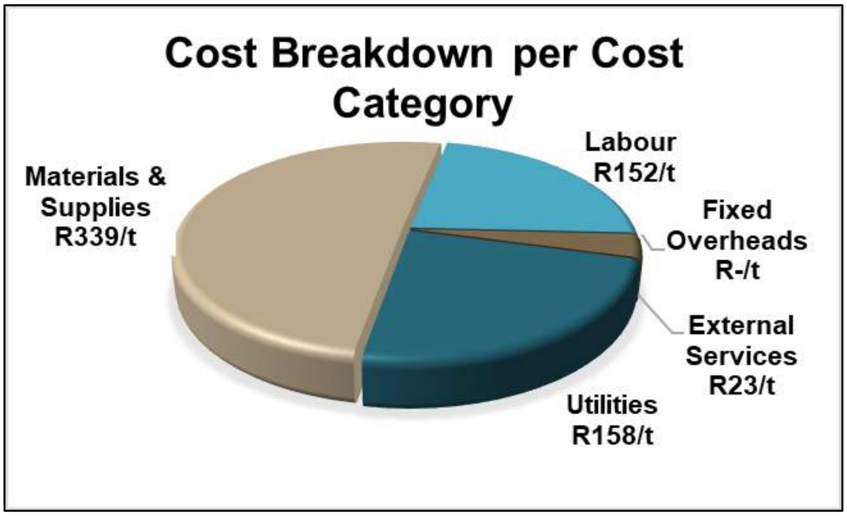


Figure 21-4 shows that mining costs comprise the bulk of the capitalised OpEx cost at 48%. This cost is largely driven by materials and supplies directly associated to production, ore development, and stope crosscut development and amounts to approximately R1 000 M (R182 per tonnes milled). The cost of materials and supplies for process and infrastructure amounts to R83 and R72 per tonne milled in relation to mining.

The remainder of the capitalised operating cost is made up of labour and utilities across the different areas as displayed in Figure 21-5 with mining labour comprising the greater part.

Figure 21-5: Average R/t Capitalised Operating Cost Breakdown per Cost Category



21.2.6 Exclusions from Capital Estimate

The following items were excluded from the capital cost estimate.

- Foreign exchange rate variations.
- Escalation beyond estimate base date of 01 July 2019.
- Duties and taxes on imported goods and services.
- Delay costs for permitting (e.g. excavation permits, confined space permits etc.) beyond what is reasonably expected.
- 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.

21.2.7 Direct Field Costs

Direct costs include the permanent facilities and services required for installation, including plant and equipment, bulk material, contractor / subcontractor costs, freight, and vendor representatives. These items are explained further below.

- Plant and Equipment include the mechanical, electrical, and instrumentation components of a plant that are either shop assembled, modularized, or preassembled on site.
- Bulk Materials are materials such as rebar, piping, cables, and light steel that are purchased based on quantity.
- Installation refers to the labour and contractor costs to install the plant equipment and bulk materials.

- Contractor costs cover construction equipment and other support required to support and deploy installation labour. Following are the cost components covered by these rates.
  - 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 preoperational testing of equipment, including mobilisation / demobilisation of the representative and any special tools.

21.2.8 Indirect Costs

Indirect costs are the costs associated with supporting the purchase and installation of the direct costs. These costs include the materials and services required for field construction, that 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 21-3 reflects all the indirect cost for the Waterberg Project.

Table 21-3: Indirect Costs

Subheading	Total Cost (ZAR M)	Total Cost (US$ M)
Temporary Power Supply	R20.0	$1.33
Utilities	R1.5	$0.10
Construction Water Supply and Reticulation	R1.3	$0.09
Site Security	R54.2	$3.61
Preproduction Vehicles	R19.2	$1.28
Initial Fills, Spares and Inventories	R135.9	$9.06
EPCM Fees	R600.4	$40.03
Owners Management Team – Home Office	R53.1	$3.54
Drilling	R32.5	$2.17
SLP	R9.0	$0.60
Water Servitude Leases	R16.0	$1.07
Community Agreements	R16.4	$1.09
Training	R135.7	$9.05
Accommodation Camp	R39.7	$2.65
Insurance	R28.3	$1.89
Land Purchases / Lease	R2.3	$0.15
Total	**R1 165.5**	**$77.7**

Site support services are inclusive of temporary construction camps, labour, security, utilities, supplies, and power to operate the site during the construction phase as well as for plant commissioning and spares.

Cost for EPCM are based on estimates from consultants and Owner’s team. Other capitalized costs, including drilling, environmental closure, and land leasing were provided by PTM.

21.3 Mining Capital Costs

Mining capital costs amount to R25 208 million. Table 21-4 provides a breakdown of the capital cost per facility.

Table 21-4: Total Life-of-Mine Mining Capital Cost Breakdown per Cost Category

Mining Cost per Zone	Total LOM (ZAR Million)	Total LOM (US$ Million)
Portal South Complex	R467	$31
Portal Central Complex	R798	$53
Portal North Complex	R746	$50
T Zone	R4 425	$295
F Central Zone	R6 701	$447
F Boundary North and South	R3 008	$201
F North Zone	R6 104	$407
F South Zone	R1 546	$103
Site Support Services	R1 293	$86
Project Delivery Management	R120	$8
Total	**R25 208**	**$1 681**

21.3.1 Underground Mining Contractor Costs

A mining contractor will complete all underground development, construction, and commissioning during the Capital Project period. All raiseboring and diamond drilling will be completed by contractors for the life-of-mine.

The underground mining contractor costs include the following elements.

- Contractor Direct Labor
- Contractor Indirects, Overhead, and Markup
- Permanent Materials
- Direct Charge Equipment
- Equipment Operating Costs
- Service and Supplies
- Equipment Rental

21.3.2 Contractor Direct Costs

Contractor labor costs and typical crew rotation and buildup information were received from a South African mining contractor. Detailed overtime and Sunday work premiums were provided and used to calculate the composite labour rates based on the specified shift cycles. The Contractor's labor rate schedule includes the following elements.

- Wages
- Overtime Allowance

- Absentee Allowance
- Payroll Burden
- Work Premiums
- Vacation Benefits
- Site Allowances

21.3.3 Contractor Indirect Costs

Mining contractor indirect labour costs and plant rental costs, including mark-up were provided by a South African Mining Contractor. Indirect labor includes the following job classifications.

- Management Staff
- Administrative Staff
- Supervisory Staff
- Maintenance and Support Personnel
- Technical Services Support
- Corporate Overhead

21.3.4 Contractor Overhead and Markup

The contractor's overhead and markups were quoted at 20% and were included in the contractor direct and indirect costs.

21.3.5 Hours of Work

The mine will operate 24 hours per day and 7 days per week. The staffing basis will be 2 10.5-hour shifts per day. There will be three crews on rotation and scheduled production is 365 days per year.

21.3.6 Contractor-to-Owner Labor Transition

A hard finish of contractor crews is scheduled for completion of the project capital phase at the end of 2025 and a hard start for Owner crews is in January 2026. It is anticipated a portion of the contractor labour will transition to the Owner's team, which would result in a low negative impact transition to the operating phase.

21.3.7 Equipment

21.3.7.1 Mobile Equipment Fleet

The mobile equipment fleet is based on specific work activities per the mine schedule as discussed in Section 16.6.

21.3.7.2 Fixed Equipment

Major fixed equipment (e.g., main ventilation fans, main dewatering pumps, shop equipment, electrical motor control centres) is based on the mechanical equipment list. Vendor budgetary quotes were provided for all equipment. Multiple quotes were received where possible. Minor fixed equipment (e.g., auxiliary fans, face pumps, safety equipment) is based on the mechanical equipment list and costs are based on vendor budgetary quotes and recent Stantec project experience or allowances.

21.3.7.3 Initial Fleet

Direct and Indirect

Contractor development activities will be supported by major mobile equipment leased by the Owner and used by the contractor with auxiliary gear provided by the contractor and a rental fee charged to the Owner. Owner mobile equipment will be directly purchased by the Owner in the time period required following leasing agreement conclusion and/or end of contractor rentals. During preproduction, the ramp-up of Owner crews will utilize the excess equipment operating hour capacity of contractor equipment on site in lieu of buying dedicated equipment, which would otherwise result in low utilization across the entire fleet.

Rebuild and Replacement

Initial and sustaining capital mobile equipment leasing costs, acquisition costs, rebuild costs, and replacement costs were calculated based on the operating hours of an individual piece of equipment during its useful life. Equipment life was vendor-provided as part of the budget quotation requests.

Table 21-5 lists mobile equipment types with typical rebuild / replacement hours, based on engine hours. Replacement hours start following the rebuild completion. The sum of both rebuild and replacement hours is the equipment total life.

Table 21-5: Mobile Equipment Operating Hours

Development and Production Equipment	Operating Hours Prior to Rebuild	Operating Hours Prior to Replacement
2-Boom Jumbo	6 000	3 600
Mechanical Bolter	10 000	6 000
Cable Bolter	10 000	6 000
LHD – 17-t	18 000	10 800
Haul Truck – 50-t	25 000	15 000
Haul Truck – 40-t	25 000	15 000
Explosives Loading Truck	25 000	15 000
Shotcrete Sprayer	20 000	12 000

Mobile Equipment Operating Costs

Stantec calculated the equipment operating costs from first principles, which includes the following items.

- Diesel Fuel
- Lubricants
- Operating Parts
- Tires
- Ground Engagement Components / Wear Parts (excludes drilling bits and steel)

These costs do not include equipment rental or rebuilds. Maintenance labor is captured in the indirect labor costs.

21.3.8 Development

Development CapEx can be divided into labour, materials, and equipment operating costs. Materials and supplies comprise most of the development unit costs. Costing was derived from zero-based costing by combining relevant metre drivers with rates for drilling, blasting, mucking, and ground support installation. Mining rates used for development are listed in Table 21-6.

Performances applied to the multiple face development headings are detailed in Table 21-7.

Table 21-6: Contractor Development Rates

Rate Type	Rate (R/m)
5 m x 5 m Heading Waste Development	R22 228
5.5 m x 5 m Heading Waste Development	R24 163
5 m x 4 m Heading Ore Development	R18 720
6 m x 5 m Heading Ore Development	R24 227

Table 21-7: Contractor Development Rates

Rate Type	Rate (m/day)
5 m x 5 m Heading Waste Development	6.2
5.5 m x 5 m Heading Waste Development	5.8
5 m x 4 m Heading Ore Development	7.2
6 m x 5 m Heading Ore Development	5.8

21.3.9 Mass Excavation

Mass excavation performance rates were developed based on general arrangement drawings for these types of facilities. Considerations for extra ground support, multiple excavation cuts, as well as increased attention to decrease overbreak apply to the performance rate. Mass excavation in this project includes main workshops, satellite shops, explosives storage, rock breaker stations and conveyor transfer stations.

21.3.10 Vertical Development

Vertical development will be completed by raiseboring and costs were provided for unsupported raises by a South African mining contractor. Where required, Stantec estimated additional costs for ground support (i.e., for example in raises used for egress or poor ground condition areas).

21.3.11 Waste Haulage

Haulage costs include truck and LHD labour and equipment operating and account for the initial LHD truck loading and where applicable the subsequent re-handing with an LHD. Waste will be hauled to one of three locations, including to underground stopes, rock breakers to batch feed conveyors when not conveying ore, or hauled directly to surface. Distances from truck loading areas to the various dumping locations were estimated to establish haulage tonnage performances and costs which vary by zone and by activity.

21.3.12 Construction

Construction activities during the project period will be executed by the contractor as project capital investments in mining infrastructure. During the operating period, the owner will assume responsibility for the construction activities as a sustaining capital investment. All construction costs were developed based on detailed quantity take-offs, by facility, to which commodity costs were then applied.

21.3.13 Maintenance

Maintenance activities related to mobile fleet, fixed plant equipment, mining infrastructure and underground upkeep will be performed by the contractor during the project period. Labour requirements for contractor maintenance have been assessed based on the demands of the tasks to be executed. The contractor will provide maintenance supervision and planning and will coordinate with owner's team maintenance management personnel. Maintenance handover to the owner's team, who will assume responsibility during the operating period, will occur in the final quarter of 2025.

21.4 Concentrator Plant Capital

21.4.1 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, including compressed air, service water, potable water reticulation, return water columns, and electrical supply and reticulation from the plant consumer substation.

Plant infrastructure includes stormwater berms and drains to divert rainwater from within the plant to 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, store, workshop, and weighbridge are included to support plant operations.

21.4.2 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 Section 17.

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, and the required instrumentation and control systems. The estimate made provision for indirect costs, including EPCM, maintenance support vehicles, first fills of consumables, and 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 a material takeoff (MTO).

MTOs were completed for the structural steel, platework, and electrical and civil engineering disciplines. Costs for the fabrication and erection of structural steel and platework, as well as the construction of the civil engineering works were estimated by applying rates received from South African contractors to these quantities.

The cost of the electrical equipment, instrumentation, and the installation of this equipment was derived from the DRA South Africa database rates to an MTO completed for this engineering discipline.

The costs for in-plant piping were determined by factorization. Overland piping was estimated from measurements taken from the site plan.

Preliminary and general (P&G) costs for site establishment, ongoing site management, and supervision, various items of plant, transport and accommodation of labour, and costs for human resources functions were provided for the main contractors.

Provisions were 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 were included in the CapEx based on factoring the equipment estimates.

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 Class 2 estimate.

21.4.3 Estimating Assumptions

In preparing the processing plant capital estimates, the following assumptions were applied.

- The project will be executed using an EPCM project execution strategy.
- The construction activities of each phase will be completed in a continuous program.
- Fill material for earthworks, G5 or higher quality, is available from borrow pits within a 5 km radius of the site. The source of the borrow pits must be confirmed before detail design phase starts.

- All concrete will be 25 MPa at full strength.
- The civils contractor's rates are inclusive of supply of all materials. No materials are free-issued.
- The concrete batch plant will be established at site and adequate aggregate will be available within 80 km 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 was verified with suppliers.
- Construction work pricing based on unit price rates.
- The supplied budgetary quotes for major equipment and materials are within the required accuracies.
- 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.
- Plant commissioning based on experienced operations team involvement and includes training of operators.

21.4.4 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 crushed ore stockpile.
- Electricity is received as an 11 kV supply at the incomer of the consumer substation.
- Plant tailings are pumped to the fence/boundary of the TSF or the backfill plant.
- Return water is received at the suction of the return water pumps at the RWD.
- Concentrate is dispatched from the filter building by truck.

21.4.5 Exclusions from Concentrator Costs

The following costs are excluded from the process plant capital estimate.

- 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 that 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 preproduction costs not specified in the preproduction 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.

21.4.6 Concentrator Plant Cost

The cost breakdown for the Concentrator Plant is presented in Table 21-8.

Table 21-8: Concentrator Plant Cost Breakdown by Discipline

Discipline	Cost (ZAR Millions)	Cost USD (Millions)
Earthworks and General Services	R93.5	$6.23
Packages by Others	R21.0	$1.40
Civils	R181.9	$12.13
Buildings	R98.7	$6.58
Structural Steel	R278.5	$18.57
Platework and Mechanicals	R978.3	$65.22
Electrical Control and Instrumentation	R490.7	$32.72
Piping and Valves	R362.8	$24.19
Transport	R74.5	$4.97
Sub-Total	**R2 580.0**	**$172.00**
Preproduction Expenses, including EPCM, Spares and Preproduction Costs	R479.5	$31.97
Total	**R3 059.5**	**$203.97**

21.5 Paste Backfill Plant Capital

The paste backfill plant cost estimate was prepared by SSBS.

21.5.1 Scope of Estimate and Methodology

Capital estimates for the paste backfill plant are based on the equipment and structures described in Section 18.

The capital cost estimate methodology involved identifying each cost element and compiling a bill of quantity (BOQ). Subsequent requests for quotation were sent to potential suppliers and costs were assigned to each item based on the quotations received.

21.5.2 Accuracy and Basis of Estimate

The cost estimates for the civils and structural steel were measured from the DFS design drawings. Civil, earthworks, concrete, and structural steel rates provided by DRA from the Concentrator Plant were then applied.

The cost estimates for the electrical equipment, components and distribution were prepared by Buhrmann Consulting Engineers and provided to SSBS.

The cost estimate for tanks and platework is based on preliminary tank dimensions. All the platework is based on EN 10025 S355JR material as a minimum.

Quotations were obtained for the supply and delivery of the mechanical equipment to site.

The BOQ for piping, fitting flanges, and gaskets were compiled from the 3D model prepared for the study. The BOQ for valves and instruments were compiled from the piping and instrumentation diagrams. Piping costs are based on quotes received from the market applied to these BOQs.

EPCM and P&G costs are included in the overall estimate.

21.5.3 Backfill Plant Direct Field Cost

The cost breakdown for the backfill plant direct costs is presented in Table 21-9.

Table 21-9: Backfill Plant Direct Cost Breakdown

Subheading	Total Cost (ZAR Million)	Total Cost (US$ Million)
Civils and Earthworks	R16.1	$1.07
Concrete	R12.2	$0.81
Structural Steel	R24.3	$2.03
Platework and Liners	R12.1	$0.81
Mechanical Equipment	R219.9	$14.66
Piping and Valves	R23.5	$1.57
Electrical	R50.0	$3.33
Control and Instrumentation	R7.0	$0.47
Total	**R364.8**	**$24.32**

21.6 Infrastructure Capital

This section covers the shared and regional infrastructure for the Waterberg Project inclusive of bulk power, water supply, TD, and access roads; however, it excludes the specific concentrator infrastructure covered above.

21.6.1 Tailings Storage Facility

The TSF estimation was completed by Epoch.

The estimated capital costs associated with the construction of the preparatory works of the TSF as described in Section 18 were compiled and are based on a schedule of quantities describing the work completed. The construction rates for the GCL used were determined by WBHO Construction (Pty) Ltd. and Aquatan for Phase 1 and applied to the subsequent phases.

The estimated Project Capital cost associated with the construction of the TSF is R315 M with an additional R165 M in sustaining capital, which will include the expansion of the TSF lining area and wall lifts in four phases until 2030.

21.6.2 132 kV Electrical Supply

The estimate for the 132 kV supply line was completed by TDx Power as described in Section 18.

Following are the items included in the scope.

- One 132 kV line feeder bay at Eskom's 400/132 kV Burotho transmission substation.
- One 132 kV overhead line (74 km line length) from Eskom's 400/132 kV Burotho transmission substation to the Eskom 132 kV switching station and from the switching station to mine 132/11 kV substation (further 3 km line length).
- Eskom 132 kV switching substation on boundary of Goedetrouw property.
- Waterberg 132/11 kV distribution substation comprising a single 132 kV busbar, one incoming 132 kV feeder bay, and four 40 MVA 132/11 kV transformer bays.

Following are the items excluded from the scope.

- The 11 kV main consumer substation.
- Standby generator equipment, which is provided by others.
- Power factor correction equipment.
- The 11 kV and control cables to connect the 132/11 kV transformer feeders to the 11 kV indoor switchgear.
- Earthwork terraces for the substation and switching station.

21.6.3 Shared Services and Surface Infrastructure

The estimate for site infrastructure was compiled by DRA based on general arrangement drawings and layouts. Quantities were measured from these drawings and priced based on rated from tenders received from the market.

21.6.3.1 Bulk Earthworks, Roads, and Terraces

Bulk earthworks quantities are based on preliminary drawings for terraces. Bulk earthworks rates are based on contract rates obtained from the market. No survey info was available for a large section of the access road. Google Earth contours were used for the road alignment and quantity takeoffs. Detailed surveys are required before detailed design phase starts. Waste rock dump type D liner is measured.

- Main access road alignment is per the route identified as optimal in the traffic study.
- DRA assumed that fill material for earthworks (G5 or higher quality) will be available from borrow pits within a 5 km radius of the site. The source of the borrow pits must be confirmed before the detail design phase starts.
- The rates for excavations include a free haul distance of 2 km.
- Provisions for blasting of hard rock are made depending on the location of the respective structures and available geotechnical information.

21.6.3.2 Concrete Work

Concrete work rates are based on contracts received from the market for the Waterberg Project and applied to the MTO derived from the preliminary drawings.

21.6.3.3 Brick Buildings

The building works quantities are estimated from the block plan and general arrangement drawings by DRA Cost Engineers. The estimated quantities were used to produce the BOQ. Items such as air conditioners, electrical lights, small power, hot water generation, and furniture are included as provisional sums.

Rates were received from the market for the Waterberg Project and applied to the BOQ to create the estimate.

The contractor’s unit rates are all inclusive for supplying fuel and operating and maintaining the equipment.

P&G costs assume that the contractor will supply and install all materials, including steelwork identified in the BOQ.

21.6.3.4 Structural Steelwork

Rates for structural steelwork are based on contracts received from the market for the Waterberg Project and applied to the MTOs derived from the preliminary drawings.

21.6.3.5 Security and Fencing

Security costs for capital installation of security infrastructure were obtained from a security provider. Rates used for fencing are based on rates obtained from current contract rates applied to measurement made from the site layouts. The cost of security services during the construction period is included in the capital estimate under preproduction costs.

21.6.3.6 Potable Water

Rates for the potable water treatment plant and piping are based on recent quotes obtained by DRA and the rates applied to the MTO.

21.6.3.7 Sewerage

Sewer water reticulation quantities are based on preliminary layouts. The treatment plant and piping rates are based on recent quotes obtained by DRA.

21.6.3.8 Preliminary and General

P&G costs used in the estimate are based on the rates obtained from the issued tenders. Costs were determined by applying various percentages for the various disciplines.

21.6.4 Primary Crushing

Direct costs associated with the installation of the primary crusher and feed conveyors are included with surface infrastructure and costed on the same basis as the Concentrator Plant.

21.6.5 Summary of Infrastructure Costs

The costs associated with the infrastructure are shown in Table 21-10.

Table 21-10: Surface Infrastructure Costs

Cost Centre	Cost (ZAR Millions)	Cost USD (Millions)
Access Road	195	13
132 kV Supply	380	25
Bulk Water Supply	196	13
Surface Infrastructure		
Storm Water Management	41	3
Earthworks	79	5
Buildings	205	14
Conveyors and ROM Materials Handling	107	7
Waste Rock Materials Handling	176	12
Water Systems and Sewerage	90	6
Laboratory	41	3
Electrical Reticulation	357	24
South Portal Primary Crusher	21	1.4
Fencing	7	0.5
Total	**1 895**	**126**

21.7 Contingency Assessment

The contingency in the capital model was assessed by conducting a qualitative assessment, the assessment considered the level of engineering undertaken, accuracy of the rates, and quantities applied to the estimate for their scope of work. These assessments were undertaken by the all the contributors to the estimate and then combined to form the contingency allocation in the estimate.

The underlying rationale supporting development of the contingency amount is based on capturing risk and uncertainty arising from the following items.

- 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, Cu, energy prices; unit price-based changes to equipment supply, etc.)].
- Excludes foreign exchange variations.

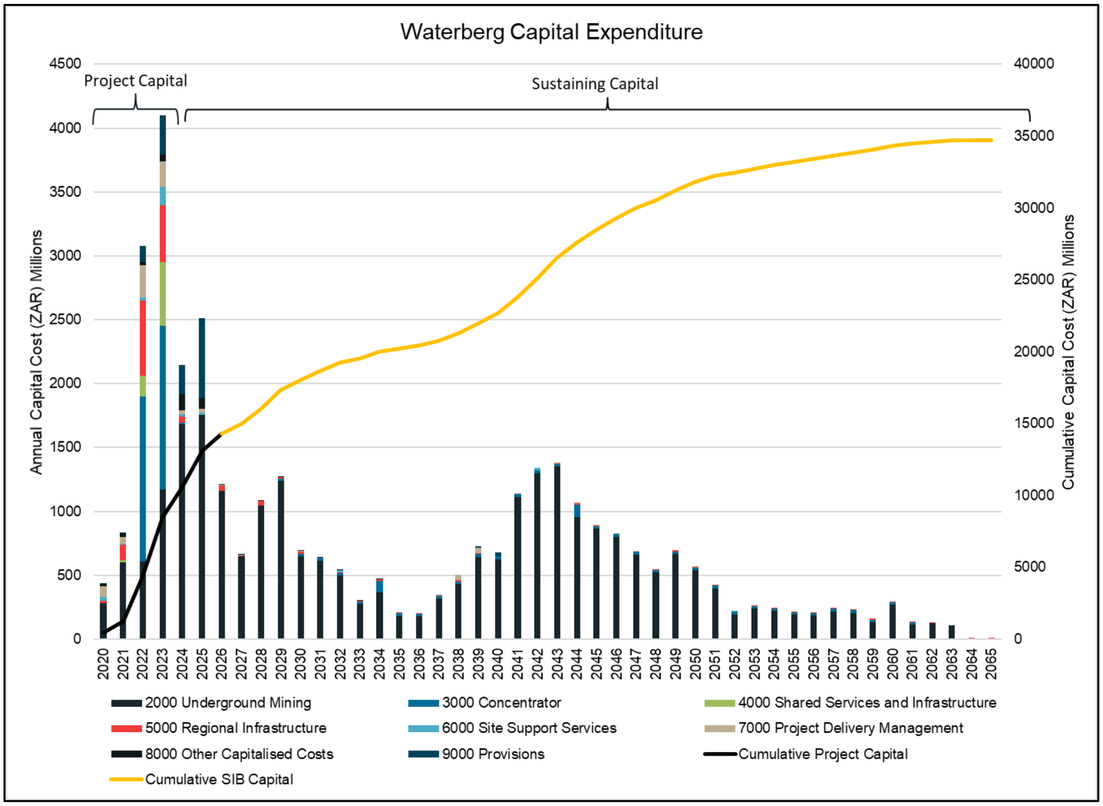
There is no contingency applied on mining costs following the project capital period. Additionally, there is no contingency for refurbishment and replacement costs. The contingency applies to risks specific to estimating accuracy. Risks that could not necessarily be quantified such as schedule delays arising from labour disputes are not covered by the contingency allowed.

The contingency allowed is 11.03% of the estimated capital cost.

21.8 Capital Expenditure Profile

The CapEx excluding the capitalized operating cost for the Waterberg Project is demonstrated graphically in Figure 21-6.

Figure 21-6: Waterberg Capital Expenditure Over Time



21.9 Project Implementation

The project objective is to complete the design, construction, commissioning, and ramp-up to 70% of the steady-state production rate of the Waterberg Project.

The project schedule was determined by assessing the project information such as the mine production schedule, engineering design data, supplier lead times, and the construction schedules. The project critical path was determined be to be the design of surface infrastructure, portal construction, decline development, lateral development, and ramp-up to full production of the underground mining operation.

The Waterberg Project is to be executed as an integrated programme consisting of three main projects (listed below) to be executed at different points in time.

- The design and development of the mine and supporting infrastructure.
- The design and construction of the 132 kV power supply to the project site.
- The design and construction of the concentrator, backfill plant, TSF, and regional and local infrastructure.

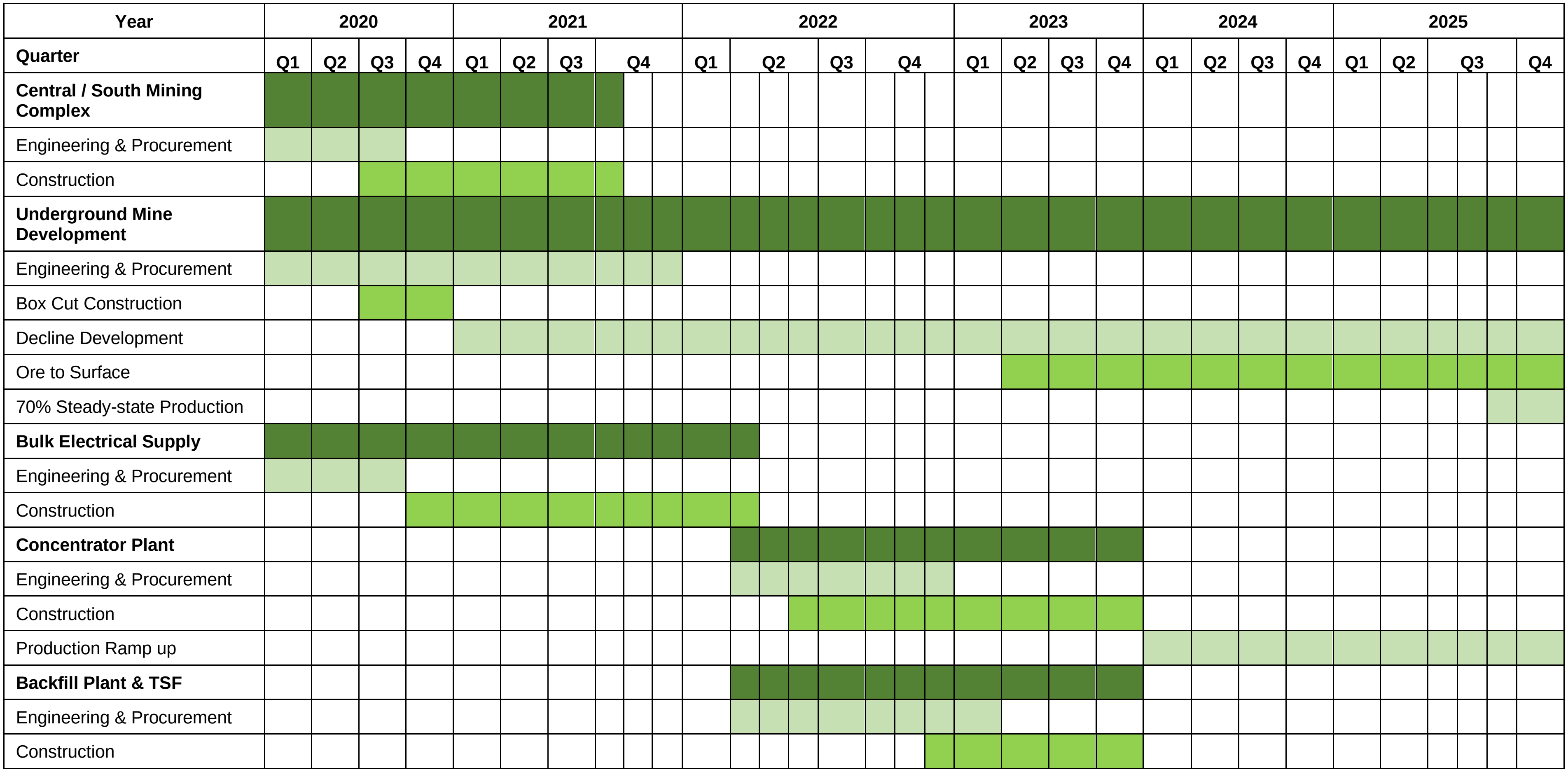
The project programme assumes a start date of January 2020, with the first activity, following the Project Execution decision by the Waterberg JV, being the commencement of the detailed engineering. The programme aims to achieve the integration of the projects by achieving the following key milestones.

- Start of Project – January 2020
- Start of Construction of Central / South Complex – June 2020
- Start of Decline development – January 2021
- Completion of the 132 kV Bulk Electrical Supply – April 2022
- Start of Ore Processing in Concentrator – January 2024
- Achievement of 70% of Steady-state Capacity – September 2025
- Completion of Capital Period – December 2025

The production ramp-up will continue until steady state is reached December 2026.

The project schedule is summarised graphically in Figure 21-7.

Figure 21-7: High-level Implementation Schedule



The development of the North Portal and the supporting infrastructure would be undertaken as a separate sustaining capital project commencing in 2038.

To facilitate the control of the project an execution WBS aligned to the intended execution strategy was developed that groups the project into the work packages described in Table 21-11.

Table 21-11: Work Packages

Work Package	Description
WP1	Construction of Mining Complex surface infrastructure
WP2	Underground Mine Development
WP3	Bulk Electrical Supply
WP4	Concentrator Plant and Plant Infrastructure
WP5	Tailing Storage Facility
WP 6	Backfill Plant
WP 7	Bulk Water Supply
WP 8	Main Access Road
WP 9	Construction Services
WP 10	Construction Camp

The scope of the initial project (WP1) will include the engineering and construction of earthwork terraces and portal box cuts for the South and Central Mines, including the surface infrastructure required to support the mining development.

The development of the underground workings (WP2) will be completed by a mining contractor. The selected mining contractor will complete all underground development, construction, and commissioning during the project period. Towards the end of the project period, the underground operations will start to transition to an Owner-operated model. All raiseboring and diamond drilling will be completed by contractors for the LOM.

The 132 kV electrical supply project (WP3) is assumed to continue from the work related to environmental authorizations and ESKOM commercial process that are already in progress, so that construction can be run in parallel with the initial projects to provide a 132 kV power supply to site by the end of 2022.

The remaining work (WP4-WP8) is planned to start in January of 2022, the scope of which will be the Concentrator Plant, Backfill Plant, TSF, and regional and local infrastructure, such as the roads, bulk water supply, and 11 kV reticulation required for the operation of the plants and the mine workings.

The development and maintenance of construction support services, camp, construction power, and construction water supply (WP9-WP10) will continue from the start of the project until the end in December 2025.

21.10 Operating Cost Summary

21.10.1 Basis of Estimate

An OpEx model was developed to consolidate surface and underground operating costs. Various methodologies were utilised to derive costs, including first-principle costing for the labour; lifecycle costing for fleet, equipment, and infrastructure; and zero-based costing for mining and consumables. The model was built up from either fixed or variable unit cost rates multiplied with appropriate cost drivers. Drivers were mostly production schedule related. In some cases, outputs from the fleet model, labour model, or LOM absolute costs were provided by consultants who performed the engineering calculations to substantiate the provided output. Operating costs occurring during the Project Capital period, will be capitalised. The consolidated OpEx model details operating costs and capitalised costs over the LOM on separate worksheets.

A base date of 01 July 2019 was used as the costing basis. Costs were reported in real money terms with no escalations or contingency modelled.

The OpEx model is on a monthly, quarterly, and annual basis corresponding to the timeline of the production schedule. Reporting areas include per zone, area, and cost category. Figure 21-8 details the zones, areas, and cost categories.

Figure 21-8: Operating Expenses per Zone, Area, and Cost Category

Per Zone	Area	Cost Categories
• T Zone	• Mining	• Labour
• F-South	• Process	• Power
• F-Central	• Engineering and Infrastructure	• Materials and Supplies
• F-North	• G&A	• External Services
• F-Boundary		• Fixed Overheads
• Shared Services		

All costs not associated with a mining zone were reported under shared services and include general, administrative, and processing costs. The operating estimate is further aligned to the project work breakdown structure (WBS).

- 2000 Underground Mining
- 3000 Process Plant
- 4000 Shared Services and Infrastructure
- 5000 Regional Infrastructure
- 6000 Site Support Services

21.10.2 Model Results

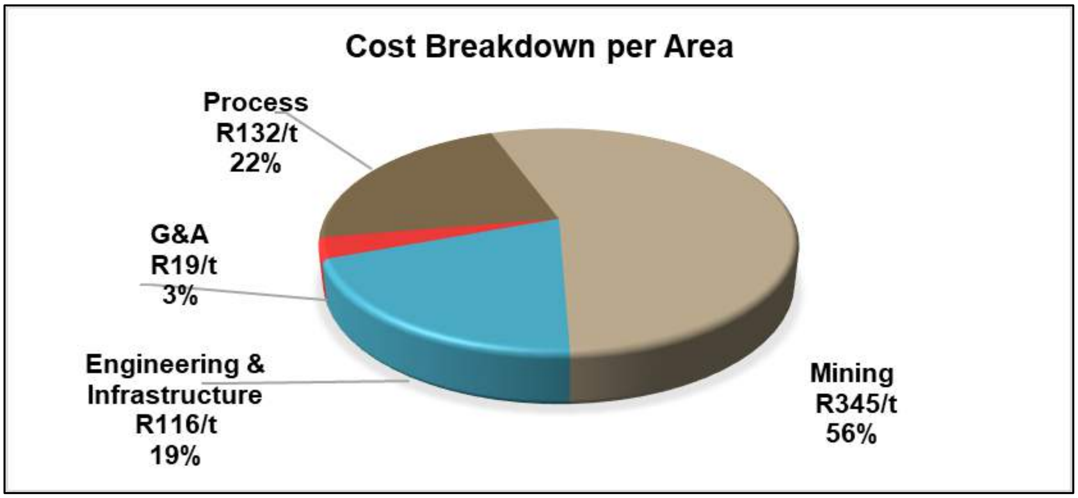
21.10.2.1 Results Overview

The total estimated LOM Operating Costs are R111.6 billion (US$7.4 billion) averaging R612 per ore tonne milled (US$40.80/t) as summarized in Table 21-12 and Figure 21-9.

Table 21-12: Average Life-of-Mine Operating Cost Rates and Totals per Area in ZAR and US$

Area	Average LOM (ZAR / Ore Tonne Milled)	Average LOM (US$ / Ore Tonne Milled)
Mining	R345.10	$23.01
Engineering and Infrastructure	R116.36	$7.76
General and Administrative (G&A)	R18.75	$1.25
Process	R 131.78	$8.79
Total OpEx Cost	**R 612.00**	**$40.80**

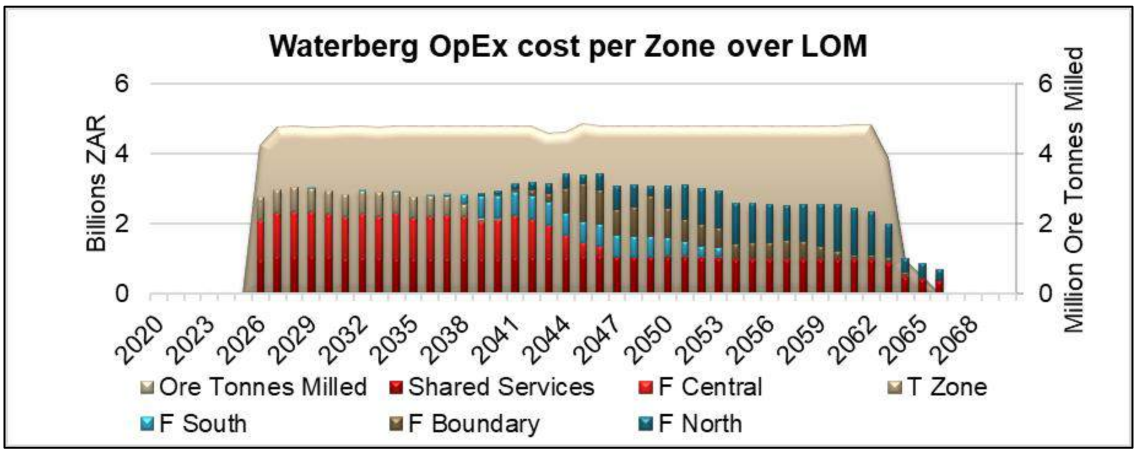
Figure 21-9: Life-of-Mine Average ZAR per Tonne Operating Cost Breakdown per Area



Mining comprises the bulk of the operating costs at 56%, followed by process at 22%, and engineering and infrastructure at 19%. G&A costs represent a small portion (3%) of the total operating costs.

Figure 21-10 presents the total operating costs over the LOM overlaid with the ore tonnage profile. The cost increase observed in 2042 is due to starting up the North Complex. Steady state is observed in 2031 when the process plant will process 4.8 Mtpa. The process, G&A, and engineering and infrastructure operating costs remain constant throughout the LOM, while the mining operating cost closely resembles the tonnage profile. The ramp down starting in Year 2061 is clearly visible towards the end of LOM. The dip in operating cost displayed in Year 2064 is a result of reduced power and materials / supplies associated to the reduced tonnage processed by the plant.

Figure 21-10: Operating Cost per Zone over the Life of Mine Relative to Ore Tonnes



The operating cost model was developed to enable reporting per zone, per area, and per cost category.

21.10.2.2 Results per Mining Zone and Area

Table 21-13 presents the total operating cost per zone and area, of which shared services comprises the bulk at 35%.

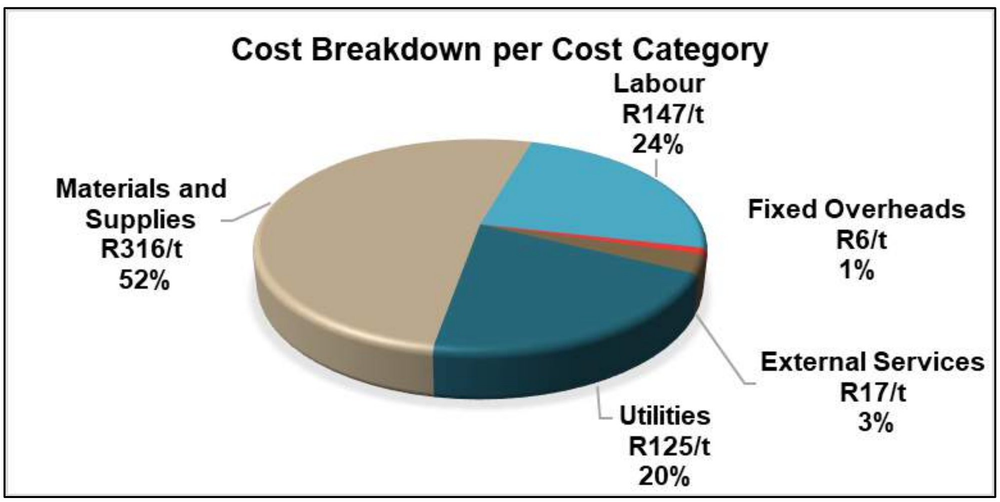
Table 21-13: Summary of Total Life-of-Mine OpEx Cost per Mining Zone and Area

Area	T Zone	F-South	F-Central	F-Boundary	F-North	Shared Services	Total
	Average LOM (ZAR / t)	Average LOM (ZAR / t)	Average LOM (ZAR / t)	Average LOM (ZAR / t)	Average LOM (ZAR / t)	Average LOM (ZAR / t)	Average LOM (ZAR / t)
Mining	R472.08	R518.11	R296.88	R338.29	R320.00	R0.32	**R345.10**
Engineering and Infrastructure	R55.71	R43.01	R50.56	45.02	R52.60	R66.37	**R116.36**
G&A	R0	R0	R0	R0	R0	R18.75	**R18.75**
Process	R0	R0	R0	R0	R0	R131.80	**R131.78**
Total OpEx Cost	**R527.79**	**R561.12**	**R347.43**	**R383.31**	**R372.60**	**R217.24**	**R612.00**

21.10.2.3 Results per Cost Category

Various cost categories used to further detail the operating costs include materials and supplies, labour, utilities, fixed overheads, and external services. Figure 21-11 provides an overview of the cost breakdown per cost category for the total LOM average operating cost.

Figure 21-11: Life-of-Mine Average ZAR per Tonne Operating Cost Breakdown per Cost Category



Materials and supplies constitute the bulk at 52% of the total cost followed by labour at 24% and utilities at 20%.

Materials and Supplies

Materials and supplies comprise operating consumables, maintenance consumables, and spares as listed below.

- Mining Consumables and Spares
  - Explosives
  - Drilling
  - Support
- Process Consumables and Spares
  - Grinding Media
  - Reagents
  - Crushing and Mill Liners
  - Maintenance Consumables and Spares
- Surface / Underground Fleet (Mobile Equipment) Consumables, Maintenance, and Spares
  - Fuel
  - Lubrication
  - Tires
  - Maintenance
  - Ground Engagement Tools
- General Consumables
  - Office Consumables
  - Exploration Drilling Consumables
- Surface / Underground Fixed Equipment Consumables, Maintenance, and Spares
  - Backfill Binder
  - Backfill Maintenance Consumables and Spares
  - Cooling Plant Maintenance Consumables and Spares

Mining materials and supplies comprise more than half of the total LOM materials and supplies cost of R316 per tonne milled. Mining materials and supply cost is driven by production consumables such as drilling, explosives, support, fleet fuel, tires, and maintenance. Refer to the Section 21.10.3 for the basis of estimate. The breakdown of the total operating cost per area is provided in Table 21-14.

Table 21-14: Total Life-of-Mine Materials and Supplies Cost Breakdown per Area

Area	Average LOM (ZAR / t)
Mining	R183.53
Engineering and Infrastructure	R63.05
G&A	R2.63
Process	R67.27
Total Materials and Supplies OpEx Cost	**R316.48**

Labour

Labour costs constitute 24% of the total operating cost at R26.9 billion over LOM. Figure 21-12 provides the total Owner's labour cost over LOM or R147 per tonne milled.

Figure 21-12: Annualised Life-of-Mine Owner's Labour Costs

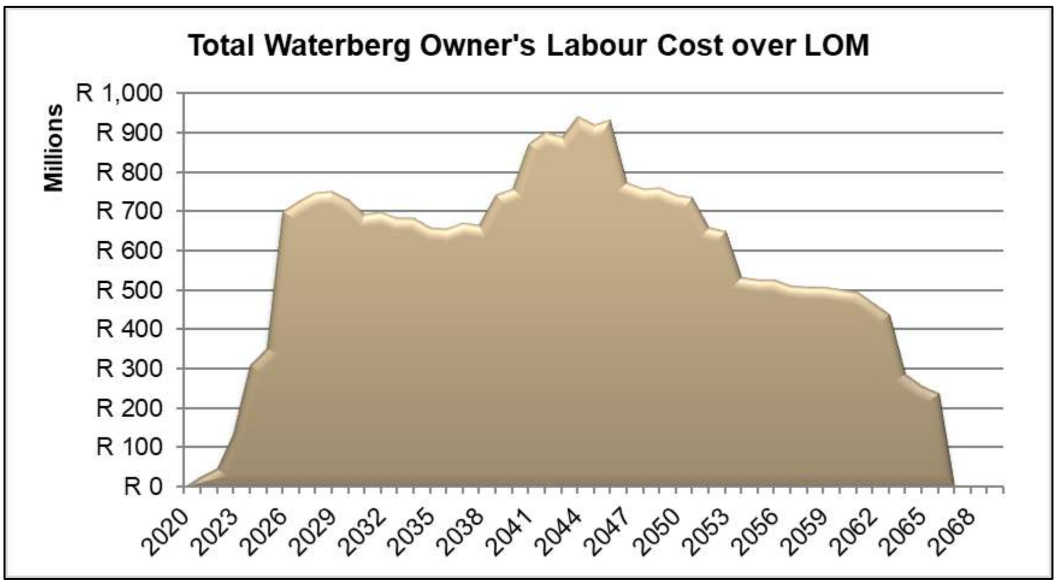


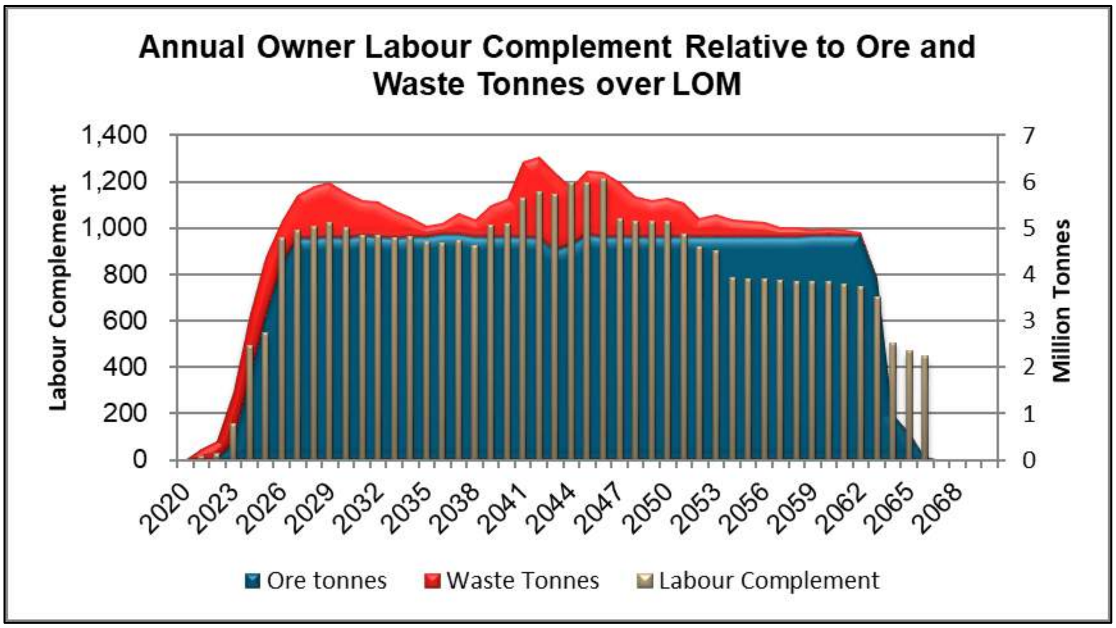
Table 21-15 shows that mining labour makes up the bulk (82%) of the total labour cost.

Table 21-15: Total Life-of-Mine Labour Operating Cost Breakdown per Area

Area	Average LOM (ZAR / t)
Mining	R120.63
Engineering and Infrastructure	R8.39
G&A	R6.26
Process	R12.10
Total Labour OpEx Cost	**R147.39**

The labour complement for the shared services remains relatively constant over LOM as displayed in Figure 21-13. The labour spike observed in 2044 is attributed to the labour requirements associated with the North Complex production tonnes that will occur during that period. A maximum complement of 1 209 can be observed. Figure 21-13 displays the labour complement per complex over LOM relative to ore and waste tonnes.

Figure 21-13: Owner's Labour Complement Relative to Ore and Waste Tonnes



For surface labour, a labour complement for surface infrastructure, G&A, and the process plant was derived. Job descriptions were associated to Patterson grades to derive labour costing for the surface labour complement. The labour rate per grade was based on benchmarked total cost to company package input data provided by the client project team. The 50th percentile input was used from the data source that included an allowance for housing.

Mining related Owner's labour rates were developed based on analysis of labour rates for the various job classifications provided by a South African mining contractor, which include base hourly rate, overtime allowance, absentee allowance, payroll burden, work premiums, vacation benefits, and site allowances. These rates were then benchmarked against Patterson grades and one of the JV partner's operating mines.

The majority of labour costs were introduced three months prior to the start of the plant to allow for training, induction, and medicals. Management labour was introduced six months earlier than the plant start date.

Utilities

The cost of utilities comprises 20% of the total LOM operating cost at R125 per tonne milled. Table 21-16 shows that approximately 42% of the power cost can be attributed to process.

Table 21-16: Total Life-of-Mine Utilities Operating Cost Breakdown per Area

Area	Average LOM (ZAR / t)
Mining	R40.94
Engineering and Infrastructure	R31.76
G&A	R0.00
Process	R52.41
Total Utilities OpEx Cost	**R125.11**

Water consumption and cost in the OpEx model relates to potable water treatment for two water treatment plants, sewerage treatment, and cooling plant water supply cost. There are no costs associated with bulk water consumption other than power.

The power costs comprise fixed and variable portions. The nominal power cost is derived from estimated consumptions for mobile equipment, mining infrastructure per zone (including cooling plant), process plant, and the backfill plant that reports to the infrastructure area. Load lists defining absorbed power together with power profiles over LOM are utilised to determine power consumption. The fixed power cost portion comprises a services and administrative fee and charges based on calculated mWh, kVA, and KVAhr. Refer to Table 21-17 for the rates. Fixed power costs are shown under engineering and infrastructure along with nominal portions of power consumed associated with engineering and infrastructure (backfill plant) – units consumed are referenced under the particular consumer. Power costs average R0.92 per kWh (total fixed and variable power cost) over the LOM and R125 per tonne milled.

Power cost rates used were based on the 2019 / 2020 Eskom Megaflex tariffs for non-local authority for a transmission distance of 300 km or less and a voltage range between and including 66 kV and 132 kV. Table 21-17 details the Eskom power tariffs used. The active energy charge was calculated based on Eskom. 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 obtain a calculated average power rate of 74.7 c/kWh.

Table 21-17: Eskom Megaflex Tariffs for Non-local Authority (2019 / 2020)

Description	Unit	Amount (Real Cost Rates)
Service Charge	ZAR/day	R217.67
Admin Charge	ZAR/day	R98.10
Total	**ZAR/day**	**R315.77**
Total	**ZAR/Month**	**R9 604.67**
Distribution Network Demand Charge	ZAR/kVA/month	R11.50
Distribution Network Capacity Charge	ZAR/kVA/month	R6.21
Transmission Network Charge	ZAR/kVA/month	R8.49
Urban Low-voltage Charge	ZAR/kVA/month	R15.32
Electrification and Rural Network Subsidy	ZAR/kWh	R0.0848
Affordability Subsidy Charge	ZAR/kWh	R0.0382
Reactive Energy Charge – High Season	ZAR/kVAhr	R0.1534
Ancillary Service Charge	ZAR/kWh	R0.0041
Average Active Energy Charge (Nominal Rate)	ZAR/kWh	R0.747

External Services

The external services cost over LOM amounts to R17 per tonne milled. Due to the mine being Owner operated, very few services impacting operating cost will be contracted; therefore, external services contribute only 3% to the total LOM operating cost. External services included in the estimate include the central laboratory, contracted security services, TSF operation and management, and waste removal. The laboratory costs are based on a quotation from SGS and amount to R52 M per year or R11 per tonne milled. Security services were estimated at R15.6 M per year. An annual TSF operation and management cost of R6.5 M was estimated and compiled by Epoch. Waste removal was calculated by estimating the frequency of trips required to remove domestic, industrial, and medical waste from the site along with cost rates based on travel distance, waste disposal, and service fee estimates.

Fixed Overheads

The fixed overhead cost amounts to R1.1 billion in total, 1% of the total LOM operating cost at R6 per tonne milled. Fixed overhead cost is made up of insurance and leasing costs associated with land and water servitude. Costs were provided by the Waterberg Project team.

The insurance cost is based on current insurance coverage for similar operations. Insurance cost were scaled to the Waterberg Project and indicative premium rates were obtained from insurance brokers. Insurance coverage included in the operating cost estimate amounts to R843 M over LOM and includes the following items.

- Property (including machinery breakdown)
- Business Interruption (including machinery breakdown)
- South African Special Risks Insurance on the above where applicable
- Mobile and mining plant equipment

21.10.3 Mining / Underground Operating Costs

Mining related operating costs total R345/ore tonne milled and account for 56% of the total site operating cost. Table 21-18 provides a breakdown of the mine operating costs.

Table 21-18: Total Life-of-Mine Mining Operating Cost Breakdown per Cost Category

Item	Average LOM (ZAR / t)
Materials and Supplies	R183.53
Labour	R120.63
Utilities	R40.94
Total OpEx Cost	**R345.10**

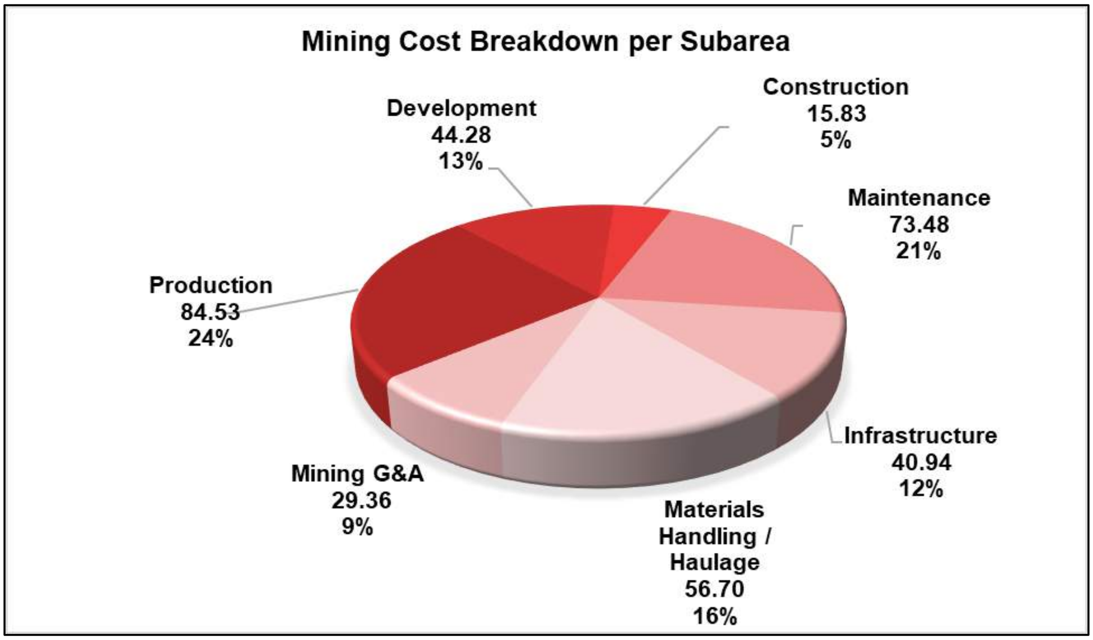
Utilities for mining include power, which was estimated for fixed equipment and infrastructure associated to mining, power related to the surface ventilation and cooling plants, and power for mobile equipment.

Mining operating costs are further detailed into development, production, logistics, construction, maintenance, infrastructure, materials handling / haulage, and G&A as shown in Table 21-19 and Figure 21-14.

Table 21-19: Mining Cost Detail per Subarea and Cost Category

Subarea	Cost Category	Average LOM (ZAR / t)	% of Total Mining Cost
Production	Labour	R20.63	6.0%
	Materials and Supplies	R63.90	18.5%
Development	Labour	R14.77	4.3%
	Materials and Supplies	R29.51	8.6%
Construction	Materials and Supplies	R15.83	4.6%
Maintenance	Labour	R51.71	15%
	Materials and Supplies	R21.77	6.3%
Infrastructure	Utilities	R40.94	11.9%
Materials Handling / Haulage	Labour	R4.16	1.2%
	Materials and Supplies	R52.53	15.2%
Mining G&A	Labour	R29.36	8.5%
Total		**R345.10**	**100%**

Figure 21-14: Mining LOM Average ZAR per Tonne Milled Cost Breakdown



21.10.3.1 Maintenance

Mining maintenance costs include labour and materials and average R73.48/t. Maintenance includes costs associated with ventilation and refrigeration systems, ore handling systems, dewatering, underground infrastructure fixed installations, and all mobile equipment.

21.10.3.2 Stoping

The longhole stoping and stope cable bolting unit rates were developed from first principles for representative stope sizes and include labour, materials, and equipment operating. The stoping unit rates and cable bolting unit rates by stope type are listed in Table 21-20.

Table 21-20: Stoping Unit Rates

Stope Type	Rate (ZAR per Stope Tonne)
Stope 21 m Thick Transverse 40 m High	R54.88
Stope 21 m Thick Transverse 20 m High	R52.09
Stope 48 m Thick Transverse 40 m High	R52.73
Stope 48 m Thick Transverse 20 m High	R49.49
Stope 3 m Thick Longitudinal 40 m High	R74.59
Stope 3 m Thick Longitudinal 20 m High	R68.54
Stope 8 m Thick Longitudinal 40 m High	R48.34
Stope 8 m Thick Longitudinal 20 m High	R44.33
Cable Bolt 8 m Thick Longitudinal Stope	R6.47
Cable Bolt 3 m Thick Longitudinal Stope	R15.23
Cable Bolt 21 m Thick Transverse Stope	R3.18
Cable Bolt 48 m Thick Transverse Stope	R2.29

Operating and maintenance consumables for primary production fleet include items such as fuel, lubrication, tires, trailing cable, hydraulic hose, ground engagement tools, maintenance consumables, and spare parts. Fleet operating costs are derived through lifecycle costing methodologies aided by the original equipment manufacturer operating metrics and costing along with utilisations and availabilities based on estimated cycle times.

Mining production labour cost averages R20.63 per ore tonnes milled over LOM.

21.10.3.3 Development

Stope crosscuts and ore sill development is included as an operating cost. Development OpEx is broken into material, supplies, and labour. Costing for mining material and supplies are derived from zero-based costing by combining relevant meter drivers with rates for drilling, blasting, mucking, and ground support installation.

21.10.3.4 Materials Handling / Haulage

Materials handling / haulage comprises materials, supplies, and labour totaling R56.70 per tonne ore milled. The cost is made up of maintenance, operating consumable costs and labour associated with truck haulage, rock breakers, and conveying.

21.10.3.5 Construction

Construction operating costs include material for backfill barricades and installing services in operating development headings. The labour component is captured in the maintenance labour costs. For construction, materials and supplies only entail consumables for the underground construction support fleet and amounts to R15.83 per ore tonnes milled as shown in Table 21-19.

21.10.3.6 Infrastructure

The nominal power consumption for underground is included under mining infrastructure and is costed per kWhr based on the power consumption for fixed underground equipment such as conveyors, pumps, and ventilation fans, the surface ventilation and refrigeration plants, and underground mobile equipment such as jumbos, mechanical bolters, and production drills.

21.10.3.7 Mining General and Administrative

Mining G&A costs comprise mine engineering, geology, safety, and mining management labour for the respective zones.

21.10.3.8 Skills Development and Training

Training cost estimates were formulated for trainee labour, trainers, training curriculum development, partnership engagement, learning technologies, training simulation hardware and software, and overall training management. These estimates were made as a result of the training needs analysis and comparable benchmarks from previous NORCAT experience while incorporating South African context and data. Table 21-21 shows the ramp-up training budget estimate.

Table 21-21: Ramp-up Training Budget Estimate

Category	2020	2021	2022	2023	2024	2025
Curriculum Development	R20 002 216	R20 002 216	R15 652 937			
Training Technology Hardware / Software				R18 574 793	R100 000	R100 000
General Education	R6 250 000	R6 250 000	R6 250 000	R6 250 000		
Skills Training					R32 134 876	R125 709 360
Total	**R26 252 216**	**R26 252 216**	**R21 902 937**	**R24 824 793**	**R32 234 876**	**R125 809 360**

During steady-state operations, the annual training budget includes costs estimates for curriculum updates, training technology maintenance support, and for continued training initiatives of operational training, cross-functional training, and upskill training based on 2% of the wage bill. Table 21-22 shows the steady-state training budget estimate.

Table 21-22: Steady-state Training Budget Estimate

Category	Annual Budget (ZAR)
Curriculum Updates	R2 782 868
Skills Training (2% of wage bill)	R15 011 815
Training Technology Maintenance Support	R100 000
Total	**R17 894 683**

21.10.4 Plant and Shared Infrastructure Operating Cost Estimates

21.10.4.1 Basis of Operating Cost Estimate

This operating cost estimate is applicable to the steady-state operation of a single 400 ktpm module.

This estimate is supported by the testwork conducted as part of the PFS and DFS (as outlined in Section 13) and engineering input (as per Section 17 and 18). The plant operating costs were based on costs from Q2 2019 and calculated in ZAR.

The process plant LOM operating cost was calculated as R131.78/t milled and excludes concentrate transport to Rustenburg area.

The pie chart in Figure 21-15 provides a breakdown of the process cost per subarea.

Utilities comprising mainly power makes up the bulk of the process costs followed by reagent consumables.

Figure 21-15: Process Breakdown per Subarea

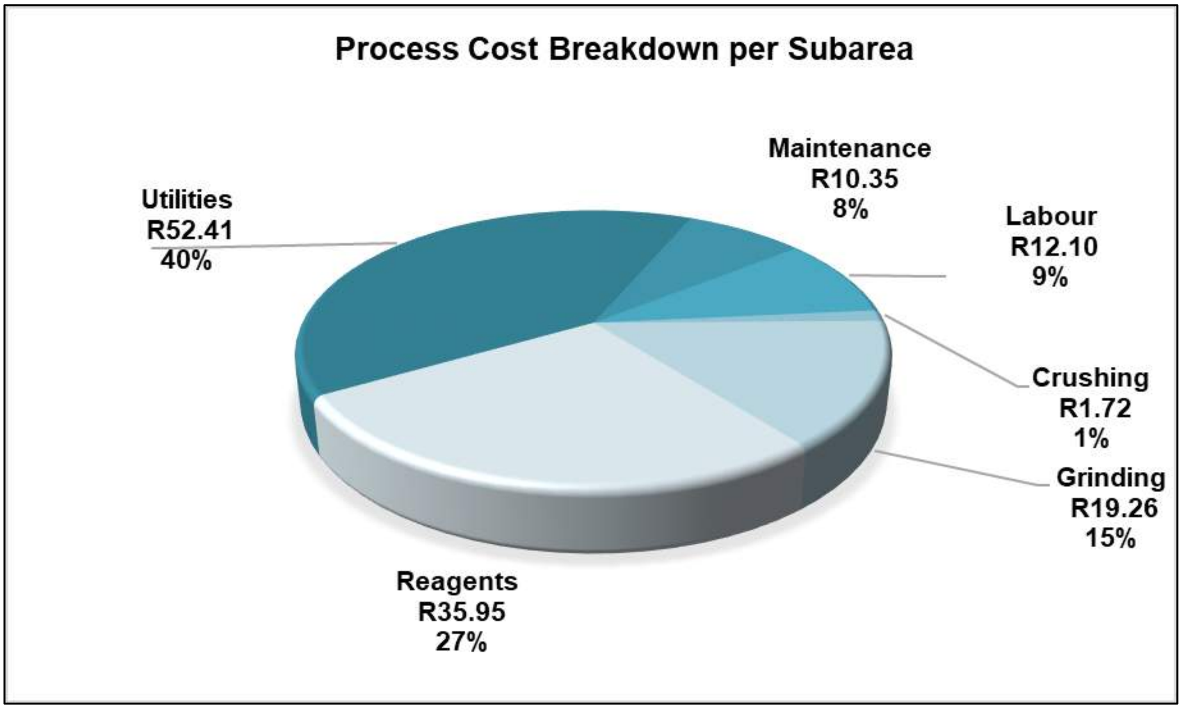


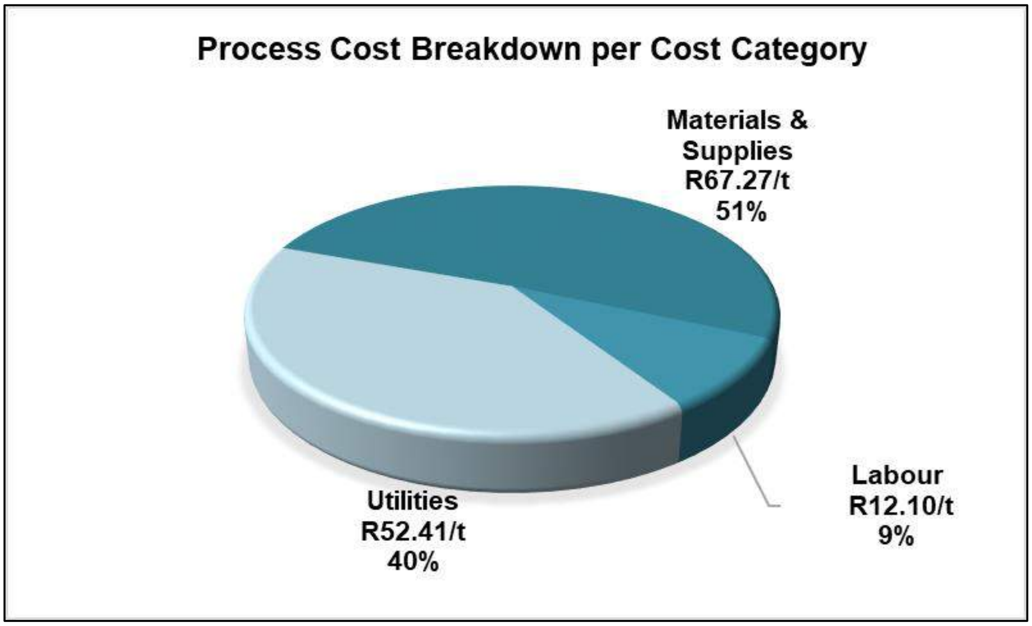
Table 21-23 provides a breakdown of the process cost per subarea and cost category.

Table 21-23: Process Cost per Subarea and Cost Category

Subarea	Cost Category	Average LOM (ZAR / t)	% of Total Process Cost
Utilities	Utilities	R52.41	40%
Maintenance	Materials and Supplies	R10.35	8%
Labour	Labour	R12.10	9%
Crushing	Materials and Supplies	R1.72	1%
Grinding	Materials and Supplies	R19.26	15%
Reagents	Materials and Supplies	R35.95	27%
Total		**R131.78**	**100%**

Figure 21-16 provides a breakdown of the average LOM process operating cost per cost category.

Figure 21-16: Process Plant Operating Cost Summary over Life of Mine



Materials and supplies comprise the bulk of the process costs at 51% followed by utilities at 40%. The process utilities cost calculated is based on nominal power cost directly related to power consumption. Fixed tariff charges based on process plant power demand reflects under the infrastructure cost area. Materials and supplies can be divided further in consumables such as liners, reagents, and consumables and spares related to grinding media and general maintenance. Refer to the Stores and Maintenance and Consumables sections under Section 21.10.4.2 for details.

21.10.4.2 Operating Costs Inputs

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 21-24.

Table 21-24: Waterberg Processing Plant Staffing Model

Function	At-work Compliment
Management and Overheads	4
Administration	7
Office and Change House	13
Metallurgy (Technical Support)	2
Plant Process (Operations)	72
Plant Engineering (Maintenance)	38
Plant Stores	5
Plant Sample Preparation Laboratory	8
Total	**149**

The total Concentrator Plant labour amounts to R12.10 per ore tonne milled.

Power

The rates used in the operational cost estimate for power are based on Eskom Megaflex tariffs as detailed in Section 21.10.2. The total LOM plant power cost amounts to R52.41 per ore tonne milled.

The total connected load for the Concentrator Plant is estimated at 60 MW with an absorbed load of 41 MW. The process plant has an average power consumption of 70kWh per ore tonne milled.

Water

The water consumption is based on a mine-wide water balance and includes for underground water inflows, anticipated water losses associated with the TSF, water storage dams, and calculated consumptions from mining and the Concentrator Plant.

The total complex raw water requirement supplied from drill holes is calculated at a maximum of 5.2 ML/day. This operating cost included for water supply assumed that all raw water will be sourced from drill holes, and the associated pumping costs were included in the shared infrastructure operating costs.

Stores and Maintenance

The stores and maintenance costs are based on replacement factors applied to the mechanical equipment supply costs. The total plant maintenance cost amounts to R10.35 per ore tonne milled.

Concentrate Transport

Concentrate transport costs of R425 per wet tonne were based on a quoted price from a transport contractor. The concentrate transport cost is not included in the operating cost but is included in the financial model as a realisation cost.

Consumables

Table 21-25 presents a summary of the plant consumable costs included in the estimate.

Table 21-25: Waterberg Plant Consumable Costs

Consumable	Operating Cost (ZAR / t milled)
Crusher Liners	R0.74
Mill Liners	R0.98
Grinding Media	R19.26
SIBX	R3.58
Frother	R6.03
Depressant	R12.99
Coagulant	R5.70
Flocculant	R7.65

Mill Liners

An allowance was made for liner replacement based on calculations incorporating the material Ai data from testwork and grinding media consumptions as per simulations from the DRA in-house comminution consultant. The liner costs are based on pricing received from a reputable mill supplier.

Crusher Liners

The costs used for the primary, secondary, and tertiary crusher liners are based on the two-year operational spares as received from the preferred crusher supplier.

Reagents and Grinding Media

Reagent supply costs are based on quotations received from reputable reagent suppliers. The reagent consumptions are based on testwork consumptions, and no allowance is made for buildup 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, while the supply costs were received from a reputable grinding media vendor.

21.10.5 Engineering and Infrastructure

The TSF, backfill plant, and process laboratory along with regional and shared infrastructure are in the engineering and infrastructure area. Engineering and infrastructure operating costs amount to R116.36/t over LOM, comprising 19% of the total operating cost. Table 21-26 provides a breakdown of the engineering and infrastructure cost per cost category.

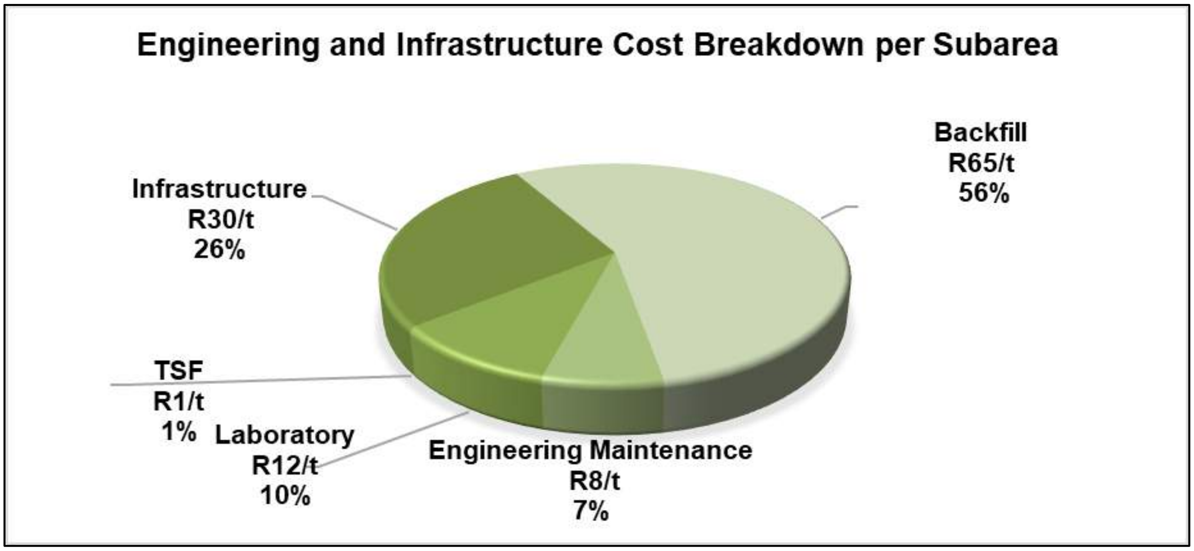
Table 21-26: Total Life-of-Mine Engineering and Infrastructure Operating Cost Breakdown per Cost Category

Engineering and Infrastructure Cost per Cost Category	Average LOM (ZAR / t)
Materials and Supplies	R63.05
Labour	R8.39
External Services	R13.16
Utilities	R31.76
Total OpEx Cost	R116.36

Materials and supplies comprise more than half of the cost and will be detailed in the subsections below.

Utilities comprise mainly power costs and a small portion for water and sewerage treatment. Power costs estimated for fixed equipment and infrastructure associated with engineering and infrastructure is in the infrastructure subarea section along with costs resulting from fixed power tariff charges calculated on total demands and usage. Power associated with backfill is in the backfill subarea. Labour cost for engineering is in the maintenance and backfill subareas. Engineering and infrastructure operating cost can be further detailed into infrastructure, backfill, maintenance, TSF, and laboratory. Figure 21-17 provides a cost breakdown of each of these categories.

Figure 21-17: **Life-of-Mine Average R/t Engineering and Infrastructure Operating Cost Breakdown per Subarea**



Backfill constitutes 56% of the total engineering and infrastructure cost, followed by infrastructure at 26%. Table 21-27 provides average ZAR per ore tonnes milled, per subarea, cost category, and subcategory.

Table 21-27: Engineering and Infrastructure Cost Detail per Subarea and Cost Category

Subarea	Cost Category	Average LOM (ZAR / t)	% of Total Engineering and Infrastructure Cost
Backfill	Labour	R1.61	1.4%
	Materials and Supplies	R60.11	51.7%
	Utilities	R2.92	2.5%
Maintenance	Labour	R6.78	5.8%
	Materials and Supplies	R1.35	1.2%
Laboratory	External Services	R11.72	10.1%
TSF	External Services	R1.44	1.2%
Infrastructure	Materials and Supplies	R1.59	1.4%
	Utilities	R28.84	24.8%
Total		**R116.36**	**100%**

21.10.5.1 Backfill

The cost of backfill is built up from a small labour complement to operate the backfill plant, power for fixed equipment associated with pumping backfill paste, maintenance, backfill barricades, and binder consumable. The cost of binder is R2 224 per tonne and is the most significant operating cost item accounting for 77% of the total cost of R64.64 per tonne milled. The cost of backfill placed is R109 per tonne. Backfill-related input was provided by SSBS and the mining team.

21.10.5.2 Infrastructure

Costs under infrastructure comprise materials and supplies related to infrastructure stores and maintenance, power, and water.

The stores and maintenance costs included in the operation costs estimate is based on replacement factors applied to the mechanical equipment supply costs related to infrastructure and amounts to R 290 million over the LOM or R1.60 per tonne milled.

The power cost is R28.67 per tonne milled and the water cost is negligible at R0.17 per tonne milled. The bulk of the infrastructure power cost reflected in this area comprise fixed power tariff charges calculated on mine-wide demands as per the tariff charges shown in Table 21-17. The variable power cost portion is based on the nominal power cost rate for the shared infrastructure load is estimated at 3.3 MW, (absorbed load of 1.45 MW).

An allowance of R0.65/m^3 is included for potable water treatment based on the quantities highlighted in the mine-wide water balance. A further allowance of R4.50/m^3 is included for the sewerage treatment plant consumables.

21.10.5.3 Maintenance

The infrastructure maintenance cost is made up of labour and mobile fleet costs associated with the waste dump. The infrastructure labour in the maintenance subarea covers all labour associated with infrastructure operation and maintenance as well as labour for stores. The cost breakdown for the maintenance subarea is shown in Table 21-27. Table 21-28 presents the shared infrastructure staffing model.

Table 21-28: Waterberg Shared Infrastructure Staffing Model

Function	At-work Compliment
Offices and Change house	30
General Surface Infrastructure	22
Surface Infrastructure – Sewerage Handling	5
Surface Infrastructure – Bulk Fuel Receiving and Generator Yard	8
Surface Infrastructure – Water Treatment	10
Surface Infrastructure – Waste Handling	4
Surface Infrastructure – Weighbridges	2
Main Stores	21
Waste Dump	16
Total	**102**

21.10.5.4 Centralised Laboratory Complex

A third-party operated centralized laboratory facility is included in the Waterberg Project design. The operating costs for this facility is based on pricing received from a reputable operator and is summarised in Table 21-29. Total staff compliment is 43.

Table 21-29: Waterberg Centralised Laboratory Operating Costs

Consumable		Operating Cost (R/t Milled)
Variable Cost		R5.69
Fixed Cost		R6.27
Total Cost		**R11.96**

Tailings Disposal

The operating costs comprise the management of the TSF deposition, as well as general maintenance of the TD, RWD, and SWD.

Table 21-30 presents a summary of the TSF operating costs.

Table 21-30: Waterberg Tailings Storage Facility Operating Costs

Consumable	Operating Cost (R/t Milled)
Site Establishment and Disestablishment	R0.002
TSF Deposition Management Costs	R0.95
TSF Operational Costs (i.e., pipeline and valve replacement costs, maintenance, etc.)	R0.39
Consulting Services (Quarterly Inspections, etc.)	R0.11
Total TSF Operating Costs	**R1.45**

The fixed cost portion of the TSF costs equate to 73%.

21.10.6 General and Administrative

G&A operating costs constitute 3% of the total LOM operating costs at R19 per tonne milled. It includes labour, exploration drilling, security services, insurance, leasing, office consumables, and waste disposal costs.

Table 21-31 provides a breakdown of the G&A costs per cost category. Figure 21-18 shows the G&A costs graphically.

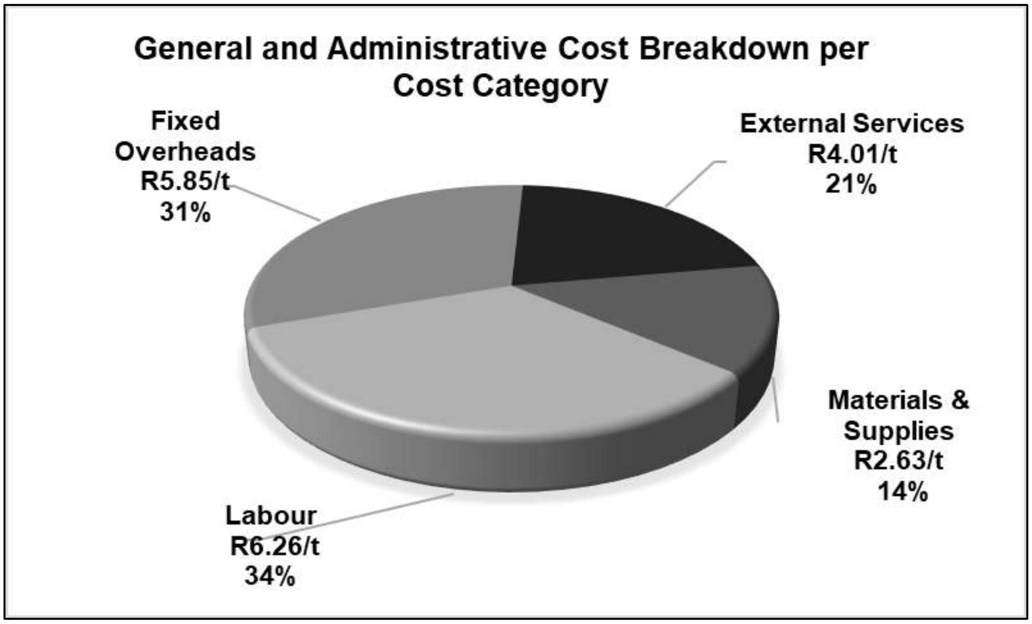
Table 21-31: General and Administrative Cost Breakdown

G&A Cost per Cost Category	Average LOM (ZAR / t)
Materials and Supplies	R2.63
Labour	R6.26
Fixed Overheads	R5.85
External Services	R4.01
Total OpEx Cost	**R18.75**

Labour comprises the bulk of the G&A costs at 34%, followed by fixed overheads at 31%, and external services at 21%.

Labour is the highest cost contributing component of G&A operating costs averaging R6.26 per ore tonnes milled over LOM. G&A labour includes general office staff such as finance, human resources, technical services, and health and safety personnel. Labour remains relatively constant over LOM at 30 personnel.

Figure 21-18: **Life-of-Mine Average R/t General and Administrative Operating Cost Breakdown per Cost Area**



The following staff is allocated under G&A.

- Information Technology
- Accounting
- Procurement
- Human Resources
- Sanitation
- Safety
- Access Control

In addition to the G&A labour categories presented above, G&A type labour was also included in the various underground mining complexes.

All management and administrative personnel required for the Waterberg Project were included as part of the labour costing and assumed to be on site. Management and labour personnel could potentially work off site or be outsourced. As such, no corporate provisions have been included in the operating cost model or the financial model.

The fixed overhead cost comprising insurances coverage and leasing related to water servitude and land is under the G&A area and amounts to R5.85 per ore tonnes milled over LOM. Costs were provided by the client team at R31 565 per month for land and R222 195 per month for the water servitude area leasing. Insurance cost comprised the bulk at 5.17 per tonne milled over the LOM.

Contracted security at R1 300 330 per month and waste removal costs form part of external services for the G&A area and averages R4.01 per ore tonnes milled of the total LOM operating cost.

G&A materials and supplies comprise exploration drilling consumables at R750/m with a small allowance for stationary, printing, and general office consumables. It is proposed that over the LOM, some 478 km of delineation drilling will be completed (an average of almost 1 000 m per month) or some 390 tonnes of ore per metre drilled.

22 ECONOMIC ANALYSIS

22.1 Introduction

This section revolves around the economic analysis and investment evaluation of the Waterberg Project, which encapsulates the following key aspects.

- A statement of and justification for the principal inputs and assumptions applied in the financial model.
- A review of the key project drivers (ore production, metallurgical recoveries, CapEx, and OpEx) developed by the various subject matter experts in support of the greater DFS.
- A tabulated summary and graphical representation of the forecast LOM free cash flow per annum.
- A summary of the regulatory costs as legislated in RSA, which largely pertain to corporate income tax, mineral royalties, SLP expenses, and mine rehabilitation and closure costs.
- A summary and analysis of the key business return metrics, which include NPV, IRR, payback period, and the peak funding requirement.
- An analysis of the business return metrics' sensitivity to movements in key inputs and assumptions such as metal prices, foreign exchange rates, and the discount rate.

22.2 Basis of Evaluation

The investment evaluation principles applied are aligned with best practices suitable for the evaluation of mineral projects at a DFS level of accuracy.

A detailed financial model was developed to analyze the economic viability of the Waterberg Project. The model develops real, post-tax, unleveraged free cash flow forecasts, which are discounted to determine the Waterberg Project's NPV. Table 22-1 lists the basis of evaluation assumptions associated with the Waterberg Project.

Table 22-1: Basis of Evaluation Assumptions

Factor	Assumption
Method of Analysis	Discounted Cash Flow
Cash Flow Terms	Real Terms
Base Currency	ZAR (R)
Secondary Currency	US$
Base Date of Evaluation	01 July 2019
Discount Rate	8.0% (Real, Post-Tax)

22.3 Inputs and Assumptions

22.3.1 Metal Prices

The following two metal price scenarios were adopted for the purposes of the economic evaluation.

- Spot prices as of 4 September 2019 (spot prices).
- Three-year trailing average prices up to 4 September 2019 (three-year trailing prices).

Table 22-2 summarises the metal prices applicable to each scenario evaluated. All metal prices are applied as single, long-term (real) prices over the 47-year LOM, adjusted to July 2019 money terms.

Table 22-2: Metal Price Scenarios

Factor	Unit of Measure	Spot Prices	Three Year Trailing Average Prices
Pt	US$ / oz (real July 2019)	980.00	931.00
Pd	US$ / oz (real July 2019)	1 546.00	1 055.00
Au	US$ / oz (real July 2019)	1 548.00	1 318.00
Rh	US$ / oz (real July 2019)	5 036.00	1 930.00
Basket Price (4E)	***US$ / oz (real July 2019)***	1 425.00	1 045.00
Cu	US$ / lb (real July 2019)	2.56	2.87
Ni	US$ / lb (real July 2019)	8.10	5.56

Primarily driven by the ~50% increase in the Pd price, it is evident from Table 22-2 that the economic evaluation at the spot metal price scenario will yield far superior financial returns compared to the three-year trailing average metal price scenario.

22.3.2 Foreign Exchange

The US$/ZAR rate is one of the key determinants of profitability on the Waterberg Project. The US$/ZAR rate adopted for the economic evaluation of the two metal price scenarios (as discussed above) are documented in Table 22-3. The long-term real rates are kept flat from 2025 onwards (i.e. until the end of the LOM in 2067).

Table 22-3: US$/ZAR Exchange Rate Scenarios

Rate	Unit of Measure	2020	2021	2022	2023	2024	Long-term Real
Spot Price	ZAR Real July 2019	15.00	15.00	15.00	15.00	15.00	**15.00**
3-Year Trailing Average Price	ZAR Real July 2019	14.52	14.91	15.19	15.51	15.95	**15.95**

The long-term real US$/ZAR exchange rate for the spot metal price scenario is set at 15.00, which is based on an intra-day traded spot rate as of 4 September 2019.

The US$/ZAR exchange rate for the three-year trailing price scenario is based on Bloomberg's nominal consensus forward-curve as at June 2019, which translates into a long-term real US$/ZAR rate of 15.95.

Since 2008, the ZAR has depreciated against the US$ at an average year-on-year rate of ~7% (nominal terms). Adjusting this rate for purchase power parity results in a real rate of depreciation of ~3% per annum. Bloomberg's consensus forecast suggests a similar devaluation of the ZAR against the US$ over the next five years (2.4%), which equates to a long-term real US$/ZAR rate of 15.95 (6% higher than the spot price assumption of 15.00). Keeping the US$/ZAR rate flat at 15.00 in the spot price scenario is considered more conservative than both the Bloomberg consensus forecast as well as the historical rate of depreciation observed. Refer to Section 19.2.6 of this Technical Report for historical information on exchange rate.

22.3.3 Inflation and Escalation

No nominal inflation was considered for the purposes of the financial evaluation. Inflationary cost increases have historically been observed in the mining sector of South Africa, which has primarily been driven by the ~4% per annum (real) increase in wages (unskilled and semi-skilled labour) and power (Eskom electricity tariffs).

In the short term, these above inflationary increases are expected to be negated by the ongoing devaluation of the US$/ZAR rate and, in the long term, are expected to normalise in line with the RSA consumer price index. In the same manner that the US$/ZAR is kept constant over the LOM (despite the observed 3% per annum real historic depreciation of the ZAR against the US$ over the past 10 years), costs are kept flat in in July 2019 real terms.

22.3.4 Revenue Realisation Costs

Revenue realisation costs applicable to the Waterberg Project are listed below and summarized in Table 22-4.

- Transport and handling – cost of transporting moist concentrate (12% moisture) from the mine to a smelting complex up to 417 km distant.
- Payable metal in concentrate – the percentage of metal in concentrate payable to the Waterberg Project, including all treatment and refining charges.
- Contractual price discounts – the contractual discounts applied to the market prices for the base metals in concentrate.

Table 22-4: Revenue Realisation Costs

Category	Parameter	Unit	Assumption
Transport	Concentrate Handling and Transport	ZAR / wmt (Real)	425.0
Payable Metal in Concentrate	Pt	% of Gross Revenue	85.0
	Pd	% of Gross Revenue	85.0
	Rh	% of Gross Revenue	85.0
	Au	% of Gross Revenue	85.0
	Cu	% of Gross Revenue	73.0
	Ni	% of Gross Revenue	68.0
Contractual Price Discounts	Cu	US$ / Tonne Metal	200.0
	Ni	US$ / Tonne Metal	100.0

22.3.5 Corporate Income Tax

Corporate income tax is calculated based on the prevailing 28% corporate income tax rate for resident companies in South Africa as of July 2019. The corporate income tax rate is levied against the assessed taxable income, inclusive of all tax allowances applicable to mining companies, as per the Income Tax Act. No changes in the RSA corporate income tax rate is expected in the foreseeable future.

22.3.6 Mineral Royalty Tax

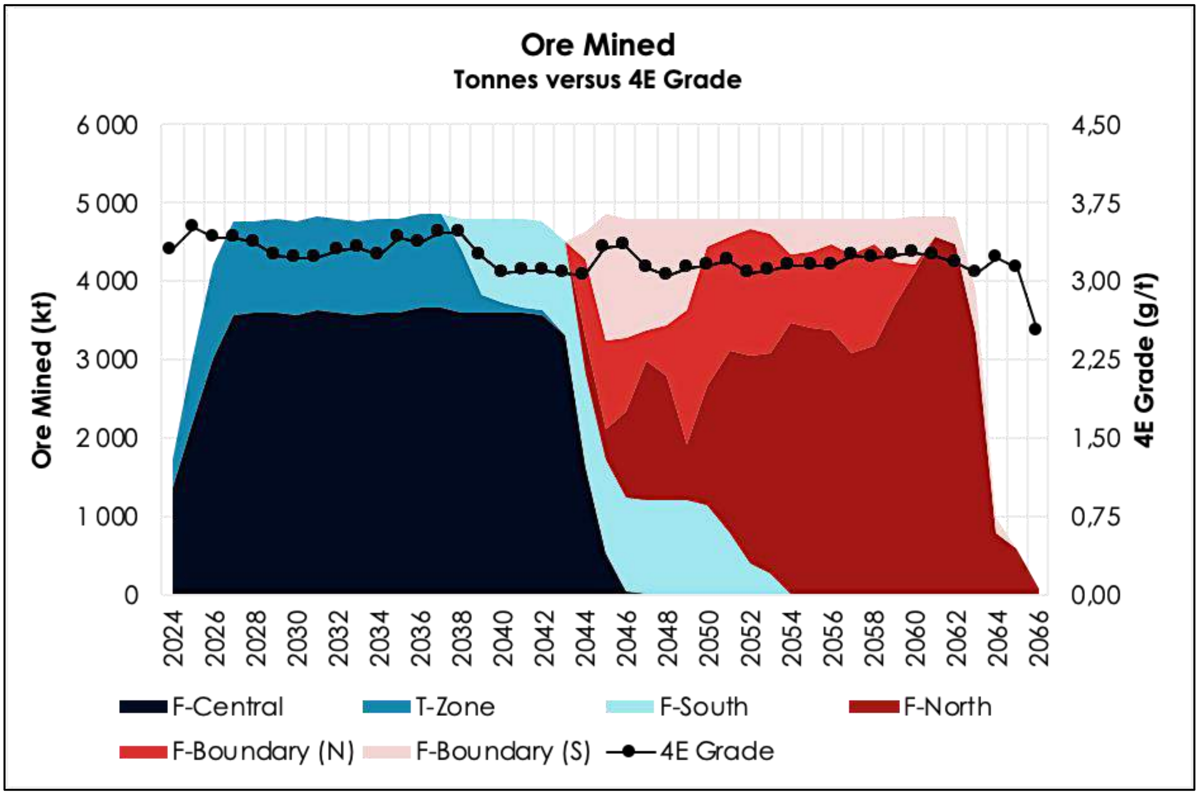
Mineral royalties are estimated based on the Schedule 2 royalty formula as documented in the Royalty Act 28 (2008; Government Gazette No. 31635), and the Mineral and Petroleum Resources Royalty (Administration) Act No. 29 (2008; Government Gazette No. 31642). The minimum payable royalty rate is 0.5% of the gross sale value of concentrate sold, with the maximum payable rate capped at 7%. No change in the royalty rate scheme is expected in the foreseeable future. Refer to Section 4.5 of this Technical Report.

22.4 Project Drivers

22.4.1 Production Schedule

A monthly ore production schedule (tonnes and grade) is included in the financial model. The production schedule encapsulates the development and stoping ore to be mined from the six various mining zones over the LOM. The annualised LOM production profile per mining zone is depicted in Figure 22-1.

Figure 22-1: Annualised Life-of-Mine Production Profile



The infrastructure for the Central and South Complexes will be established from 2022 to 2024. The F-Central Zone is mined at a steady state rate of 300 ktpm via the Central Complex decline access, whereas the T- and F-South Zones are mined at a steady state rate of 100 ktpm via the South Complex decline access. Commercial production is reached in January 2026, once 70% of the annual steady-state ore production is achieved.

The development for the North Complex infrastructure is deferred until the 2040 and production from mining zones F-North, F-Boundary (North), and F-Boundary (South) commence in 2043. Despite the minor dip in ore production in 2043, the North Complex is able to sustain the 400 ktpm production feed to the mill for the remainder of the LOM.

A summary of the mine physicals (tonnes and grade) per decline complex is shown in Table 22-5.

Table 22-5: Mine Physicals per Complex

Area	Metric	Unit	Result
Central Complex	Ore Tonnes	kt	70 131.00
	4E Grade	g/t	3.08
	Cu Grade	%	0.07
	Ni Grade	%	0.18
South Complex	Ore Tonnes	kt	32 555.00
	4E Grade	g/t	3.68
	Cu Grade	%	0.11
	Ni Grade	%	0.11
North Complex	Ore Tonnes	kt	84 821.00
	4E Grade	g/t	3.18
	Cu Grade	%	0.09
	Ni Grade	%	0.20
Summary	**Ore Tonnes**	**kt**	**187 507.00**
	Combined 4E Grade (LOM Average)	**g/t**	**3.23**
	Cu Grade (LOM Average)	**%**	**0.09**
	Ni Grade (LOM Average)	**%**	**0.18**

22.4.2 Metallurgical Recoveries

Ore produced from the various mining zones is fed to an on-site Concentrator Plant where a 4E concentrate (inclusive of base metal credits) is produced. The metallurgical recovery estimates for each saleable metal (%), the concentrate production schedule (tonnes per month), 4E grade in concentrate (g/t), and moisture content (%) are included in the economic model as key inputs.

The LOM average metallurgical recoveries achieved in the concentrator are shown in Table 22-6.

Table 22-6: Metallurgical Recoveries (Life-of-Mine Average)

Category	Metric	Unit	Result
4E Metals	Pt	% LOM Average	78.4
	Pd	% LOM Average	80.4
	Au	% LOM Average	68.6
	Rh	% LOM Average	65.8
Base Metals	Cu	% LOM Average	83.0
	Ni	% LOM Average	48.0

The Concentrator Plant is expected to produce saleable concentrate at a steady-state rate ranging between 13 500 to 14 500 wet tonnes per month, at a LOM average 4E concentrate grade of 79.9 g/t and a moisture content of 12%. At steady state, the plant will recover an average of 420 koz of 4E metal per year for the first 11 years at steady state.

22.4.3 Capital Expenditure

A CapEx estimate was prepared in accordance with the approved WBS.

All capitalized costs incurred prior to commercial production (January 2026) is reported as project CapEx and all capitalized costs incurred post commercial production is reported as sustaining CapEx. A summary of the total CapEx (project and sustaining) is reported in Table 22-7.

Table 22-7: Capital Expenditure Summary per Work Breakdown Structure Level 1

Metric	Unit	Project CapEx	Sustaining CapEx	Total
Underground Mining	ZAR M (Real)	6 097	20 277	26 374
Concentrator	ZAR M (Real)	2 580	829	3 409
Shared Services and Infrastructure	ZAR M (Real)	682	0	682
Regional Infrastructure	ZAR M (Real)	1 229	258	1 487
Site Support Services	ZAR M (Real)	234	47	281
Project Delivery Management	ZAR M (Real)	654	99	753
Other Capitalised Costs	ZAR M (Real)	331	65	396
Provisions	ZAR M (Real)	1 298	42	1 340
Total CapEx (excluding Capitalised OpEx)	**ZAR** M **(Real)**	**13 105**	**21 617**	**34 722**
Capitalised OpEx	ZAR M (Real)	3 453	0	3 453
Total CapEx (including Capitalised OpEx)	**ZAR** M **(Real)**	**16 559**	**21 617**	**38 175**

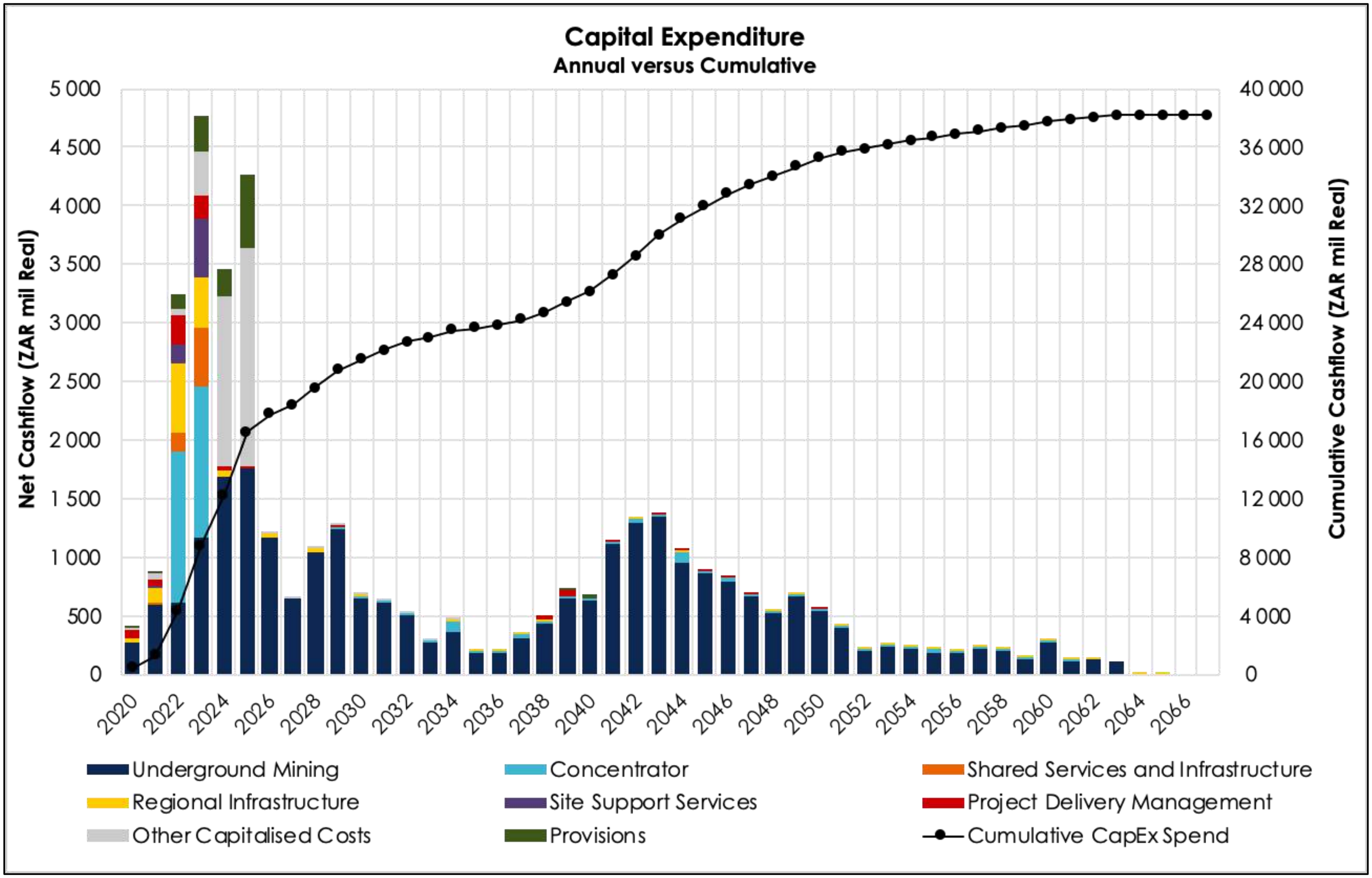
The total project CapEx amounts to R16 559 M (US$1 104 M). The CapEx includes an assessment of the capitalized operating costs incurred prior to commercial production, which equates to R3 453 M (US$230 M). The total sustaining CapEx, which includes the establishment of the North Complex infrastructure in 2040, is estimated as R21 617 M (US$1 441 M).

Apart from the ongoing capital development over the LOM, which constitutes the bulk of the sustaining CapEx estimate, two additional types of sustaining CapEx were provisioned for in the economic assessment, namely 1) replacement capital, and 2) SIB capital. Replacement capital is estimated based on the useful life of key equipment (e.g. LHD trucks), whereas SIB capital accounts for minor capital replacements that are not accounted for on an itemized basis (e.g. annual provisions factored from the mechanical equipment cost in process plant).

The CapEx estimate was cash flowed in line with an indicative execution schedule, which was developed in view of the planned development and production schedules. The annualised CapEx cash flow profile is shown in Figure 22-2.

Any capitalised costs incurred prior to the evaluation date (1 July 2019) are considered sunk and were not included in the economic evaluation model.

Figure 22-2: Annualised Capital Expenditure (Life-of-Mine Total)



22.4.4 Operating Expenditure

An OpEx model was prepared to estimate all the “on-mine” costs. The OpEx estimate was prepared in accordance with the approved WBS. The OpEx model leveraged off a number of cost modeling techniques (e.g. zero-based, first principles, etc.) to develop the forecast cost of production.

The OpEx estimate is structured to report cost per mining zone, operating area, and profit and loss element. A summary of the LOM Average OpEx unit costs for each of these reporting categories is shown in Table 22-8, Table 22-9, and Table 22-10, respectively.

Table 22-8: Operating Expenses Unit Cost Summary per Zone

Area	Unit	LOM Average
F-Central	ZAR / t ore mined (Real)	347
T Zone	ZAR / t ore mined (Real)	528
F-South	ZAR / t ore mined (Real)	561
F-North	ZAR / t ore mined (Real)	373
F-Boundary	ZAR / t ore mined (Real)	383
Shared Services	ZAR / t ore mined (Real)	217
Total On-mine OpEx	**ZAR / t ore mined (Real)**	**612**

Table 22-9: Operating Expenses Unit Cost Summary per Area

Area	Unit	LOM Average
Mining	ZAR / t ore milled (Real)	345
Processing	ZAR / t ore milled (Real)	132
Engineering and Infrastructure	ZAR / t ore milled (Real)	116
G&A	ZAR / t ore milled (Real)	19
Total On-mine OpEx	**ZAR / t ore milled (Real)**	612

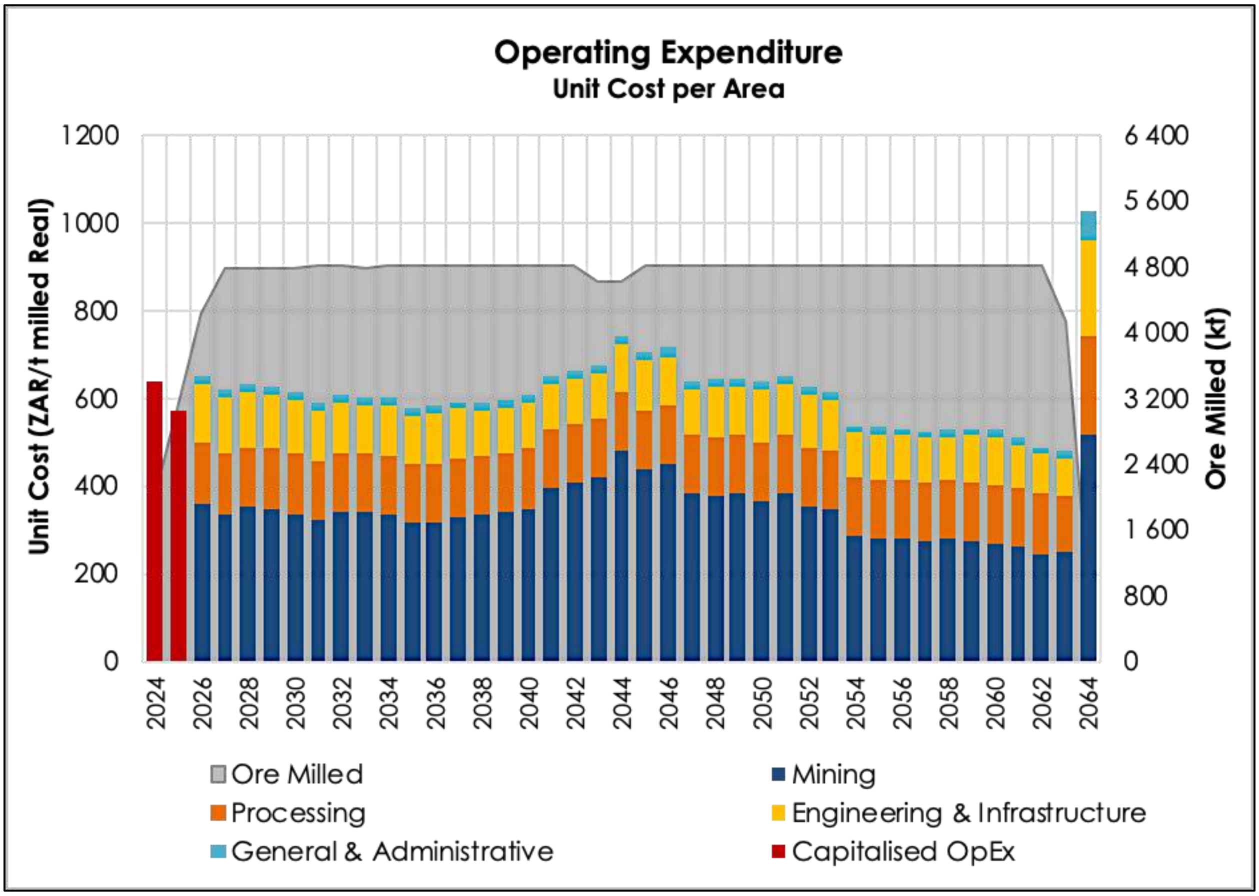
Table 22-10: Operating Expenses Unit Cost Summary per Profit and Loss Element

Area	LOM Average ZAR / t ore Milled (Real)
External Services	17
Fixed Overheads	6
Labour	147
Materials and Supplies	316
Utilities	125
Total On-mine OpEx	612

Figure 22-3 depicts the average unit cost of production per area, overlaid with the scheduled tonnes milled per annum.

The OpEx estimate only accounts for on-mine expenses to be incurred. All off-mine expenses (e.g. revenue realisation and other indirect costs) are accounted for in the economic model and are specifically excluded from the OpEx estimate.

Figure 22-3: Unit Cost of Production per Area



22.4.5 Other Indirect Costs

The following other indirect costs were provisioned for the in the economic evaluation model.

- SLP expenses – per legislative requirements in South Africa and to maintain a right to mine, all mining operations are expected to provision for local economic development (~1% of pre-tax profit), human resource development (~R2 M per annum), and retrenchment / downscaling provision (~R2.5 M per annum).
- Rehabilitation and closure costs – per legislative requirements in South Africa and to maintain a right to mine, all mining operations are required to assess the rehabilitation and closure liability applicable to the operation. The Lombard's bank guarantee product applicable to the Waterberg Project requires a 40% upfront contribution (~R44 M real) of the total assessed liability after 10-years of operation (~R111 M real). The balance (~R77 M) is provisioned for over a 10-year period in equal instalments of ~R7.7 M per annum (real). The ongoing TSF rehabilitation, which is not included in the 10-year liability assessment, was included as a standalone item in the sustaining CapEx budget.

22.4.6 Working Capital

Working capital requirements revolve primarily around the accounts receivable and payable assumptions applied in the economic evaluation model.

- Accounts receivable – 85% of the gross sale value of the concentrate is receivable on delivery to the smelting complex as an advance payment. The advance payment is subject to an interest charge of 4.43% per annum. The balance (15% of the metal in concentrate), is payable in full after 90 days (12 weeks).
- Accounts payable – All external services, fixed overheads, materials and supplies, and utility cost accounts are payable after 60 days (8 weeks).
- Finished stock – No material level of concentrate stock will be kept on-site as material is shipped immediately.

22.5 Summary of Results

22.5.1 Key Metrics

The key business metrics for the two assessed metal price scenarios are summarized in Table 22-11.

The business case is value accretive in both metal price scenarios, generating a post-tax $NPV_{8.0\%}$ of R14 736 M (spot prices) and R5 616 M (three-year trailing average prices), respectively.

When measured from the date of first capital spend (January 2020), the payback period is estimated at 8.4 years (spot prices) and 11.2 years (three-year trailing average prices), respectively.

The peak funding requirement is denoted by the maximum cumulative negative free cash flow position over the LOM (real terms) and is estimated at R9 255 M (spot prices) and R10 261 M (three-year trailing average prices), respectively.

The value investment ratio (VIR) expresses the peak funding requirement in relation to NPV. The rule of thumb suggests that projects with a VIR of greater than 1.0 resemble a highly robust investment proposition. The Waterberg Project's VIR is estimated at 1.6 (spot prices) and 0.6 (three-year trailing average prices), respectively, which further highlights the sensitivity of the Waterberg Project's returns to movements in the metal prices.

Table 22-11: Key Business Metric Results

Metric	Unit	Spot Prices	Three Year Trailing Average Prices
NPV (ZAR) [7]	ZAR M	14 736.0	5 616.0
NPV (US$)	US$ M	982.0	333.0
Peak Funding (ZAR)[8]	ZAR M (Real)	9 255.0	10 261.0
Peak Funding (US$)	US$ M (Real)	617.0	667.0
IRR	% (ZAR Real)	20.7	13.3
Undiscounted Payback Period[9]	Years	8.4	11.2
VIR[10]	Ratio	1.6	0.6

22.5.2 Cost Competitiveness

The Waterberg Project competitiveness can be summarised by considering the cost of production in relation to other similar producers in the region. The LOM average cash cost, all-in-sustaining cost and all-in cost is shown in Table 22-12.

Table 22-12: Cost Competitiveness Metrics

Metric	Scenario 1: Spot Prices (US$ / 4E oz)	Scenario 2: Three-year Trailing Prices (US$ / 4E oz)
On-Site Operating Costs	487	456
Smelting, Refining, and Transport Costs	302	227
Royalties and Production Taxes	88	54
Less Byproduct Base Metal Credits	(236)	(184)
Total Cash Cost	**640**	**554**
Sustaining Capital	94	88
Total All-in Sustaining Cost	**734**	**642**
Project Capital	34	32
Total All-in Cost	**767**	**674**

[7] Based on the aggregated unleveraged free cash flow (after-tax), discounted at the real, post-tax discount rate of 8.0%. The NPV is assessed on a 100% project basis and not at a shareholder level.

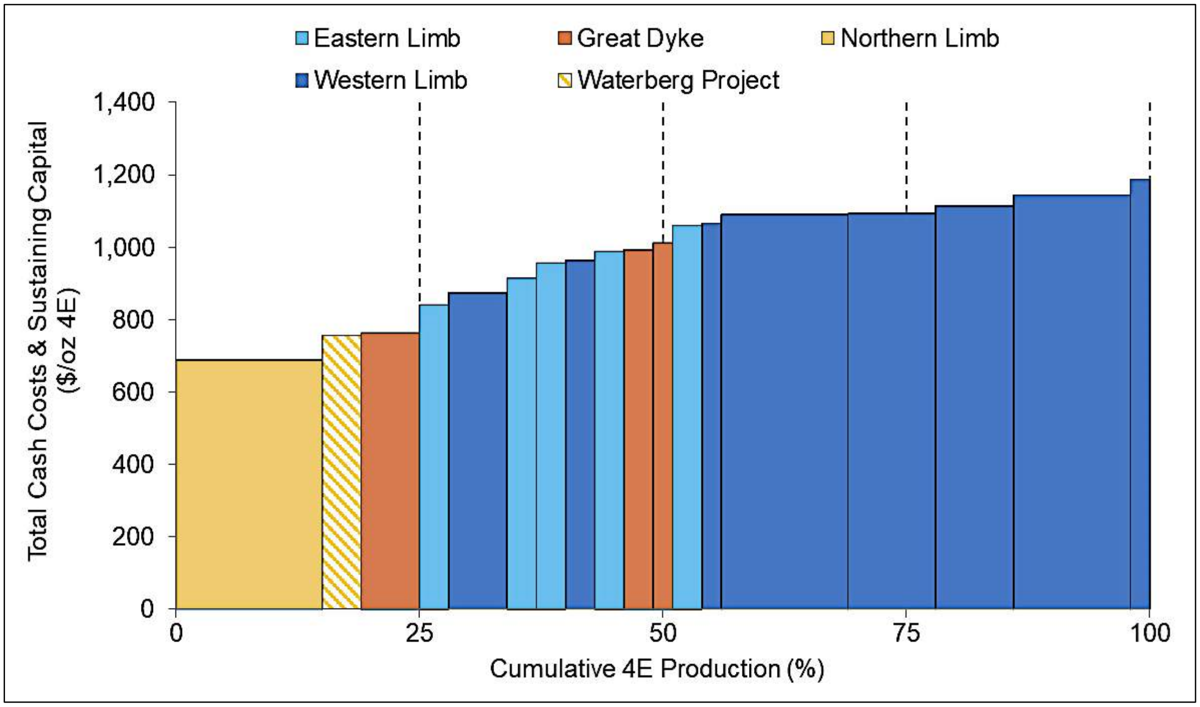
[8] Based on the maximum cumulative negative undiscounted free cash flow position (real terms).

[9] Based on the cumulative undiscounted and unleveraged free cash flow (after-tax) measured from the date of first project capital spend (January 2020).

[10] Estimated as the Peak Funding requirement (undiscounted) divided by the Project's post-tax NPV.

The all-in sustaining capital curve, net of base metal credits and inclusive of smelter payability as a cost, is shown in Figure 22-4. The all-in sustaining capital for all the producers is normalized and expressed in US$ / 4E oz.

Figure 22-4: All-in Sustaining Cost Curve per 4E Ounce (Spot Prices)



The Waterberg Project is firmly in the lowest quartile of regional PGE cost producers and, therefore, has a substantive competitive advantage over most of its peers.

22.5.3 Project Cash Flows

The key annual and cumulative cash flows for the Waterberg Project are shown in Figure-22-5 and Table 22-13, respectively. Figure 22-6 shows the key cash flow summary at three-year trailing metal prices.

Figure-22-5: Key Cash Flow Summary at Spot Metal Prices

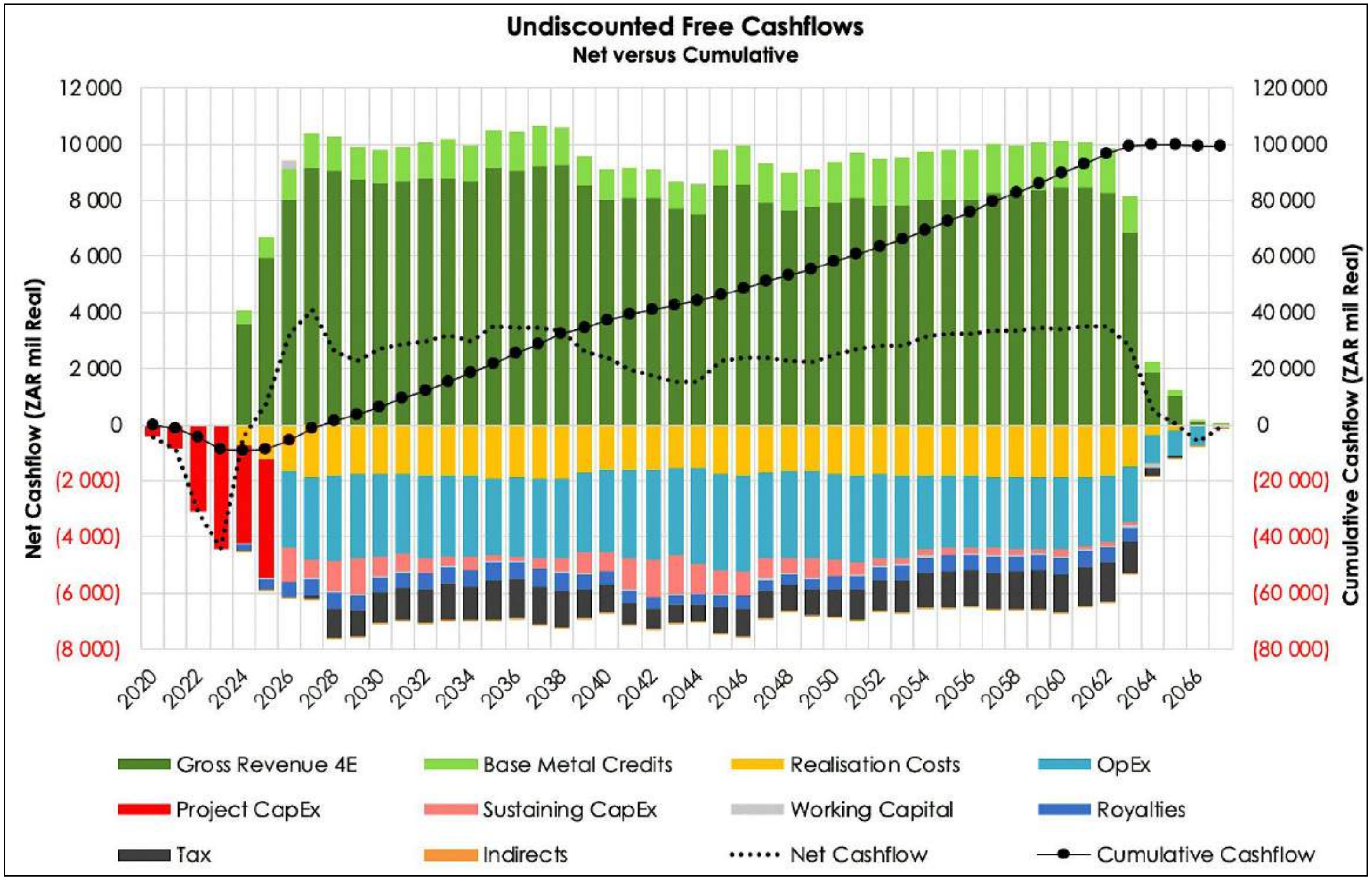


Table 22-13: Undiscounted Cash Flow Summary at Spot Metal Prices (ZAR M Real)

Metric	1st Decade										2nd Decade	3rd Decade	4th Decade	5th Decade
	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yrs 11 - 20	Yrs 21 - 30	Yrs 31 – 40	Yrs 41 - 50
Gross Revenue: 4E	0	0	0	0	3 571	5 920	8 036	9 148	9 062	8 701	88 684	79 694	80 574	34 998
Add Base Metal Credits	0	0	0	0	497	749	1 070	1 227	1 216	1 176	12 975	11 910	16 871	6 798
Less Selling Expenses	0	0	0	0	(764)	(1 197)	(1 630)	(1 850)	(1 826)	(1 757)	(18 230)	(16 473)	(18 218)	(7 669)
Less On-Mine OpEx	0	0	0	0	0	0	(2 754)	(2 953)	(3 031)	(3 008)	(28 595)	(31 877)	(27 465)	(11 926)
Less Project CapEx	(434)	(854)	(3 109)	(4 413)	(3 471)	(4 277)	0	0	0	0	0	0	0	0
Less Sustaining CapEx	0	0	0	0	0	0	(1 213)	(666)	(1 090)	(1 276)	(4 655)	(9 246)	(2 783)	(687)
less Working Capital	0	0	0	0	(16)	(30)	295	(13)	(35)	(64)	(479)	(390)	(495)	(537)
Less Corporate Fees and Costs	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Less SLP	0	0	0	0	(31)	(51)	(47)	(54)	(53)	(50)	(537)	(437)	(511)	(243)
Less Payable Royalties	0	0	0	0	(231)	(383)	(523)	(597)	(589)	(535)	(5 772)	(4 194)	(5 300)	(2 194)
Total Undiscounted Cash Flow (Pre-tax)	**(434)**	**(854)**	**(3 109)**	**(4 413)**	**(445)**	**731**	**3 233**	**4 243**	**3 654**	**3 188**	**43 391**	**28 986**	**42 673**	**18 540**
Less Payable Tax	0	0	0	0	0	0	0	(140)	(1 033)	(911)	(12 284)	(8 225)	(12 087)	(5 503)
Total Undiscounted Cash Flow (Post-tax)	**(434)**	**(854)**	**(3 109)**	**(4 413)**	**(445)**	**731**	**3 233**	**4 103**	**2 621**	**2 277**	**31 107**	**20 761**	**30 586**	**13 037**
Cumulative Undiscounted Cash Flow (Post-tax)	**(434)**	**(1 288)**	**(4 397)**	**(8 810)**	**(9 255)**	**(8 524)**	**(5 291)**	**(1 189)**	**1 432**	**3 709**	**34 816**	**55 577**	**86 163**	**99 201**
Discounted Cash Flow (Post-tax)	**(402)**	**(731)**	**(2 466)**	**(3 240)**	**(303)**	**460**	**1 883**	**2 213**	**1 308**	**1 052**	**9 547**	**2 937**	**1 980**	**499**
Cumulative Discounted Cash Flow ($NPV_{8.0\%}$)	**(402)**	**(1 134)**	**(3 600)**	**(6 840)**	**(7 143)**	**(6 683)**	**(4 800)**	**(2 587)**	**(1 279)**	**(226)**	**9 320**	**12 257**	**14 237**	**14 736**

Figure 22-6: Key Cash Flow Summary at Three-year Trailing Metal Prices

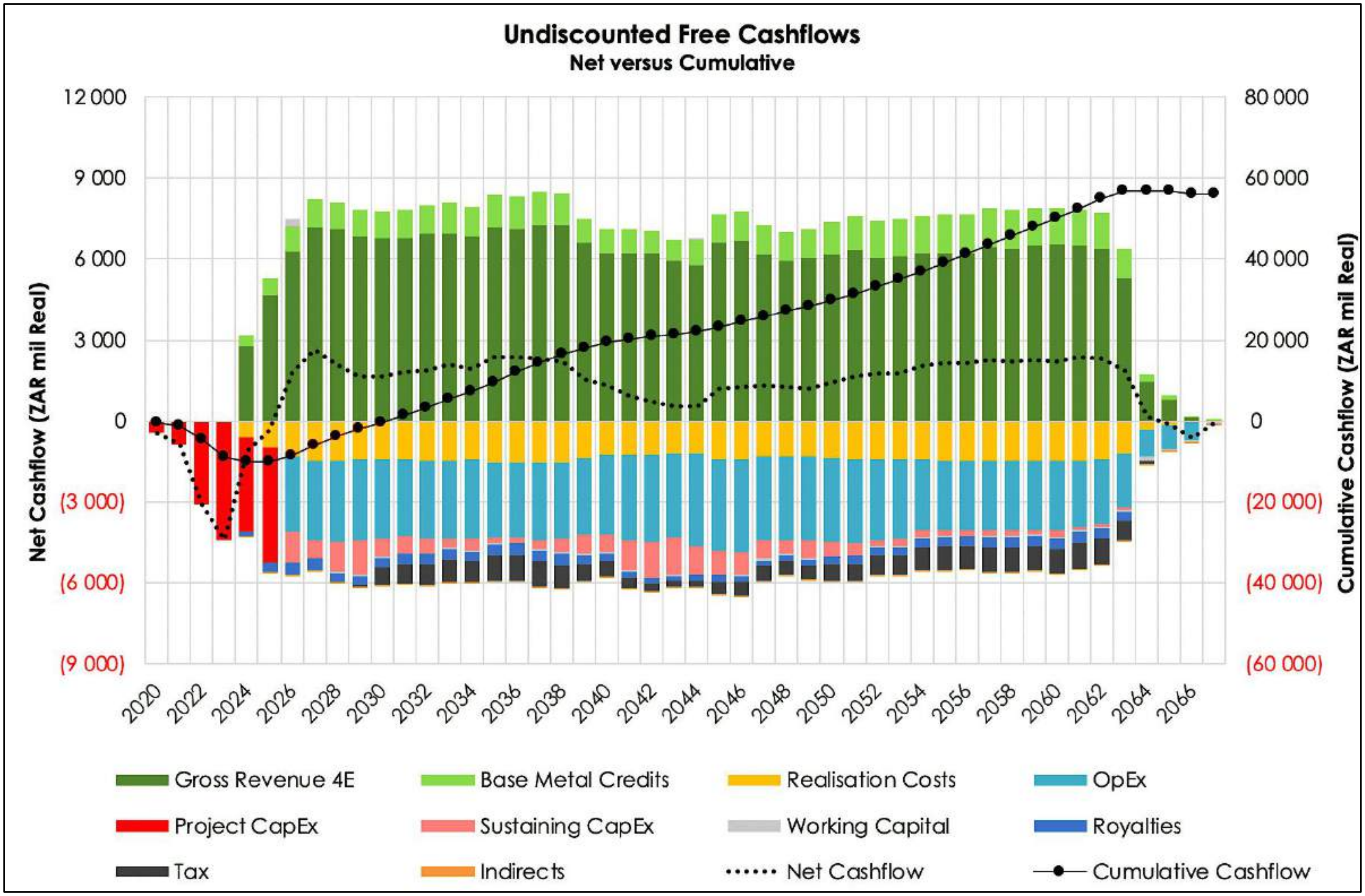


Table 22-14: Undiscounted Cash Flow Summary at Three-year Trailing Prices (ZAR M Real)

Metric	1st Decade										2nd Decade	3rd Decade	4th Decade	5th Decade
	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yrs 11 - 20	Yrs 21 - 30	Yrs 31 – 40	Yrs 41 - 50
Gross Revenue: 4E	0	0	0	0	2 776	4 641	6 294	7 162	7 110	6 832	69 676	61 748	62 567	27 085
Add Base Metal Credits	0	0	0	0	404	625	895	1 023	1 013	980	11 027	9 737	13 849	5 555
Less Selling Expenses	0	0	0	0	(607)	(961)	(1 309)	(1 484)	(1 467)	(1 413)	(14 734)	(13 073)	(14 529)	(6 091)
Less On-Mine OpEx	0	0	0	0	0	0	(2 754)	(2 953)	(3 031)	(3 008)	(28 595)	(31 877)	(27 465)	(11 926)
Less Project CapEx	(434)	(854)	(3 109)	(4 413)	(3 471)	(4 277)	0	0	0	0	0	0	0	0
Less Sustaining CapEx	0	0	0	0	0	0	(1 213)	(666)	(1 090)	(1 276)	(4 655)	(9 246)	(2 783)	(687)
less Working Capital	0	0	0	0	(13)	(23)	303	(3)	(25)	(55)	(382)	(297)	(398)	(494)
Less Corporate Fees and Costs	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Less SLP	0	0	0	0	(24)	(40)	(32)	(38)	(37)	(35)	(381)	(289)	(356)	(175)
Less Payable Royalties	0	0	0	0	(180)	(301)	(412)	(419)	(383)	(337)	(3 896)	(2 269)	(3 464)	(1 542)
Total Undiscounted Cash Flow (Pre-tax)	**(434)**	**(854)**	**(3 109)**	**(4 413)**	**(1 115)**	**(337)**	**1 773**	**2 622**	**2 090**	**1 690**	**28 059**	**14 433**	**27 422**	**11 725**
Less Payable Tax	0	0	0	0	0	0	0	0	0	(28)	(7 964)	(4 125)	(7 790)	(3 626)
Total Undiscounted Cash Flow (Post-tax)	**(434)**	**(854)**	**(3 109)**	**(4 413)**	**(1 115)**	**(337)**	**1 773**	**2 622**	**2 090**	**1 662**	**20 096**	**10 309**	**19 632**	**8 100**
Cumulative Undiscounted Cash Flow (Post-tax)	**(434)**	**(1 288)**	**(4 397)**	**(8 810)**	**(9 924)**	**(10 261)**	**(8 489)**	**(5 866)**	**(3 777)**	**(2 115)**	**17 981**	**28 289**	**47 922**	**56 021**
Discounted Cash Flow (Post-tax)	**(402)**	**(731)**	**(2 466)**	**(3 240)**	**(758)**	**(212)**	**1 033**	**1 414**	**1 043**	**768**	**6 152**	**1 440**	**1 260**	**316**
Cumulative Discounted Cash Flow ($NPV_{8.0\%}$)	**(402)**	**(1 134)**	**(3 600)**	**(6 840)**	**(7 598)**	**(7 809)**	**(6 777)**	**(5 363)**	**(4 320)**	**(3 552)**	**2 601**	**4 041**	**5 301**	**5 616**

22.6 Robustness Analysis

The robustness analysis gauges the robustness of the business case to movements in key drivers. As shown in Table 22-15, each driver is assigned a hypothetical “Bottom,” “Low,” “Base,” “High,” and “Top” case parameter based on the potential movement to be observed in each variable.

Table 22-15: Sensitivity Ranges (% Delta)

ID	Project Driver	UoM	Bottom	Low	Base	High	Top
1	US$ / ZAR	% change	(20.0)	(10.0)	0.0	10.0	20.0
2	Pt Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
3	Pd Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
4	Au Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
5	Rh Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
6	Cu Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
7	Ni Price	% change	(20.0)	(10.0)	0.0	10.0	20.0
8	Payable Metal: Pt	% change	(5.0)	(2.5)	0.0	2.5	5.0
9	Payable Metal: Pd	% change	(5.0)	(2.5)	0.0	2.5	5.0
10	Payable Metal: Au	% change	(5.0)	(2.5)	0.0	2.5	5.0
11	Payable Metal: Rh	% change	(5.0)	(2.5)	0.0	2.5	5.0
12	Payable Metal: Cu	% change	(5.0)	(2.5)	0.0	2.5	5.0
13	Payable Metal: Ni	% change	(5.0)	(2.5)	0.0	2.5	5.0
14	Contractual Discount: Cu	% change	10.0	5.0	0.0	(5.0)	(10.0)
15	Contractual Discount: Ni	% change	10.0	5.0	0.0	(5.0)	(10.0)
16	Handling &Transport Costs	% change	10.0	5.0	0.0	(5.0)	(10.0)
17	Grade: 4E	% change	(8.0)	(4.0)	0.0	4.0	8.0
18	Grade: Base	% change	(8.0)	(4.0)	0.0	4.0	8.0
19	Recovery: 4E	% change	(5.0)	(2.5)	0.0	2.5	5.0
20	Recovery: Base	% change	(5.0)	(2.5)	0.0	2.5	5.0
21	Metal in Concentrate: 4E	% change	(5.0)	(2.5)	0.0	2.5	5.0
22	CapEx: Project	% change	10.0	5.0	0.0	(5.0)	(10.0)
23	CapEx: Sustaining	% change	10.0	5.0	0.0	(5.0)	(10.0)
24	OpEx: External Services	% change	10.0	5.0	0.0	(5.0)	(10.0)
25	OpEx: Overheads	% change	10.0	5.0	0.0	(5.0)	(10.0)
26	OpEx: Labour	% change	10.0	5.0	0.0	(5.0)	(10.0)
27	OpEx: Materials and Supplies	% change	10.0	5.0	0.0	(5.0)	(10.0)
28	OpEx: Utilities	% change	10.0	5.0	0.0	(5.0)	(10.0)
29	Discount Rate	% change	20.0	10.0	0.0	(10.0)	(20.0)

Table 22-16 shows the sensitivity ranges for the three-year trailing average metal price scenario expressed in each driver’s respective unit of measure.

Table 22-16: Sensitivity Ranges (Units)

ID	Driver	Unit of Measure	Bottom	Low	Base	High	Top
1	US$ / ZAR (LT Real)	ZAR Real	12.76	14.35	15.95	17.54	19.14
2	Pt Price	US$ / ozt	745	838	931	1 024	1 117
3	Pd Price	US$ / ozt	844	949	1 055	1 160	1 266
4	Au Price	US$ / ozt	1 054	1 186	1 318	1 450	1 582
5	Rh Price	US$ / ozt	1 544	1 737	1 930	2 123	2 316
6	Cu Price	US$ / lb	2.30	2.58	2.87	3.16	3.44
7	Ni Price	US$ / lb	4.45	5.00	5.56	6.12	6.67
8	Payable Metal: Pt	%	80.8	82.9	85.0	87.1	89.3
9	Payable Metal: Pd	%	80.8	82.9	85.0	87.1	89.3
10	Payable Metal: Au	%	80.8	82.9	85.0	87.1	89.3
11	Payable Metal: Rh	%	80.8	82.9	85.0	87.1	89.3
12	Payable Metal: Cu	%	69.4	71.2	73.0	74.8	76.7
13	Payable Metal: Ni	%	64.6	66.3	68.0	69.7	71.4
14	Contractual Discount: Cu	US$ / t	220	210	200	190	180
15	Contractual Discount: Ni	US$ / t	110	105	100	95	90
16	Handling andTransport Costs	US$ / wmt	468	446	425	404	383
17	Grade: 4E	g/t	2.97	3.10	3.23	3.36	3.49
18	Grade: Base	%	0.24	0.26	0.27	0.28	0.29
19	Recovery: 4E	%	74.91	76.88	78.85	80.82	82.79
20	Recovery: Base	%	56.56	58.05	59.54	61.02	62.51
21	Metal in Concentrate: 4E	g/t	75.91	77.90	79.90	81.90	83.90
22	CapEx: Project	ZAR mil Real	18 214	17 386	16 559	15 731	14 903
23	CapEx: Sustaining	ZAR mil Real	23 778	22 697	21 617	20 536	19 455
24	OpEx: External Services	ZAR / t ore milled	18	18	17	16	15
25	OpEx: Overheads	ZAR / t ore milled	6	6	6	5	5
26	OpEx: Labour	ZAR / t ore milled	158	151	143	136	129
27	OpEx: Materials & Supplies	ZAR / t ore milled	339	323	308	292	277
28	OpEx: Utilities	ZAR / t ore milled	134	128	122	116	110
29	Discount Rate	%	9.60	8.80	8.00	7.20	6.40

22.6.1 Deterministic Sensitivity Analysis

The sensitivity analysis is performed on the three-year trailing average price scenario. The sensitivity analysis iterates through the theoretical “bottom” and “top” case parameters for each driver identified in Table 22-15 and subsequently plots the incremental NPV and IRR that results from the discrete movements in each driver. Figure 22-7 and Figure 22-8 present the incremental impact on the NPV (R5 616 M) and IRR (13.3%), respectively.

Figure 22-7: Deterministic Sensitivity Analysis – Net Present Value

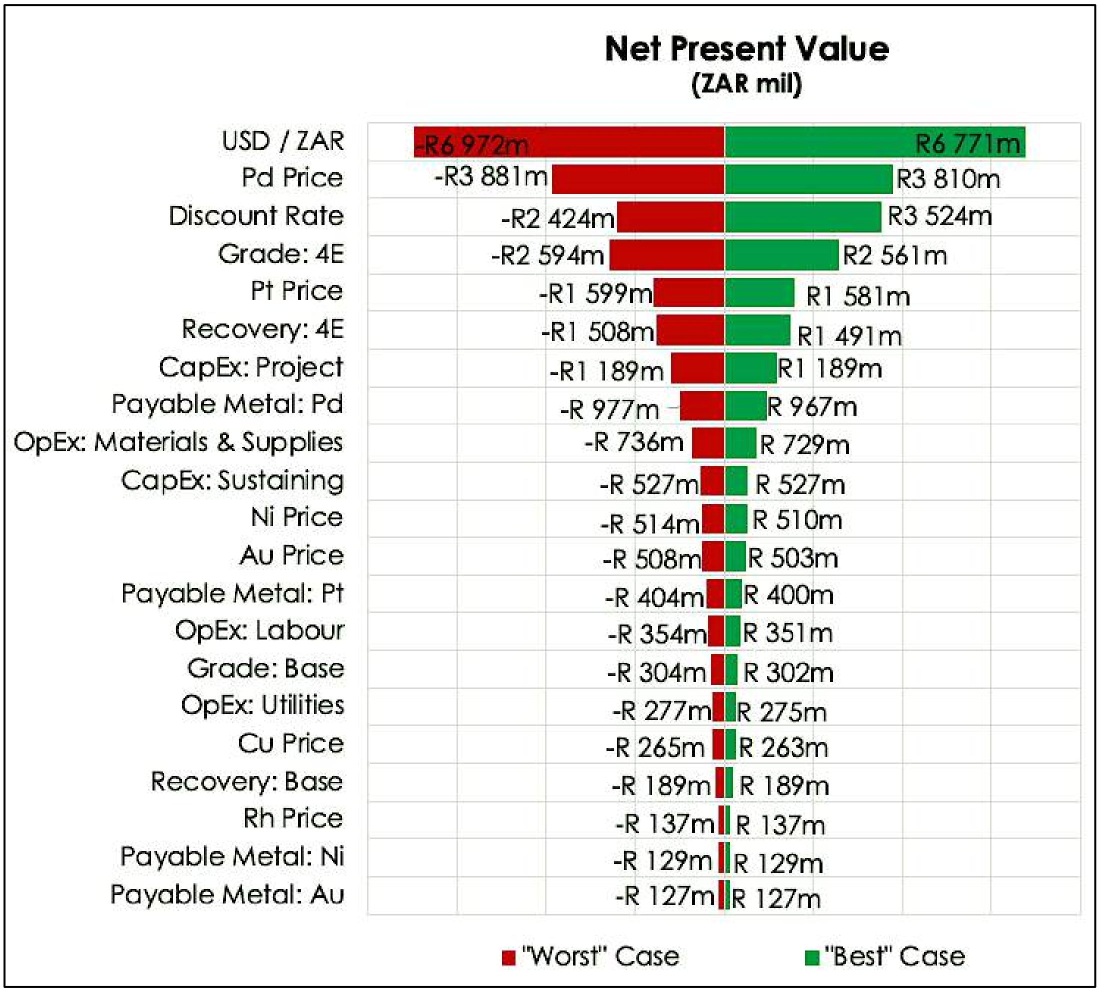
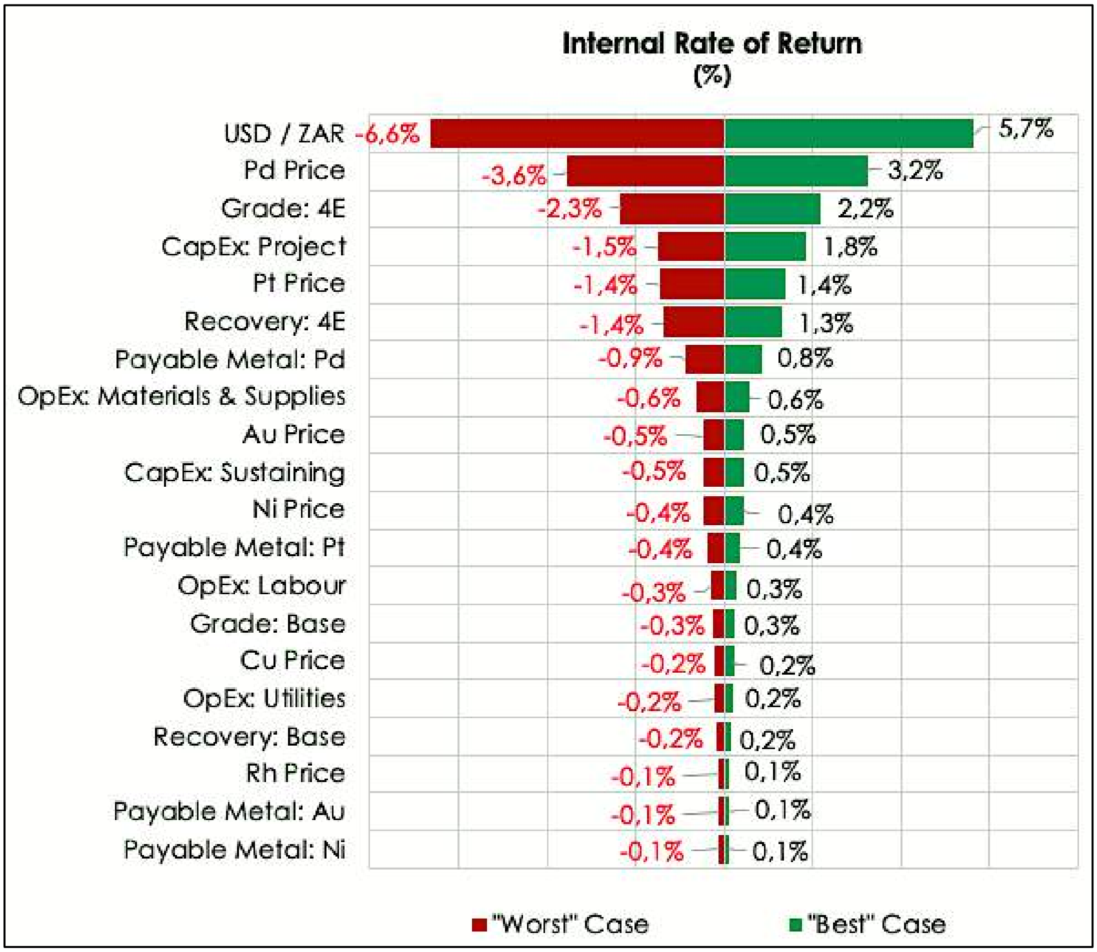


Figure 22-8: Deterministic Sensitivity Analysis – Internal Rate of Return



The NPV is most sensitive to movements in the following key drivers.

1. US$/ZAR rate
2. Pd Price
3. Discount Rate
4. 4E Grade
5. Pt Price

A 20% depreciation of the ZAR against the US$ results in an $NPV_{8\%}$ addition of R6 771 M, which would increase the base $NPV_{8\%}$ from R5 616 M to R 12 388 M. A 20% appreciation of the ZAR would result in a negative business case (NPV<0).

The three-year trailing Pd price (US$1 055/oz) is 47% lower than the spot price at 4 September 2019 (US$1 546/oz) and 27% lower than the one-year trailing average price (US$1 343/oz). The sensitivity analysis highlights the significant benefit of a 20% increase in the Pd price, which improves the base $NPV_{8\%}$ by R3 810 M.

The range of expected movement in the 4E head grade is narrower than the macroeconomic parameters (e.g. FX and price); therefore, the impact on the business case is limited. The tornado chart in Figure 22-7 illustrates that an 8% decrease in the LOM average 4E grade would result in an $NPV_{8\%}$ erosion of R2 594 M, which would still yield a positive business case (NPV>0) for the three-year trailing price scenario.

The Project IRR is most sensitive to movements in the following key drivers.

1. US$/ZAR Rate
2. Pd Price
3. 4E Grade
4. Project CapEx
5. Pt Price

Project CapEx affects the IRR to a greater extent than the NPV since IRR is largely affected by the first 10 years of free cash flow. If the project team can reduce the upfront capital outlay requirement by 10% (through further value engineering activities), it is possible to improve the $NPV_{8\%}$ and IRR by R1 189 M and 1.8%, respectively.

22.6.2 Deterministic Scenario Analysis

The scenario analysis is performed on the three-year trailing average price scenario. The purpose of the scenario analysis is to deterministically evaluate and analyse how a combination of macroeconomic and project economic scenarios can influence key business metrics. This is achieved by labelling each key project driver as either an exogenous or endogenous variable as shown in Table 22-17. An exogenous variable is not typically within the reasonable control of the project team (e.g. metal prices). An endogenous variable is largely within the reasonable control and influence of the project team (e.g. on-site costs).

Table 22-17: Exogenous and Endogenous Variables

Exogenous Variables	Endogenous Variables
Foreign Exchange Rates	CapEx
Metal Prices	OpEx
Smelter Payability and Discounts	Metallurgical Recoveries
Ore Grades	Grade in Concentrate
	Discount Rate

Utilising the “Low,” “Base,” and “High” case parameters for each driver in Table 22-15 and Table 22-16, respectively, shows the sensitivity ranges for the three-year trailing price scenario expressed in each driver’s respective unit of measure. A combination of scenarios are evaluated to determine the robustness of the business case to movements in exogenous variables and the extent to which the project team is able to effectively control the endogenous variables to ensure sustained profitability.

Table 22-18 defines the nine combination of scenarios evaluated and Table 22-19 documents the subsequent key metrics for each of these scenarios.

The analysis shows that the Waterberg Project is value accretive in seven out of the nine scenarios considered, which is indicative of a fairly robust business case. Both scenarios that generate a negative business case (NPV<0) occurs under weak market conditions (exogenous parameters = low). However, considering the metal prices in the three-year trailing average scenario relative to the current spot prices, it is highly unlikely that the Waterberg Project would experience a further weakening of market conditions than what is already provisioned for in the base case.

The value engineering case highlights the importance of good execution, governance, and operational performance. In the value engineering scenario, the IRR increases from 13.3% to 15.7%, which is largely attributable to a 5% collective decrease in CapEx and OpEx and a 2.5% increase in metallurgical recoveries.

Table 22-18: Definition of Scenarios

		Endogenous Parameters		
		High	**Base**	**Low**
Exogenous Parameters	**High**	Favourable Market Conditions Excellent Project Performance **(Theoretical Best Case)**	Favourable Market Conditions Planned Project Performance	Favourable Market Conditions Poor Project Performance
	Base	Forecasted Market Conditions Excellent Project Performance **(Value Engineering Case)**	Forecasted Market Conditions Planned Project Performance **(Base Case)**	Forecasted Market Conditions Poor Project Performance
	Low	Weak Market Conditions Excellent Project Performance	Weak Market Conditions Planned Project Performance	Weak Market Conditions Poor Project Performance **(Theoretical Worst Case)**

Table 22-19: Scenario Analysis Results

		Endogenous Parameters		
		High	**Base**	**Low**
Exogenous Parameters	**High**	NPV: R21 454 M	NPV: R15 569 M	NPV: R10 730 M
		IRR: 24.1%	IRR: 21.4%	IRR: 18.8%
		MNCF: R8 528 M	MNCF: R9 221 M	MNCF: R9 914 M
		Payback: 7.8 yrs	Payback: 8.2 yrs	Payback: 8.9 yrs
	Base	NPV: R9 899 M	NPV: R5 616 M	NPV: R2 069 M
		IRR: 15.7%	IRR: 13.3%	IRR: 10.9%
		MNCF: R9 270 M	MNCF: R10 261 M	MNCF: R11 252 M
		Payback: 10.0 yrs	Payback: 11.2 yrs	Payback: 12.6 yrs
	Low	NPV: R133 M	NPV: -R2 951 M	NPV: -R5 529 M
		IRR: 7.3%	IRR: 4.9%	IRR: 2.5%
		MNCF: R10 865 M	MNCF: R11 817 M	MNCF: R12 769 M
		Payback: 15.2 yrs	Payback: 17. 5yrs	Payback: 34.8 yrs

A similar analysis was performed on the Waterberg Project at spot metal prices. The scenario analysis yielded a positive business case (NPV>0) in each of the nine scenarios evaluated and is considered highly robust.

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 has distinctively different metal ratios with elevated Au 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 Aurora Project (Pan Palladium)

The historical Aurora Project comprised the farms, Kransplaats, Nonnenwerth, La Pucella and Altona. This was managed by Pan Palladium at the time and they reported Mineral Resources of 50 Mt at 1.19 g/t (2PGE+Au), 0.07% Ni, 0.21% Cu (Pan Palladium Annual Report, 2003). The QP 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 mineralisation on the property that is the subject of this technical report. An updated estimate was published in the 2010 Sylvania Resources Ltd Competent Persons report. The report reflects a combined Inferred Mineral Resource of 133 Mt and 5.7 Moz 2E +Au (1.34g/t 2E+Au, 0.05% Ni and 0.08% Cu). Pan Palladium South Africa (Pty) Limited is now a subsidiary of Sylvania Platinum Limited. The 2018 Sylvania Platinum Ltd. Annual Report reflects that consent was received, in terms of Section 11 of the Mineral and Petroleum Resources Development Act, to cede the rights to mine heavy minerals, Fe ore, and V ore on the farms Nonnenworth, La Pucella and Altona to Lapon Mining (Pty) Ltd, a subsidiary of Ironveld PLC.

23.2 Mogalakwena Mine

Located 30 km northwest of Mokapane and approximately 60 km south of the Waterberg Project is the world's largest opencast Pt mine, Mogalakwena Mine (formerly Potgietersrust Platinum Mine), which mines the Platreef and produced 1.170 Moz PGMs in concentrate in 2018. The Mineral Resource inclusive of Ore Reserves reported at the end of 2018 was 3 683.5 Mt and 293.3 (4E) Moz. The latest Mineral Resource and Ore Reserve Statement for Mogalakwena Mine is available at www.angloplatinum.com and Anglo Platinum Annual Report 2018.

It was announced on 27 August 2019 that Anglo American Platinum and Atlatsa completed the acquisition and inclusion of the resources specified in the Central Block and Kwanda North PRs into Rustenburg Platinum Mine's Mogalakwena mining right. The Kwanda North and Central Block PRs are adjacent to and have been incorporated into the Mogalakwena mining right. The PRs have not yet been classified as Mineral Resource.

23.3 Akanani Project

Sibanye-Stillwater holds the majority interest (74%) in the Akanani Project. The Akanani Project is down dip of the Mogalakwena Mine and is an exploration project with studies continuing to develop it into a viable operation. As of 30 September 2018, they have declared an attributable Mineral Resource of 233.1 Mt at a 4E grade of 3.90 g/t with 12.0 Moz and no Mineral Reserve was declared. Information pertaining to this project, including the latest Mineral Resource and Mineral Reserve Statement are available in their 2018 Mineral Resource and Mineral Reserve statement on the Sibanye-Stillwater website (www.sibanyestillwater.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 JV with Anglo Platinum in 2004. This project now belongs to Anglo Platinum following a 2013 asset sale.

Historically, exploration drilling was conducted at the project site, which led to the estimate of Indicated and Inferred Mineral Resources. A Mineral Resource was declared in December of 2004, which stated an Indicated Mineral Resource of 176.6 Mt, contained 7.65 Moz PGM and Inferred Mineral Resource of 104.1 Mt, contained 4.12 Moz PGM. For more details on the Mineral Resource refer to the December 2004 Technical Report by GJ Van der Heever of GeoLogix (Pty) Ltd. A preliminary economic assessment was completed in 2005. The Boikgantsho Mineral Resource Estimate is included the Mogalakwena Mine Mineral Resource Estimate by Anglo Platinum since 2017. The 2017 Anglo Platinum Mineral Resource and Ore Reserve report reflected the estimate for Boikgantsho as 83.4 Mt containing 3.4 Moz 4E.

23.5 Aurora, Harriet's Wish and Cracouw Projects (Hacra Project)

These three exploration projects (combined known as the Hacra Project) were 71% owned by Great Australian Resources Ltd. and 29% owned by Sika Bopha in 2009. Great Australian Resources was 16% held by Sylvania Resources Limited and in October 2009 operated as a 100% subsidiary of Sylvania Platinum Limited. The combined projects had a "Possible" Mineral Resource of 0.9 Moz of PGMs as stated in Sylvania Resources Limited February 2009 Fact Sheet. Sylvania undertook 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 like that of mineralisation found in the Waterberg Project. The author has not been able to verify this data. More information on these projects can be found on the Sylvania Platinum Website (www.sylvaniaplatinum.com). Ironveld PLC owns the rights to heavy minerals, Fe ore, and V ore on these projects (www.ironveld.com).

23.6 Platreef Project (Ivanplats)

The Platreef Project is owned by Ivanplats (Pty) Ltd, a subsidiary company of Ivanhoe Mines Ltd. The ownership in Ivanplats is jointly held by Ivanhoe (64%), the Japanese consortium of Itochu Corporation, JOGMEC, and JGC Corporation (10%) and a BEE entity (26%). The Platreef Project is a recently discovered underground deposit of thick, PGM-Ni-Cu 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: Turfspruit, Macalacaskop, and Rietfontein.

The Platreef Project's first shaft (Shaft No. 1) was extended to a depth of 855 m below surface. The 850-m level station was approximately 70% complete at the end of March 2019. The 850-m level station, as well as the already completed 750-m-level, will provide underground access to the high-grade ore body, enabling mine development to proceed. As sinking of Shaft No. 1 advances, one more station will be developed at a depth of 950 m. Shaft No. 1 is expected to reach its projected, final depth of 982 m below surface in early 2020.

Surface construction for Platreef's Shaft No. 2 is focused on the concrete foundation (hitch) for the headframe, which was completed in mid-2019. Shaft No. 2 will have an internal diameter of 10 m and will be equipped with two 40-tonne rock-hoisting skips with a capacity to hoist a total of six million tonnes of ore per year.

Ivanplats delineated a large zone of mineralisation within the Platreef, which essentially comprises a steeply-dipping, near-surface mineralised area and a gently-dipping to sub-horizontal (<15º) deeper zone from approximately 700 m depth downward to 1 500 m (the "Flatreef"). Ivanhoe completed an FS in September 2017. The mineralisation is considered open for expansion along the southern and western boundaries of the Platreef 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.ivanhoemines.com).

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that the QPs are aware of that is material to this Technical Report.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 Geology and Mineral Resource

Additional infill drilling in the indicated Mineral Resource category areas resulted in portions of the Mineral Resources being upgraded to the measured Mineral Resources category.

The estimate was completed using best practices in terms of geostatistics.

The objectives in terms of adherence to the scope of this DFS were met in that an updated Mineral Resource Model was produced. An objective of converting indicated Mineral Resources from the previous estimates to the higher confidence of measured was also completed. Cutoffs using previous estimates of costs and recoveries from the PFS were utilised for this Mineral Resource Estimate with updated price decks.

The delineation of the F Zone and T Zone units was advanced due to better understanding of the geology. The T Zone was divided into three distinct layers, TZ, T1, and T0.

The database used for this estimate consisted of 441 drill holes and 583 deflections. The mineralisation is considered open down-dip and along strike to the north.

The Waterberg Project represents one of the largest discoveries of 4E mineralisation in recent history. Metallurgical work completed to date at Mintek along with the work in this DFS adds to the confidence in this discovery.

The M&I Mineral Resources are at an appropriate level of confidence to be considered in the DFS for mine planning.

25.2 Mineral Reserve Estimate

The estimated Mineral Reserve for the Waterberg Project at a 2.5 g/t 4E stope cutoff grade includes a combined 187.5 million tonnes at an average grade of 3.24 g/t 4E, 0.09% Cu, and 0.18% Ni in the proven and probable categories. Individual stope and development mining shapes were created and include planned dilution and modifying factors to account for geological losses, external overbreak dilution, and mining losses. The estimated Mineral Reserves are supported by a mine plan and economic analysis and demonstrate positive economics.

The following risks could potentially impact the estimated Mineral Reserves.

- Approximately 75% of the Mineral Resource at a 2.5 g/t 4E cutoff is in the indicated category. If not all the indicated material is successfully upgraded, the estimated Mineral Reserves could be reduced.
- Metal prices are subject to fluctuation. Lower than anticipated metals prices could increase the stope cutoff grade and reduce the estimated Mineral Reserves.
- Currency fluctuations could increase the stope cutoff grade and reduce the estimated Mineral Reserves.

25.3 Mining Methods

The geometry and continuity of the mineral resource and the rock mass quality of the mineralized zones and surrounding rock mass make the Waterberg zones amenable to extraction using the Sublevel Longhole Stoping mining method using paste backfill. The mine design includes all development and infrastructure required to access the Central, South, and North Complexes and mine the estimated Mineral Reserves. A full 3D mine model was created for each complex and a LOM development and production schedule was prepared to determine the estimated tonnes, average grade, and metals profile mined and delivered to surface.

Initial production will come from the simultaneous operation of the Central and South Complexes, with the North Complex phased in once production in the Central and South Complexes begins to ramp down. There will be approximately five years of ramp up from the start of the decline development in 2021 to achieve sustainable 70% of steady-state production in January 2026. Steady-state production of 400 ktpm will be achieved in Q1 2027 with 300 ktpm from the Central Complex and 100 ktpm from the South Complex. Later in the mine life, the North Complex will ramp up to sustain 400 ktpm production. The mine will produce for approximately 44 years from first ore to the end of mine life.

The development methods and mining methods are safe and highly mechanized and use common equipment and processes that are proven and used successfully in the global mining industry. The successful execution of these methods to achieve planned underground mine development and production at the Waterberg Project will require the operation to establish a culture focused on worker health and safety, investment and emphasis on worker skills training geared toward the equipment and technology used, and structured mine planning.

25.4 Metallurgical Performance and Processing

Metallurgical testwork was conducted to select the preferred process flowsheet to be followed for the recovery of 4E metals with associated Cu and Ni. The selected flowsheet is the MF2 flotation concentrator circuit, which is well understood in the South African PGE industry and especially on similar ores to Waterberg. The testwork at PFS level was based upon blended and composited samples to select the flowsheet whilst during the DFS, variability samples were evaluated to confirm the grade-recovery relationship. The tests included comminution evaluation, flotation with reagent optimisation, mineralogical evaluation, and limited settling and thickening trials.

Material was also produced for backfill evaluation using cemented paste tailings from the concentrator.

The flotation evaluation confirmed that the T Zone performs differently from the F Zone with different reagent regimes required for optimal performance; therefore, a controlled metallurgical blend will be required in the concentrator to achieve the best performance. Additional confirmatory locked cycle flotation tests were completed on the anticipated blend of ores to be treated (nominally 25% T Zone and 75% F Zone) in the first 13 years to confirm the plant performance – the following 32 years have only F Zone material in the current mine plan.

The treatment of the ores to be delivered from the mining operations for the first 13 years will be a controlled metallurgical blend of South T Zone material with Central F Zone and South F Zone, depending upon the mining schedule. This is included in the layout and design of the concentrator.

The plant will produce a concentrate containing 80 g/t 4E with a nominal mass pull of 3.2% over LOM. The Cu content in the concentrate will be 2.3% and Ni will be 2.7% over LOM. There will not be any penalty elements in the concentrate; however, the Fe and S contents will require blending in any subsequent smelting operation. The lack of chromite makes this concentrate attractive to smelting operators. The 4E recovery will be almost 79% over LOM with Cu at 83% and Ni at 48%.

The concentrator is designed to process 400 ktpm of ROM ore to produce between 13 000 and 15 000 tpm of concentrate at 12% moisture. The concentrate will be delivered to existing smelters in South Africa for further treatment and refining.

25.5 Infrastructure

The Waterberg Project site is a greenfield location with limited existing regional and local support infrastructure that would be appropriate to the development of the mine. Gravel roads are available with the nearest regional tarmac road 34 km away. Electrical reticulation at 22 kV is available; however, capacity is constrained, although with upgrading, it will be adequate for construction power. The site is dry and all local water comes from drill holes.

To support the operation of the mine, the need for construction of the following infrastructure was identified in the study.

- Electrical Overhead Line 74 km long providing Grid Power at 132 kV
  - Associated Substations and Site Distribution at 11 kV
  - Emergency Power Generation
- Drilling of Drill Holes for Water Supply to the Mine and Plant
  - Associated Distribution Network Collecting Water from Individual Drill Holes to the Mine Site
- Paving and Upgrading of the Main Access Roads to the Nearest Regional Road – 34 km Required
- TSF to Contain 93 mt of Tailings
- Backfill Paste Preparation Plant with Distribution to Multiple Underground Drill Holes

- Waste Rock Storage Facilities
- Temporary Ore Stockpile Facility
- Pipeline and Conveyor Routing / Servitudes between Different Surface Facilities
- General Surface Facilities
  - Offices and Change House
  - Central Assay Laboratory
  - Maintenance Workshops
  - Fuel Facilities
  - Warehousing
  - Construction Camp

The process plant and mining complex infrastructure (including ventilation fans) do not form part of the general infrastructure associated with the Waterberg Project.

The design and construction of these infrastructure facilities are costed in the capital estimate.

25.6 Marketing and Contracts

The Waterberg Project is a significant Pd producer and with the international trend towards reduced Pt consumption and increased Pd usage, the price of the metals were extremely volatile in 2019. Rh and Au with Ni have increased significantly in price along with Pd while Pt and Cu remained stable. The outlook for the next few years is uncertain, but the trend is expected to remain with Pd being deficient for the foreseeable future.

The concentrate being produced by Waterberg is desirable with insignificant chromite content, an acceptable 80 g/t 4E grade, and acceptable Cu and Ni content. The tonnage of concentrate to be delivered with the contained Cu and Ni may stress the receiving smelter and base metal refining capacity.

No off-take agreements have been negotiated but the project team determined that a reasonable payability for the contained economic metals would be 85% for all 4E elements, 73% for Cu and 68% for Ni. These payabilities are comparable to industry norms within South Africa without any treatment and/or refining charges. The metals would be released after 12 weeks and the project has modelled an 85% up-front payment with the balance being received after the 12 weeks, albeit incurring an interest charge on the up-front payment.

One significant LOM contract that must be negotiated is the power supply agreement with ESKOM. ESKOM agreed to the supply and installation technical requirements and environmental approval has been obtained along with final negotiations for servitudes; however, the formal agreement is required.

25.7 Environmental

A multi-agency licensing and authorisations process will be completed by Waterberg JV Resources to construct, operate, and close the Waterberg Project in accordance with all applicable legislation. This programme will include the acquisition of several licenses and authorisations from various regulatory agencies. An analysis of the permitting process, proposed path or work done to date suggests no permitting issues are presented that would halt the Waterberg Project.

The environmental investigations highlighted the following risks.

- Mining activities could affect local groundwater flow due to groundwater abstraction activities, which could lower the water table affecting local drill holes. 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 TSF 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. The Waterberg Project area aesthetics 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.
- 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. Part of respecting villagers and their traditional beliefs is 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.

25.8 Capital and Operating Costs

Capital and operating costs were developed from first principals for the technical disciplines associated with the Waterberg Project. Project capital is defined as the expenditure required to achieve 70% of steady-state production, expected to be December 2025, if the project commences in Q1 2020. The capital cost determined is shown in Table 25-1. The capital was developed in ZAR and concerted to US$ at an exchange rate of 15.00.

Table 25-1: Waterberg Project Capital Cost

Cost Area	ZAR Total (ZAR M)	USD Total (US$ M)
Underground Mining	6 097	406
Concentrator	2 580	172
Shared Services and Infrastructure	682	45
Regional Infrastructure	1 229	82
Site Support Services	234	16
Project Delivery Management	654	44
Other Capitalised Costs	331	22
Contingency	1 298	87
Total Project Capital (excluding Capitalised OpEx)	**13 105**	**874**
Capitalised Operating Costs	3 453	230
Total Project Capital (including Capitalised OpEx)	**16 559**	**1 104**

The capital estimate was developed to a Class 2 level of detail indicating an accuracy of -10%/+15%

The SIB expenditure covers all expenditure of a capital nature following the achievement of 70% of the steady-state production. This includes all ongoing underground waste development, construction of the North Complex, required infrastructure, mobile equipment replacement, and other items of a capital nature associated with the concentrator and general mine infrastructure. The total SIB provision is R21.6 billion spread over the more than 40 years of mine life.

The LOM operating costs following achievement of 70% of steady-state production and excluding SIB expenditure is summarised in Table 25-2.

Table 25-2: Waterberg Project Operating Cost

Cost Area	LOM Average (ZAR/t milled)	LOM Average (US$/t milled)
Mining	345	23.01
Milling and Processing	132	8.79
Engineering and Infrastructure	116	7.76
General and Administration	19	1.25
Total On-site Operating Costs	**612**	**40.80**

The cash cost per 4E ounce is estimated at US$640 (spot prices) and US$554 (three-year trailing prices), respectively. The cash cost includes the smelter discount as a cost, as well as byproduct credits from Cu and Ni sales; therefore, the indicated cash costs are dependent on the prevailing metal price assumptions as detailed in Table 25-3.

Table 25-3: Waterberg Project Cash and All-In-Cost

Metric	Spot Prices (US$ / 4E oz)	Three-year Trailing Prices (US$ / 4E oz)
On-site Operating Costs	487	456
Smelting, Refining, and Transport Costs	302	227
Royalties and Production Taxes	88	54
Less Byproduct Base Metal Credits	(236)	(184)
Total Cash Cost	**640**	**554**
Sustaining Capital	94	88
Total All-in Sustaining Cost	**734**	**642**
Project Capital	34	32
Total All-in Cost	**767**	**674**

The estimated cash cost for the Waterberg Project will deliver a mine in the lower quartile of PGE producers in Southern Africa.

25.9 Economic Outcome

The metal prices used in the economic evaluation are three-year trailing price and the spot price as at 04 September 2019. As the input costs were developed in ZAR terms, the appropriate rate of exchange applied must be considered when converting from ZAR to US$. The price assumptions are detailed in Table 25-4 and the corresponding exchange rates are R15.00 to 1 US$ for the spot price scenario and the Bloomberg nominal consensus as at June 2019, which translates into a long term real US$/ZAR forecast of R15.95 for the three-year trailing price scenario.

Table 25-4: Metal Price Scenarios

Factor	Unit of Measure	Spot Prices	Three Year Trailing Average Prices
Pt	US$ / oz (real July 2019)	980.00	931.00
Pd	US$ / oz (real July 2019)	1 546.00	1 055.00
Au	US$ / oz (real July 2019)	1 548.00	1 318.00
Rh	US$ / oz (real July 2019)	5 036.00	1 930.00
Basket Price (4E)	***US$ / oz (real July 2019)***	**1 425.00**	**1 045.00**
Cu	US$ / lb (real July 2019)	2.56	2.87
Ni	US$ / lb (real July 2019)	8.10	5.56

The Waterberg Project produces a positive business case in both the spot and three-year trailing average metal price scenarios. At spot prices, the Waterberg Project yields a post-tax NPV8.0% of R14 736 M (US$982 M), at an IRR of 20.7%, an undiscounted payback period of 8.4 years, and a peak funding requirement of R9 255 M (US$617 M). At three-year trailing average metal prices, the project yields a post-tax NPV8.0% of R5 616 M (US$333 M), at an IRR of 13.3%, an undiscounted payback period of 11.2 years, and a peak funding requirement of R10 261 M (US$667 M).

At the two pricing scenarios (spot and three-year trailing average), the project generates LOM average cash costs of US$640 / 4E oz and US$554 / 4E oz, respectively, which places Waterberg firmly within the lowest quartile of regional PGE producers.

25.10 Overall Conclusions

The Waterberg Project will be a fully mechanised, shallow, decline-accessed mine and will be one of the largest and potentially lowest cash cost underground PGM mines globally.

26 RECOMMENDATIONS

26.1 Geology and Mineral Resource

Further drilling work could be capable of converting the inferred Mineral Resources to a higher category, but at this time, it is likely that future drilling may be focused on other areas and items like geotechnical characteristics for mine planning, ongoing operational Mineral Resource definition and delineation, or detailed metallurgical work. Given the variable ore body, it is recommended that ongoing geological drilling ahead of mining be prioritized to ensure optimal extraction.

It is recommended that dedicated Mineral Resource definition drilling from both surface and underground be completed. The main objective of the Mineral Resource definition drilling is to upgrade indicated Mineral Resources to measured Mineral Resources. Such infill surface Mineral Resource definition will be completed in initial years until the mine is established to allow access for underground Mineral Resource definition drilling well in advance of stoping. Capital provision will be made for infill Mineral Resource definition drilling to depths of approximately 700 m below surface.

Dedicated underground delineation drilling is described in Section 16.3.12. The variable ore bodies demand the need to continuously delineate the stopes for mine planning and grade control. The delineation diamond drilling will be completed from drill cut-outs spaced along the footwall drifts on sublevels and from other pre-developed excavations, including remuck bays in the declines. Sufficient mine development will be scheduled and in place ahead of the advancing production fronts to ensure adequate time for definition diamond drilling and subsequent Mineral Resource model updates and mine planning. Diamond drilling will be completed from the service decline and footwall drift to define the placement of sublevel infrastructure and stope sills.

Currently, only the larger structures have been modelled. It is recommended that a detailed structural analysis is done and modelled in 3D space.

26.2 Mineral Reserve Estimates

Mineral reserves are reported at a 2.5 g/t 4E stope cutoff grade. There is M&I resource material below the stope cutoff that is not included in the mine plan but is adjacent to planned development and stoping areas. A lower cutoff grade could potentially bring this material into the mine plan with incremental additional development and add to the Mineral Reserves. It is recommended to evaluate the potential for reducing the stope cutoff grade.

There is Mineral Resource that is above cutoff that could not be included in a longhole stope shape due to local geometry. This material could be amenable to mining using Cut and Fill or Board and Pillar methods. It is recommended to determine the stoping cutoff for this material and evaluate the potential to include this material in the mine plan and add to the Mineral Reserves.

It is recommended that the definition drilling and delineation drilling programs described in Section 26.1 are conducted and an updated mine plan is maintained to reflect the changes in the estimated Mineral Reserves.

26.3 Mining Methods

The current mine design is based on using diesel-powered underground mobile equipment. There have been significant advances in battery technology and the development of battery-powered mobile equipment. It is recommended to monitor the progress and application of the technology during the mine access development period and assess the opportunities this technology could present to the Waterberg Project, which may include reduced ventilation and refrigeration requirements, smaller diameter or fewer ventilation raises, and reduced electrical power consumption.

It is recommended that the following geotechnical and geomechanical work is completed as part of project execution to validate mine design assumptions and support the detailed design for infrastructure.

- Conduct systematic geomechanical logging of future diamond drill core to further develop the database used for rock mass classification.
- Conduct additional laboratory testing of future diamond drill core for rock mass properties.
- Conduct in situ stress measurements to confirm assumptions used in the geomechanical model.
- Drill geomechanical holes at each surface ventilation raise location to determine ground conditions and assess the stability of the 6.0 m diameter raises. Investigate alternate locations to position ventilation raises to reduce the depth of overburden and/or weathered Waterberg Sediments at the raise collar.
- Drill geotechnical holes at each box cut location to collect additional data, including the orientation of jointing and structures, for detailed engineering of the box cuts.
- Drill geotechnical holes along the path of the Main Declines from surface to further assess the ground conditions that will be encountered and confirm development advance rates and schedules.
- Conduct geomechanical mapping of excavations to further develop the database for rock mass classification.
- It is also recommended to review the mine stope sequencing in the lower portions of the mine that are mined later in the mine life. Optimizing the mining sequence could reduce the amount of ground deformation. This should be performed as more detailed rock mechanics information is obtained through the mining process.

26.4 Metallurgical Processing

The 400 ktpm concentrator plant is considered to be the most suitable design based on the current mine production schedule.

It is recommended that the following additional metallurgical testwork is completed during project execution.

- Further flotation testwork to confirm the effect of the available groundwater on flotation performance and to determine what adjustments to the raw water circuit would be required (if any).
- Concentrate thickening and filtration testwork.
- Further tailings thickening and filtration testwork for confirmation of backfill plant design criteria.

It is recommended that future consideration be given to the opportunities related to deferring some of the plant capital cost by phasing the installation of some mechanical equipment such as one of the concentrate filters and some flotation cells. Additionally, operating the plant as an MF1 circuit with a single mill during the production ramp-up would defer capital cost. While this approach was considered a suboptimal outcome as the MF1 configuration results in lower recoveries, it is recommended that the trade-off be revisited to account for the conditions prevailing at the start of the concentrator project execution.

26.5 Infrastructure

26.5.1 Central Assay Laboratory

It is recommended that the analytical requirements for the geological controls for the mining operation as well as the ad-hoc sampling requirements be confirmed to improve the specification for the sizing of the analytical laboratory.

26.5.2 Tailings Storage Facility

The following recommendations are provided for the TSF detailed design phase.

- Confirm design criteria and site selection.
- Further analysis and design the stream diversion.
- Further optimization of the capital and operating cost estimate, where possible, by completing the following tasks.
  - Develop a tender enquiry on the detailed design to acquire final construction rates.
  - Further optimization of earth and civil works, where possible.
  - Finalise operator responsibilities by incorporating input from all parties (contractor, client, and consultants).
- Further evaluation of geochemical risk in terms of liner requirements / details.
- Confirmation of survey data accuracy. It is recommended to complete survey points of the site to confirm elevation.
- Further geotechnical assessments of the collapsible soils, including impact roller testing to determine its effectiveness.

- Continued monitoring of the risks relating to the following items.
  - Collapsible soils.
  - Severe desiccation cracking.

The geotechnical studies identified possible sources of clay on site, an opportunity exists to reduce the liner cost should the project be able to obtain permissions to explore and exploit this source of material in sufficient quantities.

26.6 Marketing and Contracts

It is recommended that the off-take agreement for the concentrate with associated net smelter return be negotiated with IMPLATS with the right of first refusal for the project or other interested parties.

The power supply agreement with ESKOM should be finalised as well as the design and construction contracts with a considerable number of smaller contracts for services, including concentrate transport from mine site to the smelter.

26.7 Environmental

It is recommended Waterberg JV Resources continue their current permitting strategy to develop positive community support and streamline final project approval as outlined below.

- Maintain regular consultation activities with all appropriate national, provincial, and local regulatory agencies.
- Maintain engagement with local communities. These meetings are beneficial in developing and maintaining community support by being transparent on social and economic aspects of the Waterberg Project. They also provide a forum to identify and address concerns, which will allow issues to be addressed at the earliest possible opportunity and avoid potential delays.
- Hold regular meetings with appointed and elected local, provincial, and national officials. These types of meetings provide the opportunity to keep key officials updated on development, and set the stage for political assistance, if needed, at the local, provincial, and national levels.

Waterberg has a programme of work in place to comply with the necessary environmental, social, and community requirements. Following is key work that should continue.

- ESHIA in Accordance with the MPRDA and NEMA
- Public Participation Process in Accordance with the NEMA
- Specialist Investigations in Support of the ESHIA
- Integrated WUL Application in Compliance with the National Water Act
- Integrated WML in Compliance with the National Environmental Management Waste Act

26.8 Economic Outcome

Based on the positive economics from the technical inputs and the financial analysis, it is recommended that the Waterberg Project be considered by the members of the Waterberg JV for an investment decision.

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Appendix A
Comparison of Definitive Feasibility Study to 2016 Prefeasibility Study

PFS vs DFS Comparison

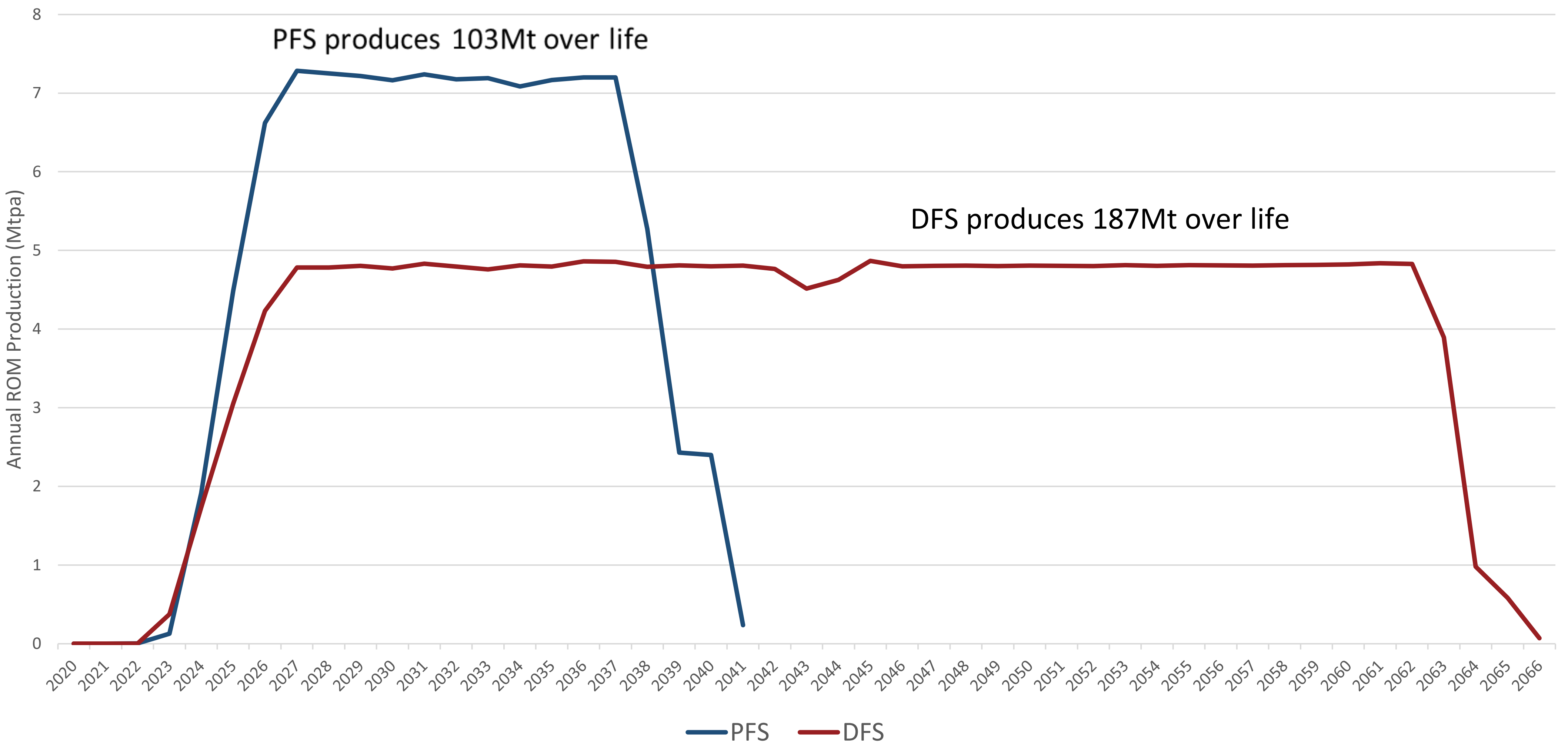
27 September 2019

Resource, Reserve and other non-financial metrics

Metric	PFS	DFS	Change	Comments
Published Date	Sep-16	Sep-19		
Resource M&I (Mt)	218	242	11%	Increase due to in-fill drilling
4E Grade (g/t)	3.55	3.38	-5%	
4E Metal (Moz)	24.9	26.3	6%	
Reserve (Mt)	103	187	82%	Large increase due to introduction of pastefill
4E Grade (g/t)	3.73	3.24	-13%	Reduction due to lower resource grade and higher dilution
4E metal (Moz)	12.3	19.5	59%	Large increase due to introduction of pastefill
Resource-to-reserve conversion	47%	77%	64%	Large increase due to introduction of pastefill
Annual production (Mt)	7.2	4.8	-33%	
Annual production 4E Metals (koz)	744	420	-44%	Lower grade and lower production
Annual production Ni & Cu (Mlbs)	23.0	16.7	-27%	
LOM (years)	19	45	137%	Large increase due to increased reserve and reduced throughput
4E Recovery	82.1%	78.9%	-4%	Lower due to lower grade
Water required (Ml/day)	11	7	-36%	Smaller mine
Peak Power demand (MVA)	160	90	-44%	Smaller mine
Headcount	3361	1170	-65%	Smaller mine
Productivity (oz/employee/month)	18	30	62%	Higher productivities due to larger stopes

Ore Production Comparison

***Normalised start-date**



Costing

Using 3-year trailing Price Analysis

Metric	PFS	DFS	Change	Comments
Project Capex (US$m)	1060	874	-18%	High infrastructure spend hence reduction not proportional to throughput reduction
Peak Funding (US$m)	914	667	-27%	Lower capex spend
Total Capex (US$m)	1825	2418	32%	Higher sustaining capex in DFS due to longer mine life and more tonnes mined
Total Capex/oz LOM (USD/oz)	148	124	-16%	Big increase in reserve
Opex (Site-cost US$/t)	38.31	40.80	6%	Larger stopes, higher productivities DFS includes pastefill
Opex (Cash-cost US$/4E oz)	481	554	15%	DFS lower grade and added pastefill

Note: DFS is 33% smaller from an annual tonnes milled perspective than the PFS

Costing

Using 3-year trailing Price Analysis*

Metric	PFS	DFS	Change	Comments
Project Capex (US$m)	1 146	874	-24%	High infrastructure spend hence reduction not proportional to throughput reduction
Peak Funding (US$m)	989	667	-33%	Lower capex spend
Total Capex (US$m)	1 974	2418	22%	Higher sustaining capex in DFS due to longer mine life and more tonnes mined
Total Capex/oz LOM (USD/oz)	160	124	-23%	Big increase in reserve
Opex (Site-cost US$/t)	41.44	40.80	-2%	Larger stopes, higher productivities DFS includes pastefill
Opex (Cash-cost US$/4E oz)	520	554	6%	DFS lower grade and added pastefill

*PFS costs have been normalised to 2019 base by 4% per annum escalation (combination of SA & US inflation)

Note: DFS is 33% smaller from an annual tonnes milled perspective than the PFS

Capex Anatomy

***PFS numbers have been escalated by 4% per annum to account for inflation**

WBS Area	PFS	DFS	Change
Underground Mining	439	406	-7%
Concentrator	205	172	-16%
Shared Services and Infrastructure	77	45	-41%
Regional Infrastructure	185	82	-56%
Site Support Services	50	16	-69%
Project Delivery Management	101	44	-57%
Other Capitalised Costs	18	22	24%
Provisions	72	87	20%
Total	1 147	874	-24%

Note: DFS is 33% smaller from an annual tonnes milled perspective than the PFS
Not all categories contain the same items

Financials

Using 3-year trailing Price Analysis (PFS as published, i.e. un-escalated)

Metric	PFS	DFS	Change
IRR	13.50%	13.30%	-1%
NPV Undiscounted (US$m)	1669	3489	109%
NPV 8% Discount (US$m)	320	333	4%
Pay back (years)	10	11	10%
After tax annual free-cash flow	240	130	-46%
Basket Price 4E (US$/oz)	899	1045	16%
Margin	46%	47%	1%

Financials

Using Spot Price Analysis (*PFS normalised on same basis)

Metric	PFS	DFS	Change
IRR	19.3%	20.7%	7%
NPV Undiscounted (US$m)	3709	6613	78%
NPV 8% Discount (US$m)	974	982	1%
Pay back (years)	9	8	-11%
Peak Funding (US$m)	1079	617	-43%

*In this analysis, the PFS has been rebased by escalating all costs to the July 2019 DFS base, and starting construction at the same time as the DFS plan. The same Spot Price deck and exchange rate is used for both PFS & DFS