OreWin
Independent Mining Consultants

Ivanhoe Mines Ltd.

Platreef 2022 Feasibility Study

March 2022

Job No. 20021

IMPORTANT NOTICE

This notice is an integral component of the Platreef 2022 Feasibility Study and should be read in its entirety. The Platreef 2022 Feasibility Study has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Platreef 2022 Feasibility Study has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by OreWin Pty Ltd (OreWin), Mine Technical Services (MTS), SRK Consulting Inc (SRK), DRA Projects (Pty) Ltd (DRA)and Golder Associates Africa (Pty) Ltd (Golder) as the Report Contributors. The Platreef 2022 Feasibility Study is based on information and data supplied to the Report Contributors by Ivanhoe and other parties. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of the Report Contributors, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. Each portion of the report is intended for use by Ivanhoe subject to the terms and conditions of its contract with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the report, by any third party, is at that party's sole risk. The Platreef 2022 Feasibility Study is intended to be used by Ivanhoe, subject to the terms and conditions of its respective contracts with the Report Contributors. Recognising that Ivanhoe has legal and regulatory obligations, the Report Contributors have consented to the filing of the Platreef 2022 Feasibility Study with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR).

The results of the Platreef 2022 Feasibility Study represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Platreef 2022 Feasibility Study. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant Section.

The conclusions and estimates stated in the Platreef 2022 Feasibility Study rely on assumptions stated in the Platreef 2022 Feasibility Study. The results of further work may indicate that the conclusions, estimates, and assumptions in the Platreef 2022 Feasibility Study need to be revised or reviewed.

The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Platreef 2022 Feasibility Study. Where the Report Contributors have made those estimates and approximations, they are subject to qualifications and assumptions, and it should also be noted that all estimates and approximations contained in the Platreef 2022 Feasibility Study will be prone to fluctuations with time and changing industry circumstances.

The Platreef 2022 Feasibility Study should be construed in light of the methods, procedures, and techniques used to prepare the Platreef 2022 Feasibility Study. Sections or parts of the Platreef 2022 Feasibility Study should not be read in isolation of or removed from their original context.

Title Page

Project Name:	Platreef Project
Title:	Platreef 2022 Feasibility Study
Location:	Limpopo Province, Republic of South Africa
Effective Date of Technical Report:	28 February 2022
Mineral Resource Amenable to Selective Underground Mining Methods:	28 January 2022
Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	28 January 2022
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	28 January 2022
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Validation of resource models as current using current economic inputs:	20 November 2020
Effective Date of Mineral Resources:	28 January 2022
Effective Date of Mineral Reserves:	26 January 2022

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Signature Page

Project Name:	Platreef Project
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Effective Date of Technical Report: 28 February 2022

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1 SUMMARY

1.1 Introduction

The Platreef 2022 Feasibility Study (Platreef 2022 FS) is an Independent NI 43-101 Technical Report prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe Mines Ltd. (Ivanhoe) on the Platreef nickel–copper–gold–platinum group elements (PGE) project (the Platreef Project), located near Mokopane in the Limpopo Province of the Republic of South Africa (see Figure 1.1).

Ivanhoe has undertaken further studies following the Platreef 2017 Feasibility Study (Platreef 2017 FS) and the Platreef Integrated Development Plan 2020 (Platreef IDP20) on the Platreef Project that have formed the basis of the Platreef 2022 FS, which summarises the current Ivanhoe development strategy for the Platreef Project.

The Platreef 2022 FS is a Phased Development Plan based on continued development and earlier production from Shaft 1 which starts with 700 ktpa production (2024–2027) and then two 2.2 Mtpa concentrator streams will be added in 2028 and 2030, increasing the production rate to 5.2 Mtpa. The Platreef 2022 FS describes a change in production rate for the project that will require separate capital costs and infrastructure. The Platreef 2022 FS includes two main Phases. Phase 1 is defined after Shaft 1 changeover until Shaft 2 reaches to the full hoisting capacity of 6.19 Mtpa. In this Phase, Shaft 1 will be utilised with the 825 ktpa rock-hoisting capacity (including up to 125 ktpa allocated for development rock). Phase 2 commences just after equipping Shaft 2 when production begins to ramp up to 5.2 Mtpa.

The summary results of the Platreef 2022 FS are in Table 1.1.

Table 1.1 Platreef 2022 FS Summary Results

Item	Unit	Value
Mined and Processed	Mt	125
Platinum	g/t	1.94
Palladium	g/t	1.99
Gold	g/t	0.30
Rhodium	g/t	0.13
3PE+Au	g/t	4.37
Copper	% Cu	0.16
Nickel	% Ni	0.34
Concentrate Produced	kt	5,545
Platinum	g/t	38.2
Palladium	g/t	39.0
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE+Au	g/t	85.0
Copper	% Cu	3.3
Nickel	% Ni	5.4
Recovered Metal		
Platinum	koz	6,813
Palladium	koz	6,954
Gold	koz	948
Rhodium	koz	433
3PE+Au	koz	15,149
Copper	Mlb	399
Nickel	Mlb	665
Key Financial Results		
Life of Mine	Years	29
Initial Capital (Pre-Production)	$M	488
Expansion Capital	$M	1480
Sustaining Capital	$M	934
Mine-Site Cash Cost	$/oz Rec. 3PE+Au	429
Total Cash Costs After Credits	$/oz Rec. 3PE+Au	452
Total Cash Costs After Credits & Sustaining Capital	$/oz Rec. 3PE+Au	514
Site Operating Costs	$/t Milled	52
After-Tax NPV8%	$M	1,690
After-Tax IRR	%	18.48
Project Payback Period	Years	7.93

1. Totals may not add due to rounding.

2. Initial Capital including $50M in Shaft 2 and $32M in contingencies.
3. 3PE+Au = platinum, palladium, rhodium and gold.
4. Economic analysis metal price assumptions: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

Figure 1.1 Platreef Project Location

Ivanhoe, 2014

Ivanhoe is a mineral exploration and development company with a portfolio of properties located in Africa. The Ivanhoe strategy is to build a global, commodity-diversified mining and exploration company. Ivanhoe has focused on exploration within the Central African Copperbelt and the Bushveld Complex. Ivanhoe currently has three key assets:

- the Kamoa-Kakula Project
- the Platreef Project, and
- the Kipushi Project.

The South African Mining Right (MR) LP30/5/2/2/1/10067MR forms the current Platreef Project area, with an additional area contemplated to be added through a prospecting right application in respect of the farm Rietfontein 2 KS.

Ivanhoe holds a 64% interest in Mining Right LP30/5/2/2/1/10067MR, while a Japanese consortium (the Japanese Consortium), comprising Itochu Corporation (Itochu); Japan Oil, Gas and Metals National Corporation (JOGMEC); and Japan Gas Corporation (JGC), holds a 10% interest, and local communities, local entrepreneurs, and staff hold the remaining 26% as a result of the Broad-Based Black Economic Empowerment (B-BBEE) transaction, implemented on 26 June 2014. The Japanese Consortium's interest in the Platreef Project was acquired in two tranches for a total investment of $290M.

Holdings in the Platreef Project are through South African subsidiary Ivanplats (Pty) Ltd (Ivanplats), formerly named Platreef Resources (Pty) Limited.

1.2 Mineral Tenure and Surface Rights

Ivanplats obtained a mining right in terms of Section 22 of the Mineral and Petroleum Resources Development Act 28 of 2002 (MPRD Act) from the Department of Mineral Resources (DMR) on 30 May 2014, which right became effective through notarial execution on 4 November 2014.

Ivanplats also received an Environmental Authorisation (EA) in terms of the National Environmental Management Act 107 of 1998 (NEM Act) from the Limpopo Department of Economic Development, Environment and Tourism (LEDET) on 27 June 2014.

The Platreef Project is located at about 24º05'S and 28º59'E. The Project is located in the Limpopo Province of the Republic of South Africa on three farms: Turfspruit 241 KR (3,561 ha), Macalacaskop 243 KR (4,281 ha) and Rietfontein 2 KS (2,878 ha) Figure 1.2.

Mining Right (MR) LP30/5/2/2/1/10067MR boundaries correspond to the perimeter boundaries of the Macalacaskop (243 KR) and Turfspruit (241 KR) farms. Collectively, the MR and the area of a new prospecting right application in respect of Rietfontein 2 KS form the Platreef Project area. Rietfontein 2 KS has a contiguous border with Turfspruit 241 KR, sharing a common boundary along the south-western border of Rietfontein and the north-eastern border of Turfspruit.

1.2.1 Macalacaskop 243 KR and Turfspruit 241 KR

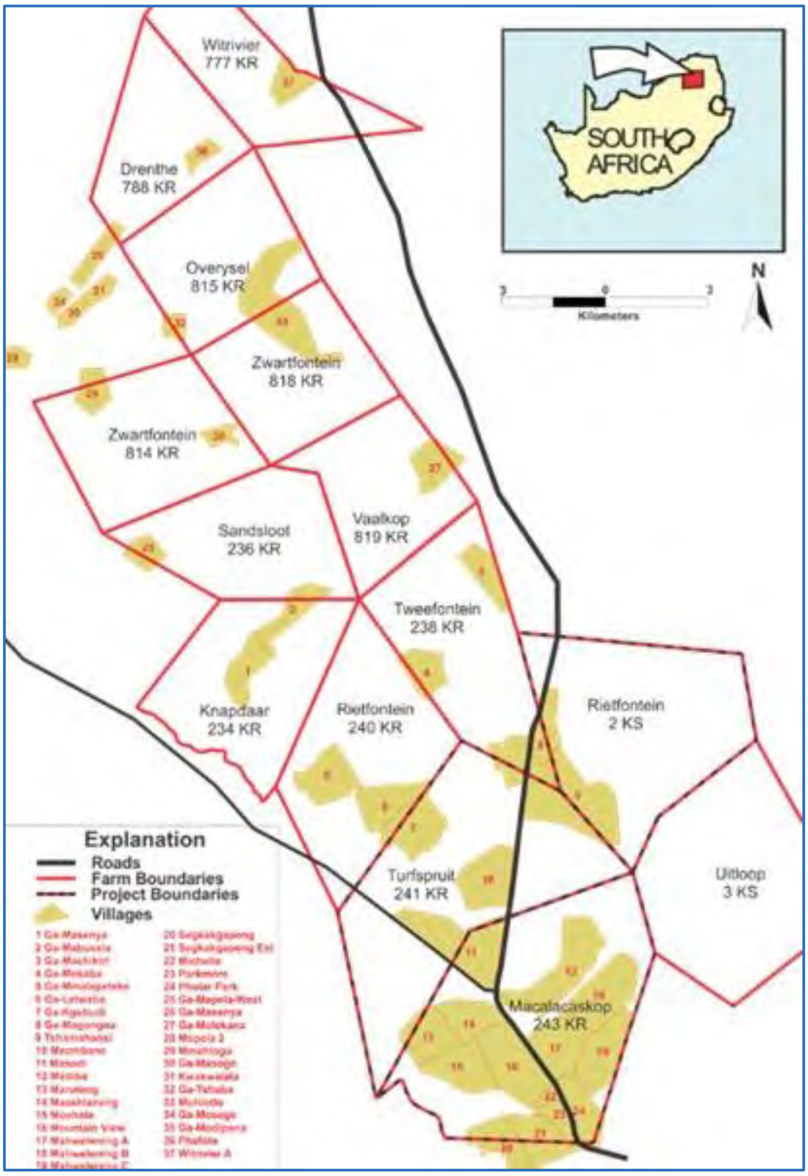
Ivanplats Proprietary Limited was the holder of a Prospecting Right, which was notarially executed on 2 February 2006 and registered in the South African Mineral and Petroleum Titles Registration Office (MPTRO) on 9 February 2006 under registration number 55/2006 PR, entitling Ivanplats to prospect for base minerals and precious metals in, on and under the farm Turfspruit 241 KR and the farm Macalacaskop 243 KR, for a period of five years commencing on 2 February 2006 and ending on 1 February 2011 ("the PR"). The PR was subsequently renewed on 1 June 2011, for a period of three years, commencing on 1 June 2011 and ending on 31 May 2014.

Ivanplats then successfully applied for a Mining Right in respect of the land to which the Prospecting Right related. The Mining Right was granted by means of a letter dated 30 May 2014 informing Ivanplats that its application for a Mining Right in respect of PGEs, gold, silver, nickel, copper, iron, vanadium, cobalt and chrome in respect of the farms Macalacaskop 243 KR and Turfspruit 241 KR (excluding areas comprising graveyards, built-up areas and protected areas) had been granted.

The Mining Right was notarially executed on 4 November 2014 with DMR reference number LP30/5/2/2/1/10067MR ("the MR"). Under the MPRD Act, notarial execution of the mining right is required before it may be exercised.

By virtue of the MR, Ivanplats is the sole and exclusive holder of the mining title in and to PGEs, gold, silver, nickel, copper, iron, vanadium, cobalt and chrome in respect of the farms Macalacaskop 243 KR and Turfspruit 241 KR (excluding areas comprising graveyards, built-up areas and protected areas). The MR commenced on 4 November 2014 and, unless cancelled or suspended in terms of clause 13 of the MR and/or Section 47 of the MPRD Act, will continue in force for a period of 30-years ending on 3 November 2044, renewable for further periods, each of which may not exceed 30 years at time.

Figure 1.2 Project Location and Farm Boundaries



Ivanhoe, 2014

1.2.2 Rietfontein

Ivanplats previously participated in a joint venture with Atlatsa Resources Corporation (formerly Anooraq Resources Corporation) through its South African subsidiary, Plateau Resources Limited, which held a prospecting right (No.PT76/2007PR) to prospect for base and precious metals on the farm Rietfontein 2 ("the Rietfontein PR") KS, over a total area of 2,878 ha.

The Prospecting Right had in the meantime expired, however, Ivanplats submitted its own application for a prospecting right in respect of Rietfontein on 2 September 2019. Simultaneously, Ivanplats applied for an environmental authorisation in respect of the listed activities which will be undertaken in the exercise of the prospecting right.

The application for an environmental authorisation was rejected by the DMR on 3 November 2020, on the basis of alleged failure by the appointed independent environmental practitioner to conduct sufficient public participation as part of the assessment of environmental impacts. It appears that the DMR's complaint is based primarily on the fact that the independent environmental practitioner did not conduct mass meetings with the affected community. However, this is not a requirement in terms of the relevant legislation and was prohibited during the period in question due to current restrictions on large public gatherings, aimed at preventing the spread of COVID-19.

Ivanplats submitted an appeal against the rejection of the application for an environmental authorisation through its attorneys, Webber Wentzel, on 30 November 2020. However, following further discussions with the DMR, Ivanplats agreed to withdraw its prospecting right application and submit a new application for a prospecting right and an environmental authorisation. The new application was submitted on 16 February 2022.

1.2.3 Surface Rights

Ivanplats entered into Surface Use and Cooperation Agreements (SUCAs) with the leadership of the following communities, on whose land Ivanplats' mining activities are taking place:

- Ga-Magongoa,
- Ga-Kgobudi,
- Ga-Madiba, and
- Tshamahansi (comprised of Baloyi, Matjeke and Hlongwane).

The SUCAs were initially concluded when Ivanplats was conducting prospecting activities on the relevant land. The chief purpose of the SUCAs at the time was to provide for adequate compensation for persons whose use of their land was adversely impacted by Ivanplats' prospecting activities and provided for seasonal payments to each beneficial landowner.

With the transition from prospecting activities to mining activities, Ivanplats recognises the need to agree with the affected communities on a long-term solution, in terms of which beneficial landowners are compensated for the loss of the agricultural use of their land, and are given alternative means of generating an income, as part of a broader livelihood restoration programme.

The SUCAs presently represent an interim measure to ensure that subsistence farmers continue to receive compensation for the use of their land, pending the conclusion of long-term surface leases with the relevant communities.

1.3 History and Exploration

1.3.1 Exploration

Early exploration on the Platreef mineralisation dates back to the 1960s. Subsequently, Rustenberg Platinum Holdings Limited, a wholly owned subsidiary of Anglo-American Platinum Corporation, began exploration on the Platreef Project in the 1970s. No data from either of these programmes were available for preparation of the Platreef 2022 FS.

Ivanhoe acquired a PR for both Turfspruit 241 KR and Macalacaskop 243 KR farms in February 1998. Ivanplats previously participated in a joint venture with Atlatsa Resources Corporation (formerly Anooraq Resources Corporation) through its South African subsidiary, Plateau Resources Limited, which held a prospecting right on the farm Rietfontein 2 KS. The prospecting right expired and Ivanplats submitted its own application for a prospecting right in respect of Rietfontein.

The initial exploration focus was on delineation of mineralisation that could support open pit mining. From 2003–2007, Ivanhoe undertook studies involving concentrator and smelter options, metallurgical testwork, and conceptual mining studies that considered open pit scenarios.

In 2007, Ivanhoe commenced a deep-drilling programme to investigate the continuity and grade in an area targeted as having underground mining potential. This resulted in multiple mineral resource estimates assuming underground mining methods between September 2010–May 2013.

Work completed to date includes geological mapping, airborne and ground geophysical surveys, percussion drilling over the Platreef sub-crop, diamond core drilling, petrography, density determinations, metallurgical testwork, geotechnical and hydrological investigations, seismic survey, Environmental and Social Impact Assessments, mineralogical studies, Mineral Resource and Mineral Reserve estimation and subsequent updates, a preliminary economic assessment, and a prefeasibility study.

1.4 Geology and Mineralisation

The Platreef mineralisation comprises a variably layered, composite norite–pyroxenite–harzburgite intrusion that lies near the base of the Northern Limb of the Bushveld Complex, in contact with metasedimentary and granitic floor rocks. The variability of lithology and thickness along strike is attributed to underlying structures and assimilation of local country rocks.

Within the Platreef Project area, five major cyclic units have been recognised, which correlate well with the Upper Critical Zone (UCZ) rock sequence described for the main Bushveld Complex. The Turfspruit Cyclic Unit (TCU) is the main mineralised cyclic unit; this unit is analogous to the Merensky Cyclic Unit (MCU) that contains the Merensky anorthosite and pyroxenite and hosts the Bushveld's principal mineralised reefs. The TCU is laterally continuous across large parts of the Platreef Project area. Mineralisation in the TCU shows generally good continuity and is mostly confined to pegmatoidal orthopyroxenite and harzburgite.

Other cyclic units that have been identified adjacent to the TCU are the Norite Cycles (NC1 and NC2), Pseudo Reef, and the Upper Group 2 (UG2). Contamination of the UCZ units by assimilation of Transvaal Supergroup metasedimentary rocks can occur within any of the stratigraphic horizons; however, in the area being considered for underground mining, contamination is predominantly confined to the units below the TCU.

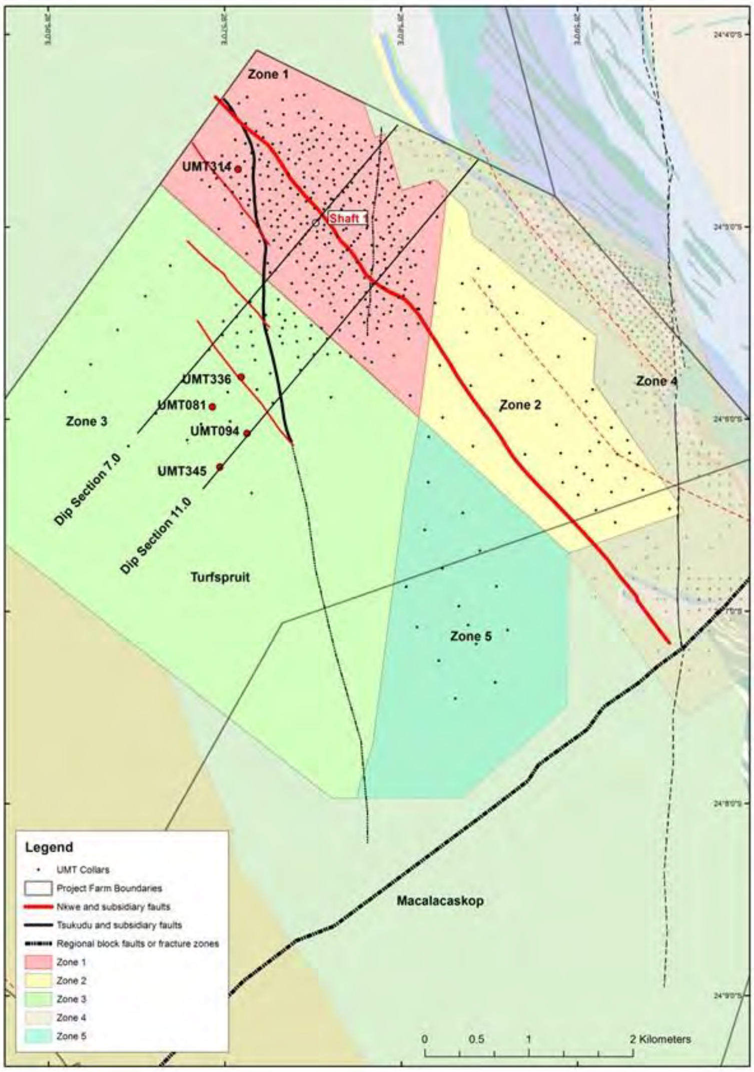
Within the TCU, high-grade PGE–Ni–Cu mineralisation is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, bound by chromitite stringers and containing coarse-grained sulfides; this is known as T2, with the mineralised portion referred to as T2MZ. The T2 pegmatoid is subdivided into an upper pyroxenitic unit (T2Upper or T2U) and a lower olivine-bearing pyroxenitic or harzburgitic unit (T2Lower or T2L). Overlying this pegmatoidal package is a non-pegmatoidal, feldspathic pyroxenite unit of variable thickness, termed T1.

A second mineralised zone of disseminated, medium- to coarse-grained sulfides (T1MZ) occurs near the top of the T1 feldspathic pyroxenite.

A geographical demarcation of the Platreef Project area into five zones (Zone 1–5 (Madiba), refer to Figure 1.3, has been developed based on exploration criteria. Three distinct geological features are recognised within these zones and include the following:

- A double reef package informally termed the Bikkuri Reef, wherein an upper pyroxenite-dominated mineralised sequence (the Bikkuri Reef) is separated from a thicker, mixed-lithology sequence by Main Zone (MZ) and metasedimentary lithologies.
- Presence of a flat-lying portion of the TCU (Flatreef) that is related to structural controls.
- Local mineralisation in the footwall (FW) to the TCU.

Figure 1.3 Project Exploration Zones Plan



Ivanhoe, 2016

The structural model includes three key deformation features:

- Folding – Pre-Bushveld low amplitude, upright open folds defined by remnant metasedimentary interlayers and xenoliths which are parallel to mineralised zones.
- Ductile shear zones – 30 cm–3 m wide, north-west trending, steeply dipping (60–70°), oblique reverse sense of movement, variable dip direction, possible antithetic riedel shear zones.
- Brittle fault zones – 5–30 m wide, north trending, moderate to steeply dipping (50–70°), extensional (east block down) normal faults.

Six faults are used to define seven fault blocks for the structural model.

The Tshukudu Fault Zone is a brittle structure that transgresses the central portion of Zone 1. It represents a significant geotechnical hazard and comprises a wide zone of imbricate fracturing in its hanging wall and intense brecciation within the fault zone. Major fall-of-ground hazards can be expected where this brittle fault intersects ductile shear zones. Significant vertical displacement is associated with this fault zone in the order of 60 m (Brits, 2015). The fault zone is generally steeply inclined and has an easterly dip direction and oblique normal sense of movement. The fault is defined by 129 drill core intersections and has a minimum thickness of 0.6 m and a maximum thickness of 26 m for an average thickness of 7.6 m.

The major ductile fault structures currently recognised include the Nkwe, Tau, Mabitso, Fisi, Tlou and Lengau.

Two-fold orientations have been observed, and these concur with the previous Northern Limb studies. The first and major fold orientation (F1) is north–north-west–south–south-east. These folds have subsequently been gently refolded with the minor fold axis (F2) trending east–north-east–west–south-west. The F1 folds are responsible for the apparent flattening of the Platreef basinward, the Macalacaskop syncline, the so called "T1-trough" and the overall 50° dip to the south-west along the open-pit fold limb. The minor folds are responsible for domes and basins within the larger folds such as the Bikkuri dome.

Broadly, Zone 1 or the 'Flatreef' can be envisioned as a monocline or parasitic fold on a major north–north-west trending, south-west dipping fold limb. Syn-magmatic sagging or uplift due to crustal loading and volume increase may have locally amplified the synclines and anticlines respectively.

Pyrrhotite, pentlandite and chalcopyrite occur as interstitial sulfides in the TCU lithologies. PGEs are mainly present as PGE–sulfides and PGE–Bi–Te and PGE–As alloys, that are fine-grained (<10 µm) and may occur within base metal sulfides, on their rims, or encapsulated in silicates).

1.5 Drilling

Drilling on the Platreef Project has been undertaken in two major phases:

- The first from 2001–2003 is termed the open-pit programme (designated AMK at Macalacaskop 243 KR and ATS at Turfspruit 241 KR/Rietfontein 2 KS). The open-pit programme drillholes are located in Zone 4 (see Figure 1.3).
- The second phase commenced in 2007, and the most recent campaign ended 11 February 2015. This second drill phase is termed the underground programme, is designated UMT (including Bikkuri), and nearly all drilling is on Turfspruit 241 KR. These drillholes are situated in Zones 1–3 and Zone 5. There were two drillholes (PUM001 and PUT001) drilled in 2012 which are located in Zone 4. These drillholes are grouped with the open-pit drillholes.

The database (closed 24 July 2015) includes 578 drillholes (196,213 m) from Phase 1 (including all redrills and deflections). The Phase 1 drilling was completed in support of historical open-pit resources (See Section 6).

The database also includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totalling 501,638 m completed by 11 February 2015. There has been no additional drilling since that date for resource estimation purposes; however, assay data is now available for three drill holes used for the geology model (GT008, GT017 and TMT015. GT008 was drilled down the location of Shaft 1. GT017 was drilled in the position of Shaft 2. TMT015 was drilled between. Additional sampling was accomplished on twenty-four drill holes to test mineralisation in the upper portions of the NC1 and lower portions of the FAZ and PNZ.

Depths for deflections are calculated based on point of deflection, and do not include the mother or pilot hole portion. This includes 33 drillholes and deflections (9,181 m) completed for geotechnical purposes and 62 drillholes and deflections (23,001 m) completed for metallurgical sampling purposes.

Standardised geological logging conventions were used to capture information from drill core. Geotechnical logging has been undertaken on selected drill cores.

In the majority of instances, core recovery is 100%. The recoveries substantially decrease within faulted and sheared zones.

1.5.1 Collar Surveys

Collar surveys were conducted by a licensed land surveyor on all completed drillholes.

1.5.2 Downhole Surveys

The majority of drillholes are down-hole surveyed. Downhole deviation surveys for the UMT drilling were completed by independent downhole survey technicians using gyroscopic (gyro) and/or electronic multi-shot (EMS) instruments. Surveys are recorded downhole at 3–5 m intervals. In Zones 1–3 and Zone 5, there are 21 drillholes without surveys. Of these, 15 drillholes were drilled for geotech purposes and are less than 30 m in depth. Five drillholes were deflections with depths ranging from 28–780 m. Additionally, UMT377 is unsurveyed with a depth of 1,409 m.

Where both an EMS and a gyro survey were completed, the gyro survey was assumed to be more accurate and therefore in most cases was used in construction of the geological model. There are 181 drillholes where the EMS has been selected, due to erroneous or uncompleted gyro surveys. A memorandum from site (Ivanplats, 2015) discussing a review of the downhole surveys states that EMS downhole surveys were selected over gyro survey results for 70 drillholes.

1.6 Sample Preparation, Analyses and Security

Over the duration of Ivanhoe's work programmes, sample preparation and analyses were performed by accredited independent laboratories. Sample preparation is accomplished by Set Point laboratories in Mokopane. Sample analyses have been accomplished by Set Point Laboratories (Set Point) in Johannesburg, Lakefield Laboratory (Lakefield' now part of the SGS Group) in Johannesburg, Ultra Trace (Ultra Trace) Laboratory in Perth, Genalysis Laboratories, Perth and Johannesburg (Genalysis), SGS Metallurgical Services (SGS) in South Africa, Acme in Vancouver, and ALS Chemex in Vancouver. Bureau Veritas Minerals Pty Ltd (Bureau Veritas) assumed control of Ultra Trace during June 2007 and is responsible for assay results after that date.

Sample preparation and analytical procedures for samples that support Mineral Resource estimation have followed similar protocols since 2001. The preparation and analytical procedures are in line with industry-standard methods for Pt, Pd, Au, Cu, and Ni deposits. Drill programmes included insertion of blank, duplicate, standard reference material (SRM), and certified reference material (CRM) samples. The quality assurance and quality control (QA/QC) programme results do not indicate any problems with the analytical protocols that would preclude use of the data in Mineral Resource estimation.

Sample security has been demonstrated by the fact that the samples were always attended or locked in the on-site core facility in Mokopane.

1.7 Data Verification

The Qualified Person reviewed the sample chain of custody, quality assurance and quality control QA/QC procedures, and qualifications of analytical laboratories. In addition, the Qualified Person audited the assay database, core logging, and geological interpretations. Based on these reviews, conducted when the Qualified Person was still employed at Wood Plc (Wood), the Qualified Person considers that the data are acceptable to support Mineral Resource estimation.

1.8 Mineral Resource Estimates

In 2015–2016, three mutually exclusive Mineral Resource models were constructed that reflect the foci of planned development.

1.8.1 UMT-TCU Selectively Mineable Model

Mineral Resources amenable to selective mining methods occur below the 650 m elevation (approximately 500 m depth) and near the stratigraphic top of the Platreef. Mechanised Drift-and-Fill and longhole stoping are being contemplated. Components of the TCU and adjacent material were modelled deterministically. Two main mineralised zones were modelled using three internal grade shells with nominal 3PE+Au cut-off grades of 1 g/t, 2 g/t, and 3 g/t. The term 3PE includes platinum + palladium + rhodium. Significant rhodium analyses were added to the database during 2014–2015 and permit the grade shells to be constructed using 3PE+Au cut-offs. An updated structural model has been completed based on significant re-logging of drill core in the Main Zone (MZ), TCU and FW units and geophysical investigations including a 3D seismic survey. The lithological units and grade shells were hung on an artificial horizontal plane, and interpolation of Pt, Pd, Au, Rh, Cu, Ni and S was performed using an inverse distance weighting to the third power (ID3) interpolation method. Ordinary kriging (OK) and nearest neighbour interpolations were completed for validation. This Mineral Resource model and validations were completed in September 2015.

1.8.2 Additional Mutually Exclusive Mineral Resource Models

Outside the selectively-mineable model, two other mutually-exclusive Mineral Resource models have been constructed since 2013. These are:

- Bikkuri area Mineral Resources are considered to be potentially amenable to underground selective mining methods. This consists of material within and adjacent to 3PE+Au grade shells in the Bikkuri Reef. This Mineral Resource estimate has been estimated using revised geological interpretations and incorporation of additional drilling in Zone 1 and Zone 2 that intercepted the Bikkuri Reef. The Mineral Resources amenable to selective underground mining methods in the Bikkuri Reef are supported by the UMT–Bikkuri model, completed in September 2015.
- UMT-FW area Mineral Resources are considered to be potentially amenable to underground mining using selective and locally possibly less selective mining methods. This consists of material that is FW to the TCU that shows a degree of grade continuity. This Mineral Resource has been estimated using revised geological interpretations for the footwall strata occurring immediately beneath the TCU in Zone 1. The Mineral Resources amenable to underground mining methods in the footwall to the TCU are supported by the UMT–FW model, completed in February 2016.

1.9 Mineral Resource Statements

Table 1.2 summarises the combined Platreef Mineral Resources that are amenable to underground selective mining methods (UMT-TCU, UMT-BIK, UMT-FW). Sensitivity to cut-off is shown with the base case highlighted. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources are reported on a 100% ownership basis. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME, Mine Technical Services (MTS). Mineral Resources have been estimated using core drill data, have been performed to industry best practices (CIM, 2019), and conform to the requirements of the CIM Definition Standards, 2014.

The estimates for individual mutually exclusive Mineral Resource models are presented in Section 14.

Table 1.2 Platreef Mineral Resources; All Mineralised Zones (Base Case Highlighted)

Indicated Mineral Resources Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	204	2.11	2.11	0.34	0.14	4.70	0.18	0.35
2 g/t	346	1.68	1.70	0.28	0.11	3.77	0.16	0.32
1 g/t	716	1.11	1.16	0.19	0.08	2.55	0.13	0.26
Indicated Mineral Resources Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
3 g/t	–	13.9	13.9	2.2	0.9	30.9	800	1,597
2 g/t	–	18.7	18.9	3.1	1.2	41.9	1,226	2,438
1 g/t	–	25.6	26.8	4.5	1.8	58.8	2,076	4,108
Inferred Mineral Resources Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	225	1.91	1.93	0.32	0.13	4.29	0.17	0.35
2 g/t	506	1.42	1.46	0.26	0.10	3.24	0.16	0.31
1 g/t	1,431	0.88	0.94	0.17	0.07	2.05	0.13	0.25
Inferred Mineral Resources Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
3 g/t	–	13.8	14.0	2.3	1.0	31.0	865	1,736
2 g/t	–	23.2	23.8	4.3	1.6	52.8	1,775	3,440
1 g/t	–	40.4	43.0	7.8	3.1	94.3	4,129	7,759

1. Mineral Resources were estimated as at 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulations were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resources is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Ivanhoe holds a 64% interest in the project. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 m x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80–90% for Pt, Pd and Rh; 70–90% for Au, 60–90% for Cu, and 65–75% for Ni.
5. 3PE+Au = (Pt + Pd + Rh) + Au.
6. Totals may not sum due to rounding.

Factors that could affect the estimates include Ivanhoe's ability to conclude surface access agreements to allow continued exploration and sampling programmes, permitting, environmental, legal and socio-economic assumptions including availability of power and water, and assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction.

The Qualified Person reviewed provisional results of a seismic survey conducted during 2014 and completed some twinned drillhole data analyses. The Qualified Person notes that the current practice of using grade shells in the area drilled in detail may under-estimate the variability of the grades within and near the T1MZ and the T2MZ. Stope boundaries that are laid out along the 2 g/t 3PE+Au grade shell surface will likely not, in practice, be able to follow the exact actual surface. The consequence would be that the effects of contact dilution and ore loss could be more than is currently projected.

1.10 Targets for Further Exploration

Beyond the current Mineral Resources, mineralisation is open to expansion to the south and west. Targets for further exploration (exploration targets) have been identified. Wood cautions that the potential quantity and grade of these exploration targets is conceptual in nature. There has been insufficient exploration and/or study to define these exploration targets as a Mineral Resource. It is uncertain if additional exploration will result in these exploration targets being delineated as a Mineral Resource.

The Bushveld Igneous Complex (BIC) PGE-Ni-Cu deposits have characteristics of lateral continuity over several thousands of metres. Based on this, four exploration targets have been identified. Target areas are defined based on the 2016 UMT-TCU Mineral Resource Model and represent currently undrilled extension areas from the model.

- Target 1 could contain 100–165 Mt grading 3.1–5.2 g/t 3PE+Au (1.3–2.2 g/t Pt, 1.5–2.5 g/t Pd, 0.18–0.30 g/t Au, 0.12–0.21 g/t Rh), 0.10–0.17% Cu, and 0.22–0.36% Ni over an area of 4.1 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 2 could contain 50–90 Mt grading 2.9–4.9 g/t 3PE+Au (1.3–2.1 g/t Pt, 1.4–2.3 g/t Pd, 0.19–0.31 g/t Au, 0.11–0.18 g/t Rh), 0.11–0.19% Cu, and 0.23–0.39% Ni over an area of 3.3 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 3 could contain 20–30 Mt grading 2.6–4.4 g/t 3PE+Au (1.2–1.9 g/t Pt, 1.2–2.0 g/t Pd, 0.19–0.32 g/t Au, 0.10–0.16 g/t Rh), 0.12–0.20% Cu, and 0.23–0.39% Ni over an area of 0.5 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 4 could contain 10–20 Mt grading 2.1–3.4 g/t 3PE+Au (1.0–1.6 g/t Pt, 0.9–1.4 g/t Pd, 0.13–0.22 g/t Au, 0.10–0.17 g/t Rh), 0.09–0.15% Cu, and 0.19–0.32% Ni over an area of 1.5 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.

Beyond these exploration target areas is approximately 48 km^2 of unexplored ground on the property under which prospective stratigraphy is projected to lie. There is insufficient information to reasonably estimate a range of tonnages and grades for this ground.

There is potential for the extent of known mineralisation to significantly increase with further step-out drilling to the south-west.

1.11 Mineral Reserves

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in Section 14 of the Platreef 2022 FS. Only Indicated Mineral Resources have been used for determination of the Probable Mineral Reserve.

The Mineral Resource block model also includes the net smelter return (NSR) variable. NSR calculation formulas and metal prices used in the block model were provided by Ivanplats. NSR is the dollar value of the metals recovered from a tonne of rock minus the cost for transportation of concentrate to the smelter, royalties, smelting and refining charges, and other smelter deductions. These parameters were used to calculate the NSR in units of $/t for each cell in the block model.

Mineral Reserves were calculated from the block model using the combination of stope optimiser and generated grade based on the economic NSR cut-off values. Two stoping methods (longhole, and Drift-and-Fill) were selected for the Platreef Project as they satisfy the following design criteria:

- Maintain maximum productivities by incorporating bulk-mining methods and operational flexibility, which will result in lower operating costs.
- Maintain high overall recovery rates.
- Minimise overall dilution.
- Prevent surface subsidence from underground mining.

Platreef 2022 FS cost estimates have been done to a feasibility study level of accuracy. For further detail on cost estimates, refer to Section 21.

The cost-per-tonne differential between the Platreef 2017 FS calculated marginal NSR cut-off grades ($47.71–$58.53) and the production schedule NSR cut-offs ($155 and $80), provides a buffer from potential future negative impacts of these factors. Based on the differential between the 2017 marginal cut-off grades and the applied production schedule cut-off grades, the applied 2017 Reserve cut-off grades were still deemed to be valid for Reserve purposes in the Platreef 2022 FS.

For areas designated as early mine production, an NSR cut-off value of $130/t was targeted for the identification and design of the Longhole Stopes. Areas within the early production that could not support a $130/t NSR cut-off were in-filled with $100/t NSR cut-off stopes to increase initial mined grades and provide increased revenue early in the mine life. The cut-off value was lowered to $100/t for areas mined later in the mine life. Lowering the cut-off grade ensures that adequate reserves are available to satisfy Ivanhoe's requirement of a 30-year mine life after mill start-up. Stope End Slash cuts and Drift-and-Fill shapes were generated at a $155/t cut-off.

In Phase 1 (700 ktpa) of the Platreef 2022 FS, stopes with the 3PE+Au grade greater than 4.5 g/t were targeted. Also, just development with the 3PE+Au grade greater than 4.0 g/t was counted as Ore. This provided increased revenue in early years of the mine. A definitive mine plan based on detailed stope layouts supports the Mineral Reserve. Due to irregularities in the geometry of the mineralised zones, not all material meeting cut-off grade can be mined without incurring some dilution. Due to inefficiencies in final mining recovery from the stopes, small amounts of mineralised material are lost during final stope cleanout, and additional losses may occur in transit from the stopes to the mill. Hence, a mining recovery factor is applied to the diluted resources to account for these losses.

The design parameters for the mining areas are based on geotechnical recommendations provided by SRK. The stope orientation and dimensions are based on a recommended maximum hydraulic radius of 8 m. SRK divides the deposit into five major geotechnical zones, with recommendations for the best stope orientation within these zones.

A series of well-defined stope shapes was generated for the entire mining area. After completion of initial stope designs, the deposit was segregated into 17 mining zones. These stope shapes were then used to query the block model and report tonnes and grades within the shapes.

The variability of factors related to mining, metallurgy, infrastructure, permitting, and other areas relevant to the Mineral Reserve calculation, the cost-per-tonne cushion between economic mining cost ($47.71/t–$58.53/t) and production schedule NSR cut-offs ($80/t and $155/t) will provide protection from future negative impacts of these factors.

Table 1.3 and Table 1.4 show the total diluted and recovered Probable Mineral Reserve for Platreef.

Table 1.3 Platreef Probable Mineral Reserve – Tonnage and Grades as at 26 January 2022

Method	Ore (Mt)	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Ore Development	11.0	142.4	1.79	1.85	0.27	0.12	4.03	0.15	0.31
Longhole	93.9	152.2	1.88	1.95	0.29	0.13	4.25	0.16	0.33
Drift-and-Fill	20.3	183.6	2.30	2.25	0.37	0.15	5.07	0.18	0.37
Total	125.2	156.4	1.94	1.99	0.30	0.13	4.37	0.16	0.34

1. Mineral Reserves have an effective date of 26 January 2022. The Qualified Person for the estimate is Curtis Smith, B. Eng., MAusIMM (CP).
2. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
3. Metal prices used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

Table 1.4 Platreef Probable Mineral Reserve – Contained Metal as at 26 January 2022

Method	Ore (Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Ore Development	11.0	0.6	0.7	0.1	0.04	1.4	37	76
Longhole	93.9	5.7	5.9	0.9	0.40	12.8	336	687
Drift-and-Fill	20.3	1.5	1.5	0.2	0.10	3.3	83	166
Total	125.2	7.8	8.0	1.2	0.54	17.6	455	929

1. Mineral Reserves have an effective date of 26 January 2022. The Qualified Person for the estimate is Curtis Smith, B. Eng., MAusIMM (CP).
2. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
3. Metal prices used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

1.12 Geotechnical Investigations

Much of the work done for the Platreef 2017 FS was used in the Platreef 2022 FS. The primary aim of the 2017 investigation was to increase the level of confidence in the current geotechnical database and to undertake various analyses, based on data from the mine site and laboratory testing, to provide geotechnical design parameters and optimise the mine design going forward. Following the completion of the 2017 FS, a detailed geotechnical investigation for the updated mine design was carried out. This work focused on the initial production period, with specific reference to the Drift-and-Fill mining (DF). No additional geotechnical data was provided, except for mapping conducted by Ivanplats during the development of vertical shaft No. 1 and respective station developments.

The geotechnical investigation was based on all available geotechnical and structural data, and included data specifically derived from geotechnically logged boreholes. Laboratory rock strength testing and stress measurement testing was also conducted to better understand the rock properties and the local stress regime. Local and regional seismicity have also been assessed. From the study, geotechnical design parameters have been derived to manage potential geotechnical risks that the mine may face. These parameters govern stope and mine access design and include the backfill and support requirements. The mine design has been reviewed and is generally in line with the geotechnical parameters provided.

Overall, the Tshukudu fault remains a major geotechnical hazard as it is often characterised by very poor-quality rock. Development through the fault should thus be planned carefully to avoid delays and costs. As the Tshukudu fault strikes from north to south and traverses the entire lease area, some development through the Tshukudu fault will be essential to provide access to ore to the west of the fault. Specialised support comprising resin injection, arch sets and void filling to be carried out by a specialist contractor is recommended for this case. An indication of the slow rate of this development is also provided.

Following the identification of the Tshukudu fault it has been established that the type of alteration within the fault is variable, indicating that improved characteristics of the fault zone in some areas may exist. It is therefore possible to develop through the fault in these cases with fewer delays and less intensive support, provided that there is no water ingress. Geotechnical drilling will be required to delineate and characterise the Tshukudu fault during implementation.

The Platreef project area is traversed by faults, low angled features (LAFs) and weaker chromitite partings which have the potential to create adverse ground conditions such as key block creation and falls of ground. Support strategies have been designed to cater for these features.

The updated mining layout and schedule includes a few cases where undermining of previously backfilled drift and fill and longhole stopes. Backfill dilution can be mitigated by increasing the cement content in the backfill of the stope to be undermined. However, in the case of undercutting drift and fill stopes, up to 25% for a single cut and 50% for a double cut must be anticipated.

Tightfilling in the drift and fill is essential to ensure successful mining. Instances where poor tight filling has occurred may lead to signficant stability problems, particularly in the backs of tertiary drifts. Long anchors will be required to support the backs and if the problem is more widespread, it is likely that some tertiary drifts will be abandoned.

1.13 Mining

The Platreef 2022 FS includes two main Phases. Phase 1 is defined after Shaft 1 changeover until Shaft 2 reaches to the full hoisting capacity of 6.19 Mtpa. In this Phase, Shaft 1 will be utilised with the 825 ktpa rock-hoisting capacity (including up to 125 ktpa allocated for development rock). Phase 2 commences just after equipping Shaft 2 when production begins to ramp up to 5.2 Mtpa.

Key points of the Platreef 2022 FS:

- Shaft 1 changeover completed for permanent hoisting in February 2022.
- Start development from the bottom of Shaft 1 in April 2022.
- Shaft 1 capacity is limited to ~700 ktpa ore, plus waste development.
- Initial development focus from Shaft 1 is a ventilation raise, completed by February 2024.
- Reduced initial development, focusing on the nearest, highest-grade stopes.
- Shaft 2 sinking recommences in September 2023. This is a discrete decision, and can be started at any point in time, depending on funding.
- Base case is a 770 ktpa concentrator on site.
- Assumes dry stacked tailings dam (for on-site concentrator).

1.13.1 Mining Method and Mine Design

For the Platreef 2022 FS, changes were made to the mine design to meet the requirements for the Phased Development Plan including additional ventilation and exploration development and waste and ore handling system. Also, design changes were made to avoid developing through the Faults. However, the changes did not exceed more than 10% and the majority of the mine designs are the same as the Platreef 2017 FS. The development and the production schedules have been adjusted to reach the high-grade profiles in the early years and then ramp up to the steady state ore production of 5.2 Mtpa.

The Platreef 2022 FS is a phased development plan trying to make revenue from the smaller plant in the early years to be used as funding for the expansion phase. For this reason, high-grade stopes with the appropriate underground infrastructure around Shaft 1 but additional exploration and ventilation development compared to the 2017 FS mine design was taken into consideration.

The Platreef 2022 FS includes two main Phases. Phase 1 is defined after Shaft 1 changeover until Shaft 2 reaches to the full hoisting capacity of 6.19 Mtpa. In this Phase, Shaft 1 will be utilised with the 825 ktpa rock-hoisting capacity (including up to 125 ktpa allocated for development rock). Phase 2 commences just after equipping Shaft 2 when production begins to ramp up to 5.2 Mtpa.

The Platreef Project is designed based on highly-mechanised Longhole Stoping and Drift-and-Fill mining methods.

The Platreef 2022 FS evaluates a phased development of Platreef, with an initial 700 ktpa underground mine and a 770 ktpa capacity concentrator, targeting high-grade mining areas close to Shaft 1. First concentrate production for this option is targeted in 2024, with the sinking of Shaft 2 recommencing in Q3'23, to coincide with the construction of two 2.2 Mtpa concentrators to be completed by 2028 and 2030. This would increase the steady production to 5.2 Mtpa by using Shaft 2 as the primary production shaft.

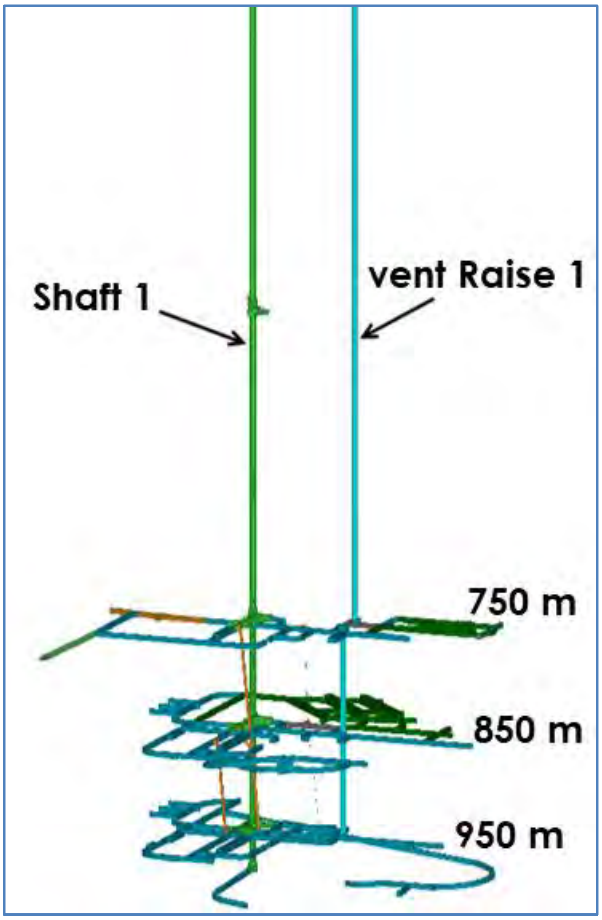
1.13.1.1 Mine Access

During Phase 1 of the Platreef 2022 FS, the main access to the mine will be via a 996 m deep, 7.25 m diameter ventilation shaft (Shaft 1). Development will be commenced around Shaft 1 via the principal access/ haulage levels (the 750 m, 850 m, and 950 m) and a series of interconnecting ramps.

After Ventilation Raise 1 is established, exhaust fans will be installed underground to provide a flow-through ventilation system with Shaft 1 as the ventilation intake. Then, until first ore production, the development needed for 825 ktpa will be run.

Figure 1.4 shows the Shaft 1 and Ventilation Raise 1 locations and main access levels in an elevated view (looking west).

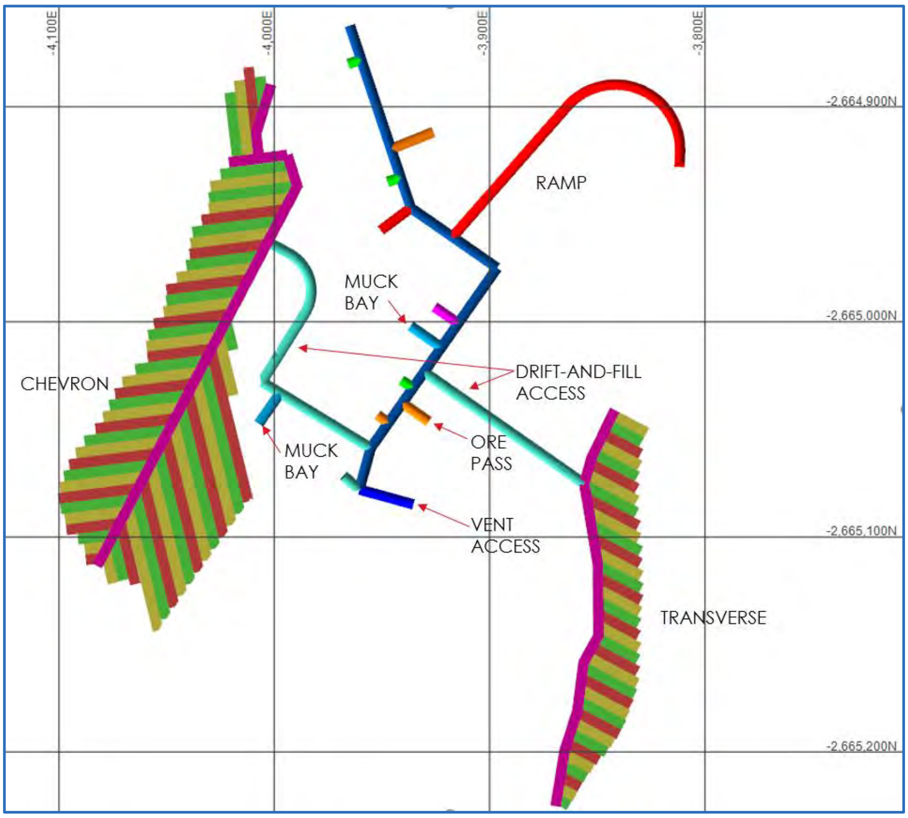
Figure 1.4 Mine Access Isometric Looking West



Mining access ramps will connect the haulage levels with the mining sublevels and other infrastructure. The mining sublevels will be developed from the ramps at regular vertical intervals and Drift-and-Fill access ramps will access the production areas. Ventilation Raise 1 and ore passes will also connect the production areas with the main haulage levels. A typical production area is shown in Figure 1.5.

Transverse or longitudinal stoping on a chevron or perpendicular layout will need to be determined when the detailed stope designs are prepared. This will be based on the information gathered from development and the close spaced drilling programme.

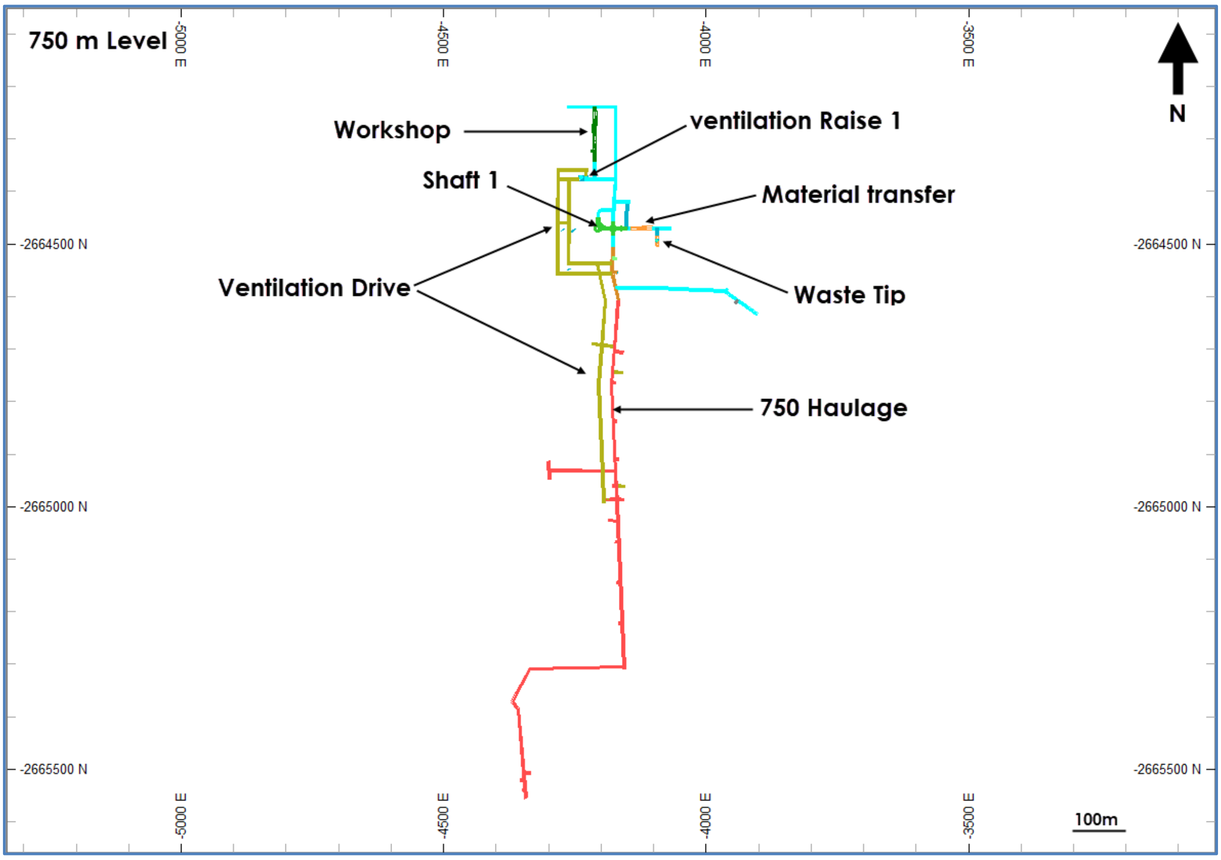
Figure 1.5 Typical Production Area – Plan View



During Phase 1, any development before the first production in Sep-24 is initial development. Then the development progresses towards the stopes in Phase 1 and underground infrastructure will be developed to cater the steady state production rate in Phase 2.

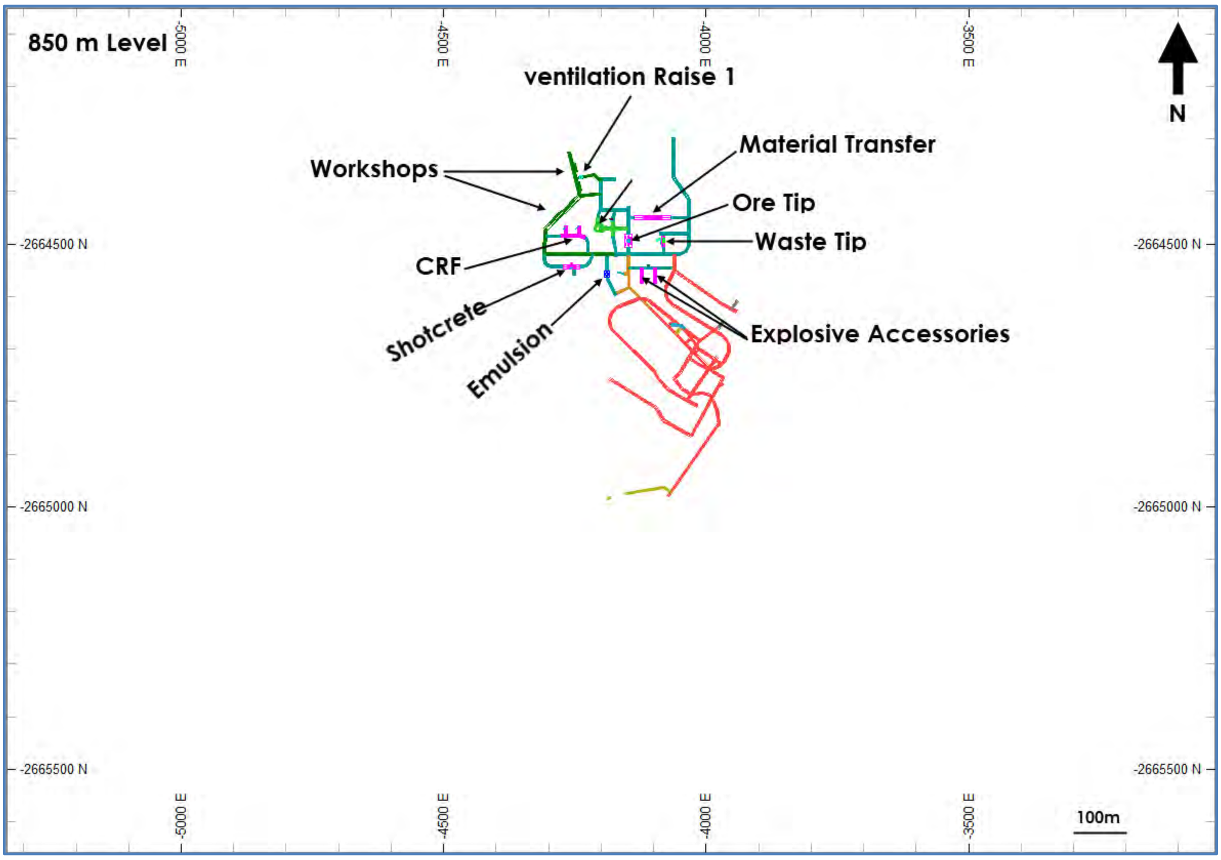
Figure 1.6, Figure 1.7 and Figure 1.8 show the plan view of the initial development on three main levels of 750 m, 850 m, and 950 m. A North-South view of the initial development is shown in Figure 1.9.

Figure 1.6 Platreef 2022 FS – Initial (Pre-Production) Development on 750 m Level



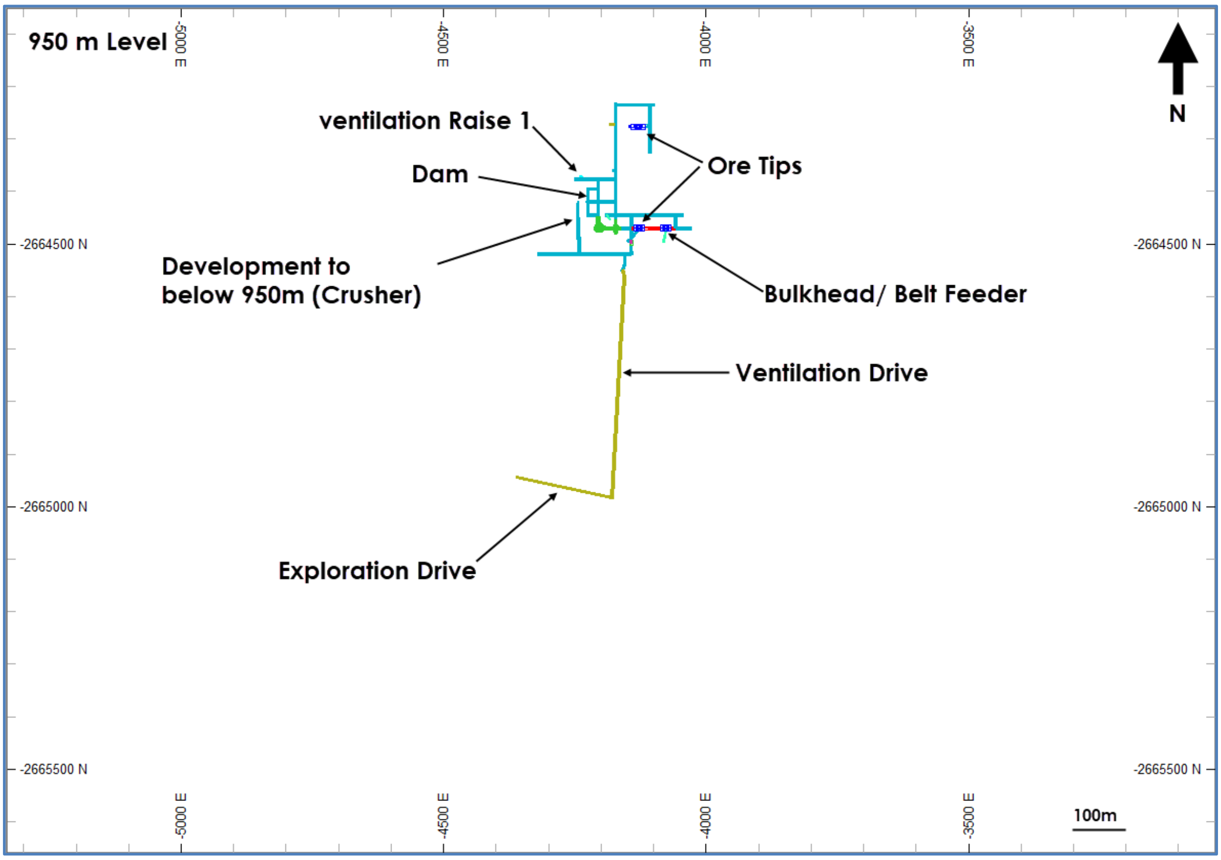
OreWin, 2021

Figure 1.7 Platreef 2022 FS – Initial (Pre-Production) Development on 850 m Level



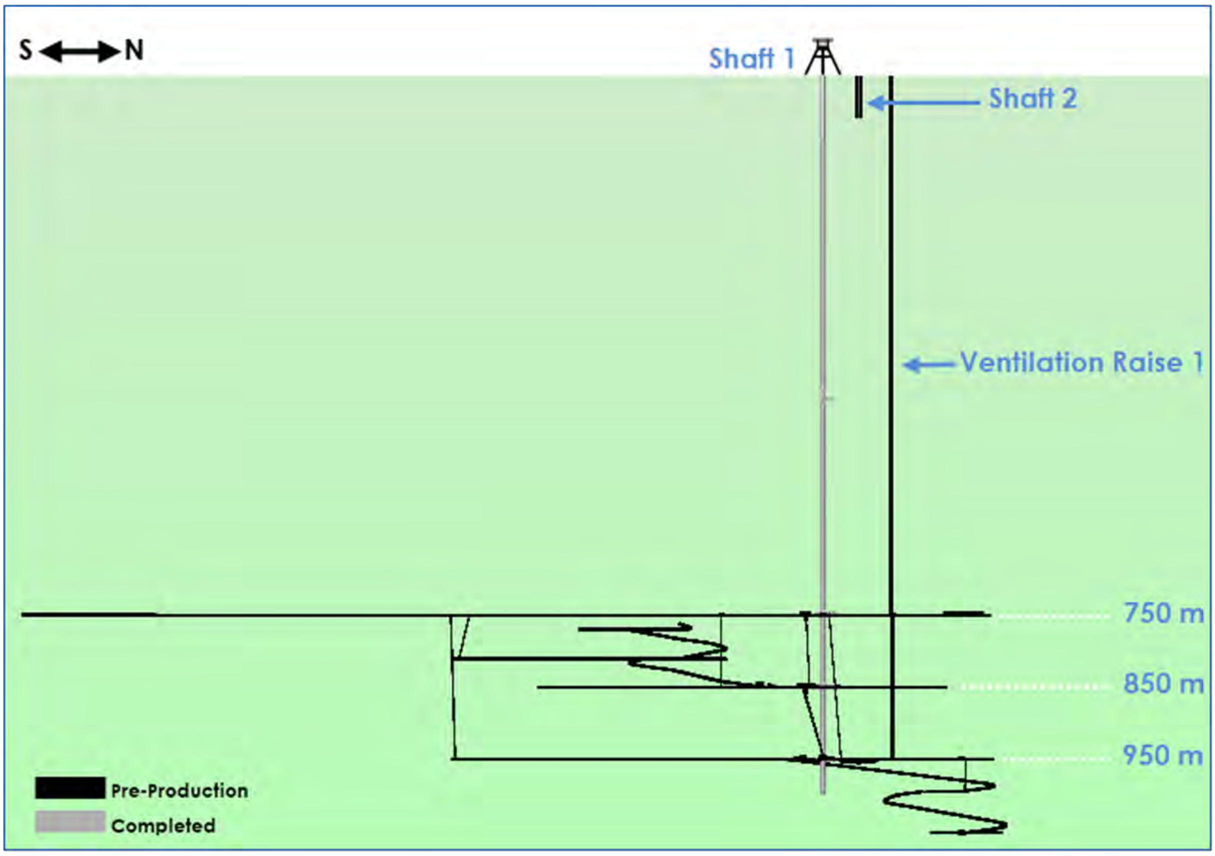
OreWin, 2021

Figure 1.8 Platreef 2022 FS – Initial (Pre-Production) Development on 950 m Level



OreWin, 2021

Figure 1.9 Platreef 2022 FS – Initial (Pre-Production) Development

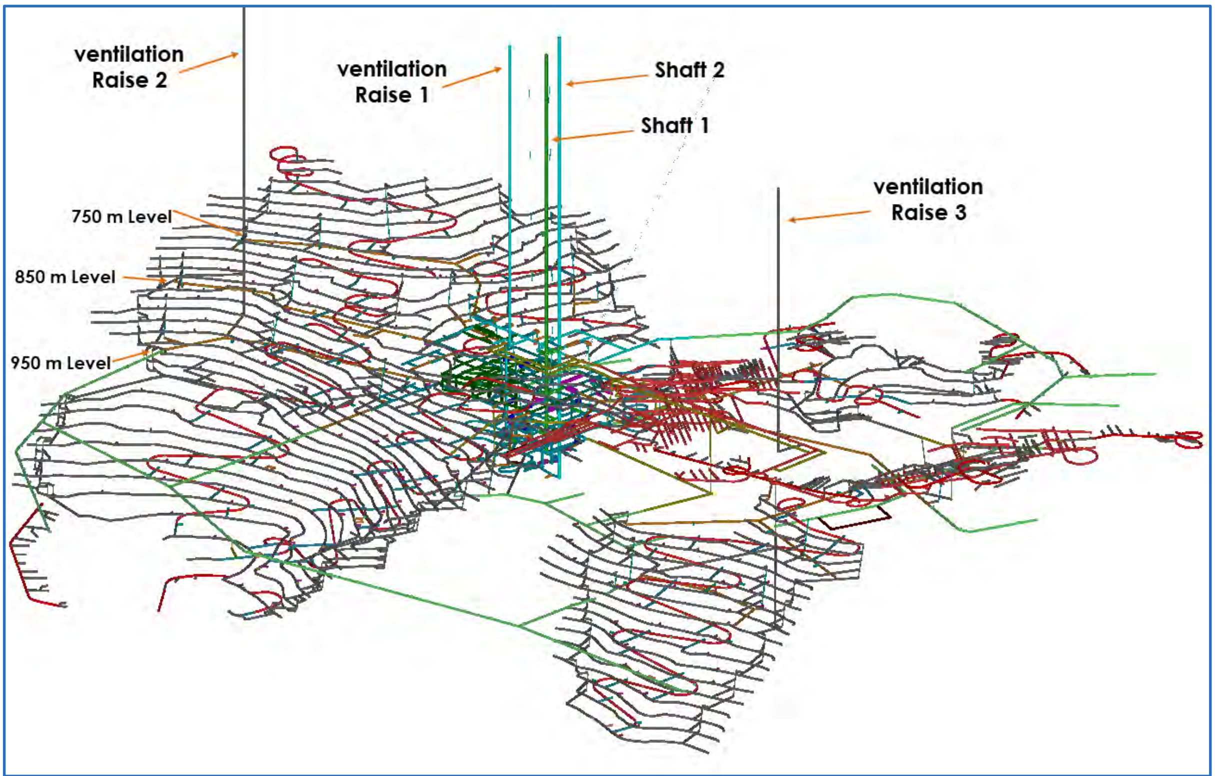


OreWin, 2021

In Phase 2, primary access to the mine will be by a 1,100 m deep, 10 m diameter production shaft (Shaft 2) and Shaft 1 will remain as a secondary access to the mine. During mine production in Phase 1, Shaft 1 will serve as a ventilation intake too. In Phase 2, both shafts will also serve as ventilation intakes. Three additional ventilation exhaust raises (Ventilation Raise 1, 2, and 3) are planned. Ventilation Raise 1 will be a 950 m deep, 6 m diameter raise located near the centre of the mining area and adjacent to the two intake shafts. Ventilation Raise 2 will be an 800 m deep, 6 m diameter raise located near the northern edge of the mining area. Ventilation Raise 3 will be a 725 m deep, 6 m diameter raise located near the southern edge of the mining area.

Three main access levels will be established as primary haulage levels. These are the 750 m, 850 m, and 950 m Haulage Levels. Figure 1.10 shows the proposed shaft and raise locations and the main access levels in an elevated view (looking north-east). Mining access ramps will connect the haulage levels with the mining sublevels and other infrastructure. The mining sublevels will be developed from the ramps at regular vertical intervals in the production areas. Drilling and extraction levels for stopes will be driven from the sublevels. Ventilation raises and ore passes will also connect the sublevels with the main haulage levels.

Figure 1.10 Mine Access Layout

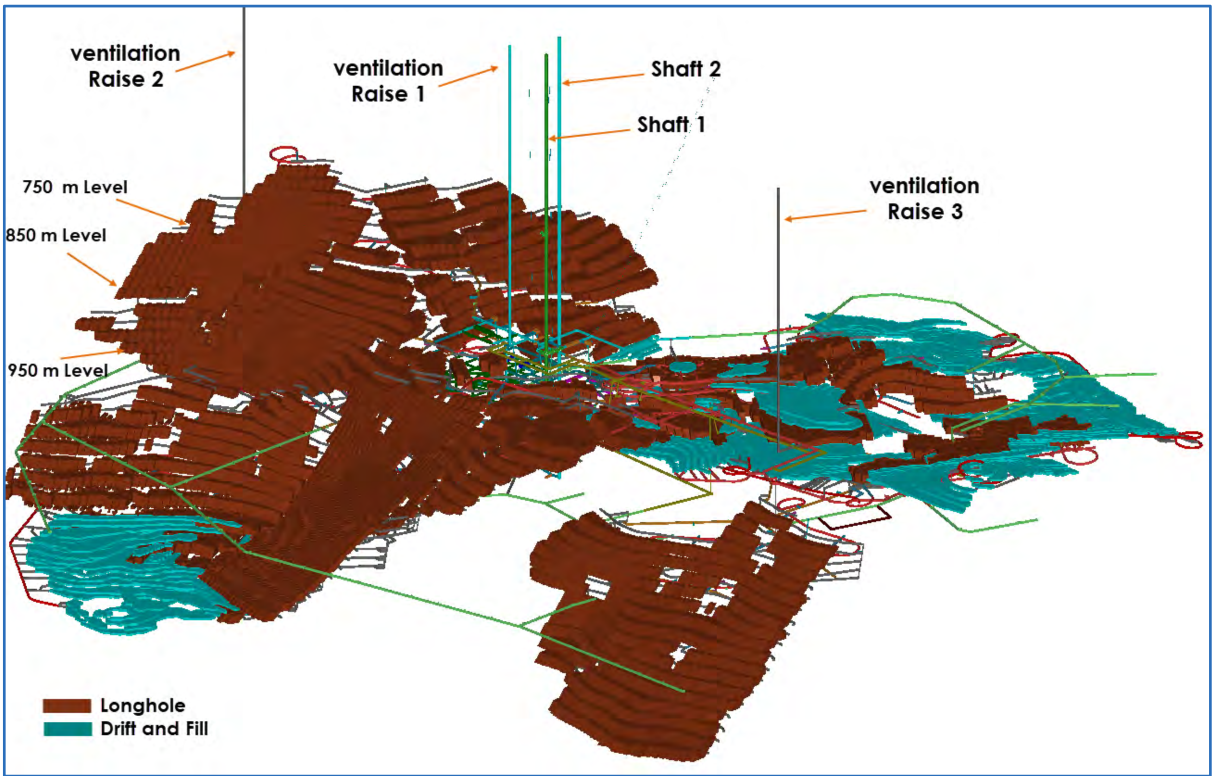


OreWin 2021

1.13.1.2 Mining Methods

The main mining methods will be Longhole Stoping and Drift-and-Fill mining. These methods provide a safe, mechanised, and productive mining plan. Longhole Stoping will be used in areas where the ore zone thickness exceeds 20 m and stopes will be oriented in a transverse or longitudinal fashion. Drift-and-Fill will be used in areas where the ore zone thickness is less than 20 m and will be mined in 5 m heights. Mining will be proceeding in an overhand fashion. Figure 1.11 is an isometric view of the mining areas by method (looking north-east). Mining methods will be explained in detail in Section 16.2.

Figure 1.11 Elevated View of Mining Areas by Method



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1.13.2 Development Plan

Mine development has been broken down into four main stages.

- Stage 1 – Lateral Development During Shaft 1 Sinking.
  - The first stage is the lateral development off Shaft 1 during shaft sinking. Lateral development is limited to stations off the shaft in preparation for level and infrastructure development and construction.
- Stage 2 – Lateral Development after Shaft 1 Sinking (Ventilation Restricted).
  - The second stage is the level and infrastructure development around Ventilation Raise 1. Until Ventilation Raise 1 is completed, a limited amount of ventilation is available, which limits the number of development crews. The priority during this stage is to commission Ventilation Raise 1 so that ventilation can be increased, and additional development crews can be added.
- Stage 3 – Lateral Development after Shaft 1 Equipping (Ventilation Raise 1 in place).
  - The third stage is the ramp-up and acceleration of level development off Shaft 1 until Shaft 2 is completed and functional. This stage is no longer limited by ventilation but is limited by the 2,500 t/d hoisting capacity of Shaft 1.
- Stage 4 – Lateral Development after Shaft 2 Commissioned.

- The fourth stage occurs after Q3'27 when Shaft 2 is commissioned for full ore and waste rock-hoisting. Additional production crews can now be added to meet the 39-month production ramp-up period to full production of 5.2 Mtpa. Development during the 22-year full production period focuses on just-in-time development and infrastructure to meet the production schedule.

1.13.3 Underground Infrastructure

Underground infrastructure entails dewatering, rock handling, cemented rock fill, paste fill and other infrastructure that includes workshops, re-fuelling stations, explosive storage and shotcrete facilities.

1.13.3.1 Dewatering Infrastructure

Mine production return water will include drill water, mine service water, fissure water, backfill flush water, and backfill seepage. All development drives on the main levels will be driven on a positive gradient and will include a ditch system to allow mine production water to flow back to a series of collection sumps spaced every 400 m throughout the mine workings.

The Platreef underground dewatering system will be separated into a face dewatering, tertiary dewatering, secondary dewatering, and primary dewatering system. Face dewatering will make use of electric face pumps powered by the drill rigs to transfer water from the face to the collector sumps. The tertiary system will transfer water from the collector sumps to the secondary dewatering system with larger dams.

There are three types of dams in the secondary dewatering system, gravity transfer dam, transfer dam and settling dams During mine production, water will be collected in sumps located throughout the mine. The upper level sumps will The gravity transfer dams will be equipped on the upper levels that will transfer water through boreholes to lower levels. On the lower levels gravity cannot be used hence, transfer dams equipped with suitable pumps will be installed to pump the water back to the primary dewatering infrastructure. Settling sumps will be utilised in areas further away from the shafts and will include a de-gritting/settling section on the inflow side of the dam before the water is transferred back to the primary system. sumps where mine duty pumps will then transfer the discharge pumps) water through a pipeline and borehole system to the main pump.

The primary pumping system will be a dirty water pumping system for Phase 1, located at Shaft 1, that will cascade water from 950 m Level to 450 m Level and then to surface where settling and clarifying will be done. During Phase 2, a UG settling system and a clear water pumping system directly from 1,050 m Level to surface via Shaft 2.

Mine water inflow is estimated at 35 L/s during maximum production in the Platreef 2022 FS. The pumping system is designed for 150 L/s to account for spikes from initial groundwater inflows and paste backfill flushing which will accommodate the higher inflows associated with the increased production rate. Each sump will not operate on a continuous basis but will have an overall utilisation of approximately 25%. This system will allow for improved settling in the main sumps and less fines reaching the main pumps at Shaft 2.

1.13.4 Ore and Waste Handling System

1.13.4.1 Phase 1 Ore and Waste Handling System

The Phase 1 rock handling system will initially be a LHD tip on 750 m Level and 850 m Level that will be connected by a rock pass. The material loaded on these levels will land inside a muck bay on 950 m Level and will be loaded with a LHD into a tip at Shaft 1 loading flask. As the mine starts with ore production a Phase 1 permanent rock handling system will be constructed. The Phase 1 permanent rock handling system will comprise a truck tip for ore and waste on 850 m Level and a waste only truck tip on 850 m Level that will feed onto a conveyor on 950 m level (Skip Feed conveyor). This conveyor will feed directly into the loading flask. Material generated on 950 m Level will be loaded onto the same belt directly by installing a LHD tip over the belt.

Longhole stoping will also be introduced during Phase 1 and a crusher will be required for this material. One of the two crushing systems designed for Shaft 2 will be commissioned for this purpose. The crusher system will be loaded on 950 m Level and the crusher will discharge the crushed material on 1,050 m level. Trucks will be used to transfer the material from 1,050 m Level back to 950 m Level. On 950 m Level a truck tip with a conveyor will feed the crushed material onto the skip feed conveyor.

1.13.4.2 Phase 2 Ore and Waste Handling System

The ore and waste handling system is designed with a capacity of 6.19 Mtpa to meet the steady state production requirements of 5.2 Mtpa of ore plus waste but allows for future potential increased ore production. Two ore handling facilities and one waste handling facility are planned. In addition, there are two underground crushing stations, one below each of the coarse ore bins midway between the 950 m Level and the 1,050 m Level loading station.

Ore will be loaded from the stopes and ore development headings using Load Haul Dump (LHD) loading units. LHDs used in mucking Longhole Stopes will be operated via remote control as required.

For mining areas located above the 950 m Level, ore will be hauled by LHD from the stopes to grizzly stations on each sublevel. Finger raises into the ore passes will transfer the ore to the 850 m or 950 m main haulage levels. At these haulage levels, the ore will be chute loaded into trucks for haulage to the truck ore dumps near Shaft 2.

For mining areas located below the 950 m Level, ore will be loaded directly into trucks by LHD. The trucks will then haul the ore up the access ramps to the 950 m Level and then to the truck ore dumps near Shaft 2.

On each of the three main haulage levels, two ore and one waste truck dump will be located near Shaft 2. The truck dumps will be equipped with grizzlies and fixed hydraulic rock breakers. One main waste pass will extend from the 750 m Level to the 1,050 m Level loading station. Two main ore passes will extend from the 750 m Level to the 950 m Level. At the 950 m Level, the ore will be diverted to the coarse ore bins. The 950 m Level truck dump grizzly will directly feed the coarse ore bin.

There will be two crusher stations, one below each coarse ore bin, midway between the 950 m Level and the 1,050 m Level loading station. Rock will be reduced in size by the jaw crusher and fed into the crushed ore bin. Each crusher will have a capacity of 3 Mtpa for an overall capacity of 6 Mtpa. This will allow for future production plan increases with the current rock handling infrastructure.

Apron feeders will load the crushed ore or sized waste onto the high-speed weigh conveyor belt on the 1,050 m Level. A skip-load of ore or waste will be loaded onto the high-speed conveyor while the skips are in transit in the shaft. Upon arrival at the loading station, the load will be discharged directly from the conveyor into the skips via diverter chutes.

On surface, ore and waste will be discharged from the skips into the headframe bin. Discharge conveyors will transport it to either the mill or waste storage area.

1.13.5 Other Infrastructure Underground

Fleet maintenance will be done underground, and workshops will be constructed on each main level neasr the shafts. Satellite workshops will also be located closer to working areas to service primarily the slow-moving equipment such as the drill rigs. The workshops will be fit for purpose, equipped with the necessary lifts, cranes, ramps and tools for major servicing of large mobile equipment.

Refuelling stations will be constructed on each of the main levels with a fuel line from surface, delivering fuel to 750 m, 850 m and 950 m Levels. 850 m and 950 m Level will be equipped with a refuelling station on the south and the north of the shaft and 750 m Level will only have one station.

Shotcrete is required in primary access drifts for support and hence a shotcrete facility will be installed on 850 m Level. The shotcrete mixture will be prepared on surface, without adding the fibres, and sent underground via boreholes. Underground the fibres will be added and the final mixture will be loaded into suitably sized agicars that will transport the material the area of application.

An emulsion drop facility will be installed during the latter parts of Phase 2. This system will have surface storage tanks of emulsion and sensitizer and the emulsion will be dropped down a pipe, installed in a borehole down to 850 m Level and 950 m Level directly. Underground the emulsion will be stored in batch tank and dispensed into the emulsion cassette. Sensitizer will be transported underground separately in tanks and will fill the explosive cassettes as required.

Explosive accessories that include high explosives, detonators, shocktubes, etc. will be transported underground in an explosive truck, which will be a dedicated LDV marked and equipped for these purposes. Accessories and detonators will be stored in underground magazines located on 850 m and 950 m level. Explosives will be issued from these magazines for daily use. Old explosives will be stored in old explosive boxes and transported to surface for destruction.

1.13.6 Mine Ventilation System

The Platreef Project will use highly mechanised mobile equipment for the underground mining operations (e.g., LHDs, trucks, drill rigs, personnel carriers). Ore transport will be via trucks to the main ore passes and collection points. Development of the main accesses and sublevels will be through drill-and-blast operations as well as LHDs, trucks, and drill rigs. Internal ramps will be used to facilitate truck hauling from stoping and development sections to the main haulages. For this reason, an annual ventilation estimate was developed based on requirements for mobile equipment, although it was also heavily influenced by the heat load calculations and required refrigeration. As a pull system arrangement, fresh air will be downcast via the main shafts and exhausted through a number of internally bored raises serving each mining sublevel and the main exhaust shafts.

A bulk air-cooling system located on the surface was identified as the best option due to its simplicity and low construction and maintenance costs. The proposed refrigeration plant is designed for 28 MW of refrigeration capacity. All cooled air will intake into the mine via Shaft 1. An additional 10% in capital costs for Ventilation Infrastructure and Cooling was added to allow for the system to accommodate the production increase.

1.14 Metallurgical Testwork Overview

Various metallurgical testwork campaigns have been conducted since October 2001 on a number of drill core samples originating from the Platreef deposit. Metallurgical testwork has been focused on providing data for flow sheet development whilst aiming to maximise the recovery of platinum group elements (PGEs) and base metals, mainly nickel, and producing an acceptably high-grade concentrate suitable for further processing and/or sale or toll treatment by a third party.

Prior to 2006, testing was predominantly conducted on lower-grade PGE material from the potentially large open-pit area. In 2008, a deep drilling exploratory programme was launched, and the resource was updated to include deeper higher-grade PGE material.

Between 2010 and November 2014, a series of metallurgical testwork campaigns were carried out on the Platreef mineralised material. This, named Phase 1–6, testing included comminution characterisation, bench scale flotation testing and laboratory scale dewatering testwork. The findings from this, Phase 1–6 testing, were presented in Platreef 2015 PFS.

As part of the Platreef 2017 FS, comminution variability testing was conducted on approximately 1,286 kg of HQ drill core representing the geometallurgical units T1, T2U, T2L and the Contaminated Zone (footwall). Flotation testing was conducted on approximately 1,140 kg of quarter PQ drill core samples representing the geometallurgical units T1, T2U, T2L as well as the Contaminated Zone (footwall). Testing was conducted at the Mintek laboratories in Johannesburg, South Africa.

The Platreef 2017 FS testwork programme included, comminution variability testing, mineralogical characterisation, open circuit flotation development and optimisation testwork, open circuit flotation variability testwork, bench scale locked cycle testwork and tailings dewatering testwork.

Further test work was conducted in 2020 and 2021 as part of the current Platreef 2022 FS. The aim of the Platreef 2022 FS test work campaign was to evaluate the potential for inclusion of an HPGR circuit, to further characterise the flotation response of Platreef composite samples, evaluate the potential for Jameson cell technology in the cleaner circuit, determine the effect on flotation response when using site water and conduct preliminary pilot scale test work to produce bulk concentrate samples for settling, filtration and Kell Process test work. The Platreef 2022 FS test work was conducted on drill core sample intervals that reflect drill core remainders from the 2017 FS variability test work campaign as well as two bulk samples comprised of crushed material from the current surface stockpiles.

The selection of samples, done in conjunction with the mining and geological teams, submitted for the Platreef 2017 FS metallurgical testwork and used in the Platreef 2022 FS, are testwork campaigns, deemed to be sufficient.

The Platreef 2015 PFS flow sheet was based on a single-stage milling circuit followed by flotation (MF1) using an oxalic acid and thiourea reagent suite with the inclusion of a post mill conditioning stage. At the time of publishing the Platreef 2015 PFS, MF1 testing of an alternate reagent suite containing a targeted copper collector indicated potential for a simpler flow sheet, using a more conventional reagent suite. Further development of this MF1 copper collector flow sheet was conducted as part of the Platreef 2017 FS. Testing confirmed that this simpler flow sheet was able to achieve a similar metallurgical response compared to the MF1 circuit using oxalic acid and thiourea. As per the Platreef 2015 PFS outcomes, flotation testwork confirmed an optimal target grind, for this flow sheet, to be 80% passing 75 µm. It was decided to base the remainder of the Platreef 2017 FS testing on the MF1 flow sheet, using the targeted copper collector reagent suite.

Comminution variability test work during the 2017 FS confirmed the previous testwork findings, indicating that the plant feed can be characterised as being hard to very hard and thus not suitable for Semi-Autogenous Grinding (SAG) milling. The comminution variability testwork indicated no significant difference in competency between the ore from the early production years (Year 1 to Year 5) and the later production years. Minor variation in hardness was noted for the T1 and T2U domain samples as compared to the T2L and CZ domain samples.

For the 90% confidence interval, Bond ball work indices were in the range 19.0–24.2 kWh/t. The Bond ball work index data has indicated an approximate 0.5 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1 to Year 5 samples. The abrasion index indicated that the ore can be classified as having a medium abrasion tendency.

The 85^{th} percentile comminution test data was used as the design basis for the crusher and grinding circuit. The crusher circuit was sized based on the 85^{th} percentile crusher work index results. The grinding circuit sizing was based using the 85^{th} percentile Bond work index data, which were used in combination with particle breakage rates derived from grindmill testing. The average ore hardness and abrasion index data has been used in the derivation of the operating cost estimates.

The Platreef 2022 FS test work programme included multiple single pass crushing tests by Thyssenkrupp using a semi-pilot scale HPGR (SMALLWAL). The test work was conducted on a bulk shaft intercept composite sample sourced from the current surface stockpiles.

The test work confirmed that Platreef samples are amenable to HPGR technology, achieving a specified throughput of 310 to 338 ts/m^{3}h with a specific energy ranging 0.94 to 1.47 kWh/t.

A three-stage crushing (primary crushing within mining scope) and ball mill circuit was identified as the preferred option, with lowest associated technical risk during the 2017 FS. A high level HPGR assessment which considered differential capital and operating costs, has indicated that an HPGR circuit offered no/limited benefit at the lower throughput rate for the 0.77 Mtpa Phase 1 concentrator plant. However, for the larger 4.4 Mtpa Phase 2 concentrator, this option offers the potential for an approximate 7% operating cost reduction for the operation of the crushing and milling circuits. This option will be considered in more detail during the phased implementation programme.

Bench scale, batch open circuit flotation testwork was performed during the 2015 PFS and 2017 FS to derive the optimal flow sheet. This development work, indicated that a MF1 flow sheet, using the targeted copper collector reagent suite and a split cleaner flotation circuit configuration was optimal. The split cleaner circuit allows for the fast-floating fraction to be treated in a separate cleaner to the medium and slow floating fractions, resulting in optimal 3PE+Au, Cu and Ni recoveries for the targeted concentrate grades.

Once the optimum flow sheet had been derived, bench scale, batch open circuit flotation variability testwork was performed.

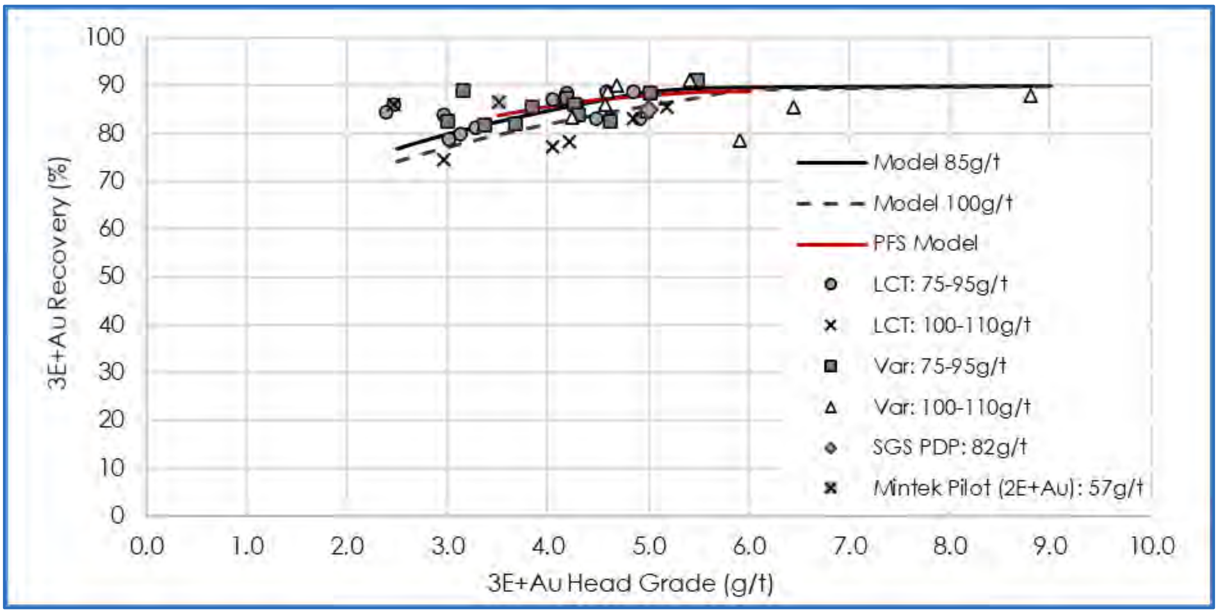
In addition to the open circuit variability testwork, locked cycle testwork was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore schedule, with the focus placed on testing of an ore blend representing the first five years of mining. This locked cycle flotation testing achieved recoveries (3PE+Au) in the range 83.1–88.7% with a final concentrate containing approximately 60–95 g/t (3PE+Au). Locked cycle testwork on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries (3PE+Au) in the range 77.3–85.5% with a final concentrate containing approximately 80–120 g/t (3PE+Au).

A further locked cycle test using the Platreef FS flow sheet, was conducted at SGS Lakefield in 2020. This test was conducted on a blend composite containing geometallurgical units from various drill core intervals in the ratio 23% T1, 45% T2U, 24% T2L and 8.8% CZ with a head grade of 5.1 g/t (3PE+Au). This test achieved a 3PE+Au recovery of 85% at a concentrate grade of 82 g/t (3PE+Au).

Locked cycle testwork indicated that targeting higher concentrate grades, in excess of 100 g/t (3PE+Au) resulted in reduced metal recoveries. 3PE+Au recovery was found to be dependent on the 3PE+Au head grade and target concentrate grade, which is typically referred to as an upgrade ratio in South African platinum processing terms. For Platreef, as higher 3PE+Au concentrate grades are targeted (High upgrade ratio), the overall concentrate mass pull will decrease, and consequently the 3PE+Au recovery would be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio).

This relationship between 3PE+Au head grade, mass pull and final concentrate grade was incorporated into the recovery modelling in order to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as summarised in Figure 1.12.

Figure 1.12 3PE+Au Recovery as a Function of Concentrate Grade



DRA, 2022

Further flotation test work was conducted in 2020 and 2021 as part of the current Platreef 2022 FS. This included open circuit, bench scale, batch flotation test work using filtered site representative of the expected water from the Masodi grey water system. These tests indicated that the filtered site water achieved a similar 3PE+Au upgrade profile as the baseline tests using Mintek tap water. There was however notable variance in the ranges for repeat baseline tests. A similar trend was observed for Cu and Ni.

Open circuit bench-scale Jameson cell test work was conducted at SGS in 2020 and Mintek in 2021. These tests, at both laboratories, indicated the potential for a significant reduction in first pass cleaner circuit 3PE+Au recovery relative to the baseline open circuit cleaner tests. A similar result was observed for Cu and Ni. The equipment vendor has subsequently indicated that the test work was conducted using an outdated procedure which may have impacted on the results. Repeat test work using the updated vendor procedure is scheduled to take place in the first quarter of 2022.

An initial mini-pilot plant commissioning run was conducted at Mintek in June 2021 with a further commissioning run in November 2021. The original aim of the mini pilot plant programme was to produce bulk concentrate samples for Kell hydrometallurgical refining test work and concentrate de-watering test work. Additionally, the intention was to derive process design information to supplement the design data as derived from bench scale flotation test work. The latter objectives were only partially achieved as the MINTEK mini-plant was not adequately commissioned, stabilized and optimized due to a number of operational challenges which included stoppages due to power interruptions, inability to consistently dose copper collector reagent on a continuous basis, mechanical breakdowns and lack of assay data for operational control.

Concentrates generated during the June 2021 pilot run were re-floated in an 80 L flotation cell in batch mode to produce timed kinetic samples for Kell test work and concentrate de-watering test work at Metso Outotec South Africa (MO).

The November 2021 commissioning run of the Platreef circuit was conducted on a low grade bulk shaft intercept samle with a measured 2PE+Au head grade of 3.8 g/t. The run achieved stable mass flows however large variances in final concentrate mass pull resulted in combined final concentrate grades of approximately 50 g/t to 78 g/t 2PE+Au at a mass pull of 5% to 8%. The averaged metallurgical projection data indicates that the run achieved an average concentrate grade of 57 g/t at a recovery of 87% and 5.3% mass pull. A copper recovery of 87% was achieved and a nickel recovery of 81% was achieved.

The mini pilot plant data from the June 2021 and November 2021 runs are considered to reflect preliminary commissioning results on low grade samples and do not reflect representative metallurgical performance data.

1.15 Recovery Methods

The 2017 Feasibility Study (2017 FS) was based on the development of a large scale, mechanized, underground mine accessed via two vertical shafts, with a processing plant and associated infrastructure. The Platreef 2017 FS was based on a processing rate of 4.0 Mtpa, aligned to the mine plan and schedule at the time. The process plant, however, was adequately sized to treat a maximum of 4.4 Mtpa (2 x 2.2 Mtpa modules). This higher processing rate has been utilised in the current Platreef 2022 FS.

Ivanhoe commenced with the construction of a development shaft (Shaft 1) in 2017. By April 2018, Shaft 1 had reached a depth of 750 m and construction of Shaft 2, which was to be the mine's main production shaft, then also commenced. An opportunity was identified to possibly generate early revenue by processing the Run-of-Mine (RoM) material that could be hoisted from Shaft 1. The 2020 Preliminary Economic Assessment (PEA) evaluated this phased production option and based on the positive outcome, the current Platreef 2022 FS proceeded based on constructing an initial concentrator plant with a nominal throughput of 0.70 Mtpa and a design throughput of 0.77 Mtpa to treat un-crushed RoM material hoisted through Shaft 1. This will be followed by the addition of the 2 off 2.2 Mtpa concentrator modules, constructed in parallel with the sinking of Shaft 2, as per the original 2017 FS to process primary crushed RoM material hoisted through Shaft 2. The 2017 FS process design basis remains unchanged with the exception of the required changes to cater for a dry-stack tailings facility. The Platreef 2022 FS is based on phased production ramp up from 0.7 Mtpa (Phase 1) to 5.2 Mtpa with the addition a 4.4 Mtpa concentrator plant (Phase 2).

The Platreef concentrator for each phase includes all ore processing requirements from the receipt of Run of Mine (ROM), either in storage silos or the feed chute of the crushing circuit, through to final concentrate load out and tailings disposal systems.

The Phase 1 design concept is based on a stand-alone 0.77 Mtpa small case concentrator with dedicated crushing and water recovery circuits. The Phase 2 concentrator design is based on two modular 2.2 Mtpa milling and flotation circuits which were selected based on the concentrator production ramp-up profile. This modular approach with shared crushing, tailings disposal and concentrate handling circuits, allows for increased processing flexibility and introduces process redundancy whilst allowing for phasing of capital.

The process design has been developed based on the testwork findings and assessments, various desktop level trade-off studies and relevant DRA design information. The Platreef concentrator flow sheet is based on a conventional three-stage crushing and ball milling circuit followed by flotation and dewatering stages. This single-stage milling and flotation process flow sheet is well known in the industry, and has been proven as a suitable processing route for various platinum ores. The flotation cleaner circuit design provides adequate flexibility to target different final concentrate grades. The process flowsheet for Phase 1 and Phase 2 is presented in Figure 1.13 and Figure 1.14 respectively.

The concentrator engineering design is based on previously constructed and proven DRA unit processes adapted for Platreef-specific requirements. Project-specific design criteria and specifications were developed to ensure conformity across the mine site and various contractors.

The engineering design has taken cognisance of the environmental and social impacts with regard to noise, dust, light and visual pollution. Dust extraction and suppression systems have been included to minimise fugitive dust release. Dust suppression is present on all ROM transfer points. Silos are specified with concrete roofs and fitted with dust extraction and filtration units. Noise-generating equipment was identified, independently simulated, and noise-attenuating cladding designs are included in the Phase 2 design where required.

The final concentrator layouts were developed with due consideration of any environmental and social impacts. The design aims to minimise transfer and wear points in the ROM section, reduce building height and improve both the constructability and maintainability in the main concentrator area.

The concentrator plants have been designed in accordance to the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities.

Figure 1.13 Phase 1: 0.77 Mtpa Concentrator Flow Sheet

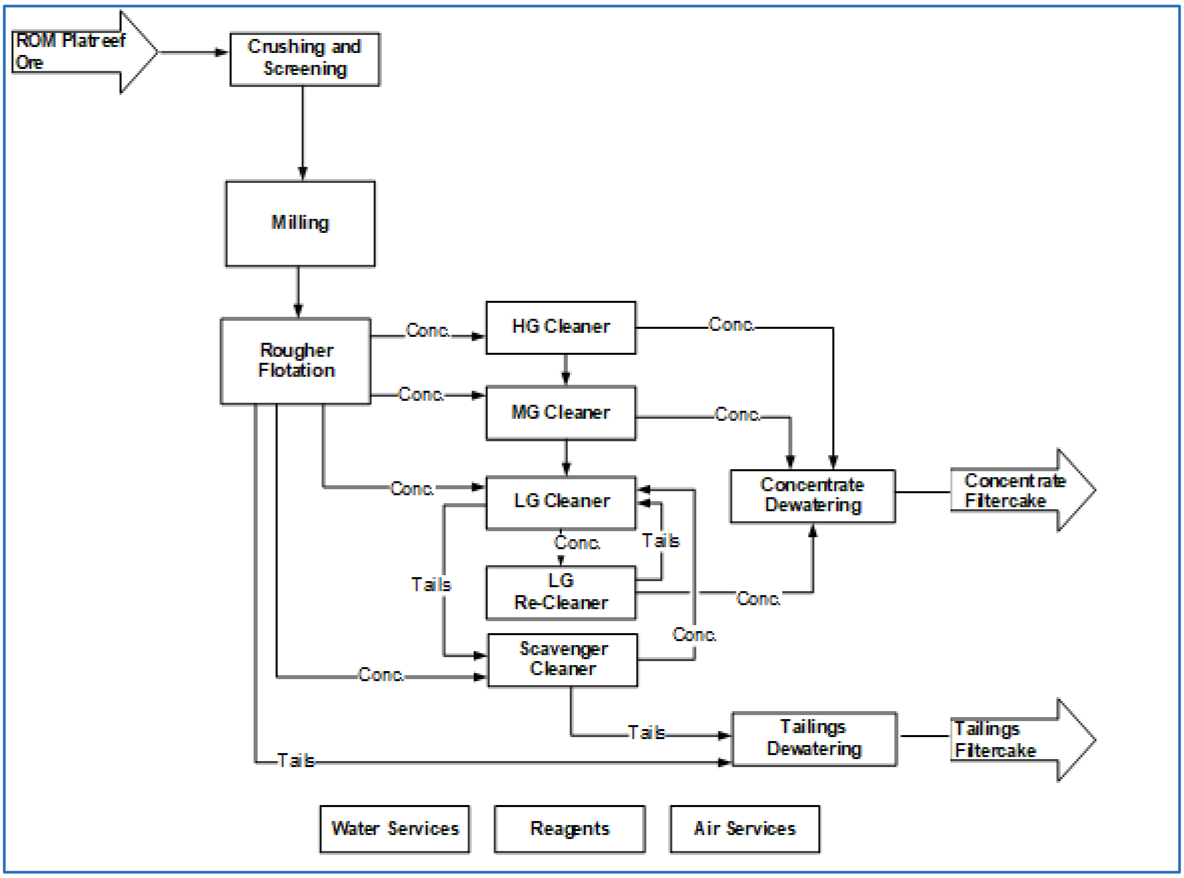
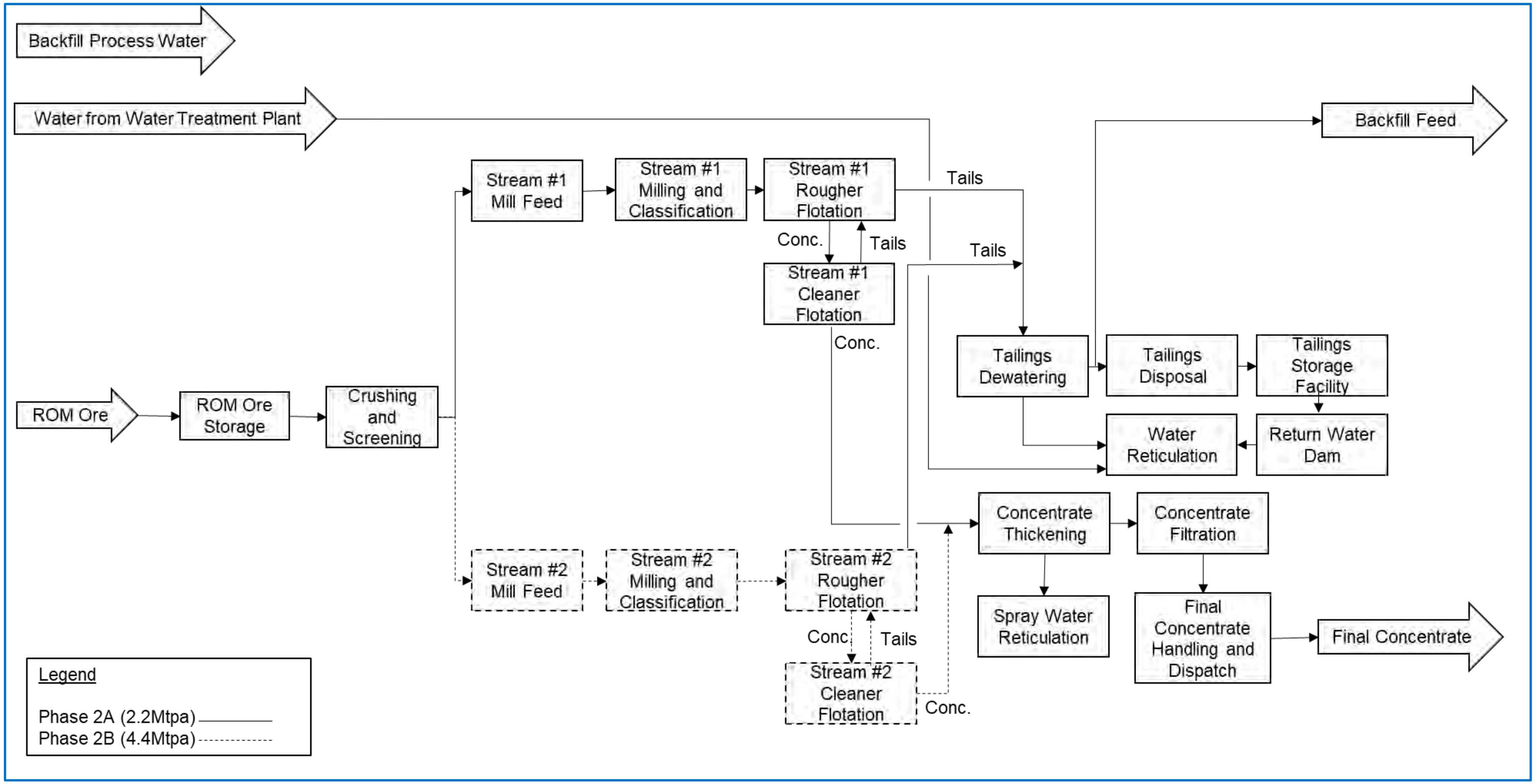


Figure 1.14 Phase 2: 4.4 Mtpa Concentrator Flow Sheet



DRA, 2021

1.16 Infrastructure

The Platreef Project site is located approximately 280 km north-east of Johannesburg in the Limpopo Province and falls under the Mogalakwena Municipality. The mine lease area is on the Turfspruit, Macalacaskop and Rietfontein farms. Year-round access to the site is by paved, all-weather national highway (N1) to Mokopane (formerly Potgietersrus). From Mokopane the access continues as a paved, all-weather national highway (N11). This road is a two-lane tarmac road suitable for heavy loads year-round.

The Platreef Project site is surrounded by many informal settlements and villages, with Ga-Kgobudi, Ga-Madiba, Ga-Magongoa, Mzombane and Tshamahansi being the closest. The close proximity of these villages to the Platreef Project site was taken into consideration in the design and engineering of all infrastructure and emphasised the importance of mitigating noise and dust pollution, as well as the visual impact that the Project will have on the communities. Figure 1.15 shows the Platreef Project location within local region.

Figure 1.15 Platreef Project and Local Region

Ivanhoe, 2017

1.16.1 Bulk Water

South Africa is a country of relatively low rainfall and, in particular, the Limpopo Province will need to augment their current water sources to meet the growing demand from the mining, power, agricultural, and domestic sectors. The Government has committed to addressing this shortage in the interest of developing the region. There are major planning, infrastructural design, and funding challenges that need to be addressed in order to ensure that the water availability is sufficient to meet development in the province.

The bulk water requirement for the mine is divided into the water required for construction and water required for operations. The water requirements include construction, dust suppression and water for developing the mine. The water volumes required for the development of the mine were based on recovering 50% of the water sent underground for mine development and no groundwater inflow. These assumptions will provide a conservative estimate of the water required for development. The sources of bulk water during the construction period are local groundwater abstracted from licensed boreholes on Ivanplats' Uitloop and Turfspruit properties, as well as storm water run-off collected on site. The yield from the boreholes on Uitloop and Turfspruit is sufficient to meet the construction bulk water requirements and will be used as the primary source of the potable water for the mine.

Ivanplats requires an average bulk water supply of 7,700 m^3/d for the operational phase of the mine. Ivanplats is actively pursuing the following sources of bulk water:

A supply from the Olifants River Water Resources Development Project (ORWRDP). This supply will be from the Phase 2B pipeline from the Flag Boshielo Dam.

A local source of treated sewage effluent (grey water) from the Masodi Waste Water Treatment Works (WWTW).

On 17 January 2022, Ivanhoe concluded an agreement to receive local treated water to supply most of the bulk water needed for the first phase of production at Platreef. The Mogalakwena Local Municipality has agreed to supply a minimum of three million litres of treated water per day from the town of Mokopane's new Masodi Waste Water Treatment Works (WWTW). Initial supply will be used in Platreef's ongoing underground mine development, surface infrastructure construction and plant operations. The agreement provides to increase the supply up to a maximum ten million litres of treated water per day, depending on the roll out of the supporting municipal infrastructure.

Under the terms of the agreement Ivanplats will provide financial assistance to the municipality for certified costs of up to a maximum of R248 million (approximately $16 million) to complete the Masodi WWTW. Ivanplats will purchase the treated water at a reduced rate of R5 per thousand litres for the first 10 million litres per day to offset a portion of the initial capital contributed.

Provision has been made to treat the water received from the Masodi WWTW to ensure it is suitable for use in the process plant. Ivanplats remains an active member of the Joint Water Forum, perusing the Phase 2B pipeline from the Flag Boshielo Dam as an alternative solution for future expansions.

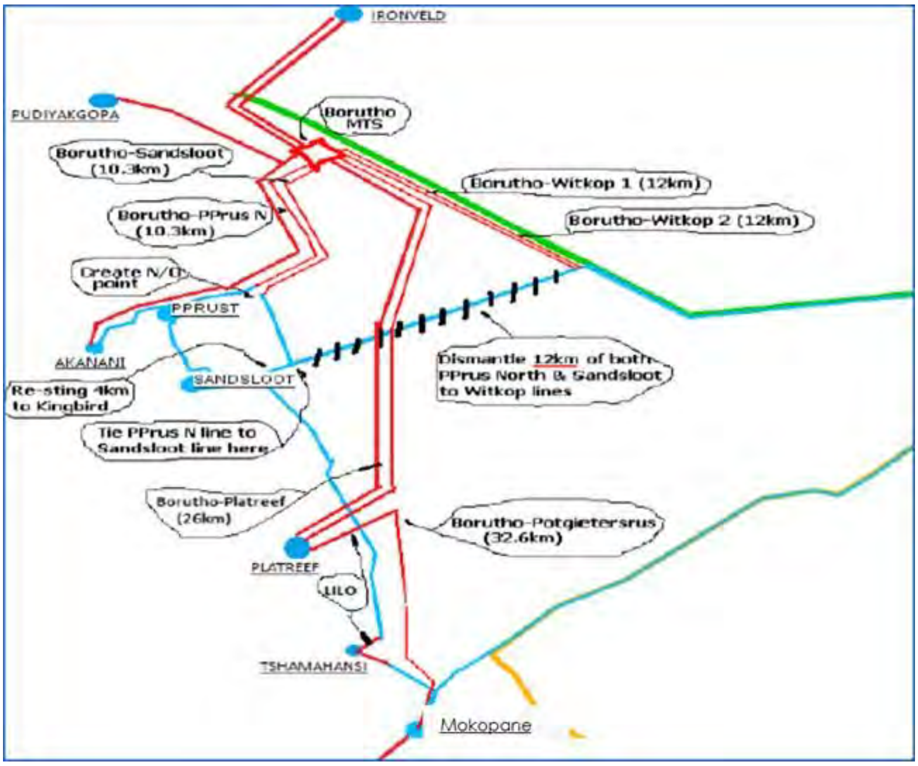
1.16.2 Project Power Supply

The bulk power supply is to be sourced from Eskom, the South African national power utility. On 24 February 2017, the initial five-million-volt-ampere (MVA) electrical power line connecting the Platreef site to the Eskom public electricity utility was energised and now is supplying electricity to Platreef for Shaft 1 and construction activities. The new power line is a collaboration between Platreef, Eskom and the Mogalakwena Local Municipality, and also established a platform to provide energy to the neighbouring community of Mzombane, which previously was without electricity reticulation and supply. The application for 100 MVA of power has been submitted to Eskom, and the budget quote received. The budget quote was accepted, and the appropriate deposits provided.

The original application requested a fully redundant premium supply project package from Eskom. The application scope has been updated to an Eskom 'self-build' project. Eskom has provided a complete design package for the works, and the construction of the works is a project responsibility. An approved Environmental and Social Impact Assessment (ESIA) together with the land and rights package for the works has been completed and received as part of the Eskom design package responsibility.

Upon completion of the works, these will be handed over to Eskom to form part of the utilities supply network. As represented in Figure 1.16, the Platreef Project is to be fed from the Eskom Borutho Main Transmission Station (MTS). Two 132 kV overhead line (OHL) feeder bays have been provided by Eskom.

Figure 1.16 Proposed Power Transmission



DRA, 2017

From the Borutho MTS 2 x 26 km Kingbird 132 kV OHLs are to be constructed to feed the Platreef 132/33 kV substation. At the Platreef 132/33 kV substation, 3 x 132/33 kV 40 MVA transformers are to be installed, with provision for future installation of an additional 40 MVA transformer. The supply is designed to provide N+1 redundancy on both the OHL and the transformers for up to 120 MVA. A future 4th transformer is catered for, and a 5th transformer can be added in order to maintain redundancy should it be required.

The electricity supply agreement caters for a ramp up period. From an initial supply of 15 MVA in 2023, the supply notified maximum demand (NMD) will be increased as required for the load build up. Eskom further indicated that a supply increase past 100 MVA is not seen as a problem as the Kusulie power station will be fully operational beyond 2023.

The construction of the OHL and substation is in progress with forecast completion and handover in the third quarter of 2023.

In addition to Eskom bulk power supply agreements and as part of the long-term sustainability plan, Ivanplats has entered into an expression of interest with an independent power producer. Ivanplats has indicated its interested in becoming an off-taker of up to 80 MW of renewable energy. The qualifying criteria would be that at inception, all energy sold to Ivanplats will be at prices below the Eskom Megaflex tariff structure. Recent promulgation of regulations allowing the operation of power plants up to 100 MW without a licence has provided an opportunity for Ivanplats to approach the market with a view to placing a Photo-Voltaic plant on the property. The plant as envisioned would be able to supplement but not replace the Eskom supply. The use of self generated renewable energy is expected to provide operational cost savings and other benefits such as a lower carbon footprint. With the rapid advancement of energy storage technology, it is envisaged that a future expansion of such a project will provide energy storage capacity and add significant flexibility in terms of energy usage during the peak tariff periods. This will contribute positively to the long-term sustainability of the Platreef Project.

1.16.3 Access Roads

Access from Mokopane to Johannesburg, Polokwane, and Rustenburg (for concentrate delivery) is via the newly upgraded N1 highway. The Platreef Project is located approximately 8 km north–north-east of Mokopane and is accessed via the N11, a single carriageway public highway with a bitumen surface.

The N11 highway connects Mokopane with the South Africa and Botswana border. The current road runs directly through the Turfspruit 241 KR and Macalacaskop 243 KR farms and serves the operating Mogalakwena mine. Accelerated mining developments and envisaged further expansions to the north of Mokopane have led to an increase in pressure on existing infrastructure in the area and specifically on the N11 at and through Mokopane. The N11 is also the only feasible road to and from the Platreef Project. Ivanplats has completed the N11 intersection construction works to the mine gate.

1.16.4 Tailings Storage Facility

The phased development plan provides for the development of two Tailings Storage Facilities; one for the initial production phase at the Platreef mine in close proximity to the plant and a larger one on the Rietfontein farm, located approximately 5 km from the Platreef mine site, for the steady state production.

The proposed tailings storage facility (TSF) site for the first phase is located approximately 1 km from the Platreef plant within the boundary fence of the mine site. This TSF has a total storage capacity of approximately 6 Mt of tailings and the tailings generated by the 700 ktpa concentrator plant will report and stored here, until the Rietfontein facility is commissioned.

The proposed tailings storage facility (TSF) site for the second phase is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed TSF site is considered a feasible site, considering all applicable engineering and environmental standards for tailings storage facilities. The TSF will have an operating life of 23 years, during this time approximately 56.1 Mt of tailings will be stored within the TSF. The remainder of the tailings will be used as backfill in the underground mine. The TSF is compliant in terms of the required tonnage profile production split between the backfill requirement and TSF storage. The estimated split has been quantified as 35% on average, however the tailings stream tonnage profile has been conservatively designed for 40% of non-ore material reporting to the TSF.

The Platreef 2022 FS proposed the use of a dry stack deposition methodology instead of the hybrid paddock deposition methodology proposed in the 2017 FS. The decision by Ivanplats to change the TSF deposition methodology from upstream design to dry stacking resulted from a study undertaken by Golder Associates in December 2016. It was concluded during this study that stacked tailings storage facilities are deemed to be safer in that there is no hydraulic depositioning, hence the risk will be minimal to flood the surrounding areas with tailings in the unlikely event of a catastrophic failure. Stacked tailings storage facilities are more water efficient in that the majority of water in the tailings is captured in the dewatering plant, pumped directly back to the concentrator and re-used back into the process.

The stacked facility will comprise a starter dam, constructed primarily of rockfill, engineered tailings, nominally compacted tailings, and random fill. Tailings will be delivered to the dewatering facility situated at the stacking facility utilising the same pumping systems from the processing plant. Dewatered tailings will be delivered to the stacking facility using load and haul transportation with trucks and conveyors from the dewatering plant.

Aside from the rock fill in the starter dam and drainage infrastructure, which includes a return water dam, the facility will be developed and operated using dewatered tailings as a dry stack facility. A liner system with leak detection will have to be in place upon start-up.

Progressive rehabilitation of the exterior slopes of the facility will be achieved through the placement of a layer of rockfill for erosion protection. An allowance for the placement of an organics layer for revegetation has been made within the design specifications. Dust control, comprising watering and the placement of dust suppressants will likely be required throughout operations.

During the initial phase of the project the dewatering of the tailings will be generated in the 700 ktpa plant. This dewatered and filtered tailings stream will be hauled and placed on the initial dry stack, tailings storage facility on the Platreef property. The first phase dry stack facility will be developed on the initial and approved waste rock dump footprint within the immediate Platreef mine. Golder has conducted the design, to inform the applicable license and permitting amendment processes, which are currently underway.

1.17 Market Studies and Contracts

The Platreef production is scheduled to come on stream at a time when the world will require additional PGM production to meet what many observers predict to be significant supply and demand deficits. Favourable positioning on the cost curve, base metal diversification and a natural South African Rand (ZAR) hedge should all conspire to make the concentrate attractive to South African toll smelters.

Discussions between Ivanplats and interested parties are underway, and Ivanplats expects this to result in a definitive concentrate sale or tolling agreement.

Ivanplats has prepared a number of marketing studies historically and maintains relationships with key smelters in South African PGM space. Ivanplats has a clearly defined development strategy to secure smelting and refining capacity in South Africa and has mapped a development path for placing concentrates and expects capacity to become available by the time that steady state production is achieved.

With the establishment of a number of smaller PGM mining firms, toll smelting and refining contracts and concentrate purchase agreements have become more prevalent in South Africa than in the past. The main PGM mining companies have some internal purchase contracts with their own mining and concentrating operations and external purchasing or toll contracts with independent or joint venture (JV) companies. Within the industry and along the value chain there are various possibilities for metal sales contracts: concentrates, furnace and converter mattes, Ni by-products, PGM residues or concentrates have all been sold or toll treated in the past.

PGM concentrate is sold within South Africa and into Europe under long-term contracts. The three major PGM producers have a full suite of process facilities to produce final PGM metal and hence tend to be purchasers rather than sellers of any PGM containing materials. Other PGM producers produce various intermediate products across the value chain ranging from flotation concentrate to high-grade PGE residue and nickel sulfate. The vast majority of these products are refined in South Africa, but some high-grade PGM residues are shipped overseas for final processing in Germany.

1.18 Environmental Studies, Permitting, Social, and Community Impact

In 2013, Ivanplats undertook the Environmental and Social Impact Assessment (ESIA) and Environmental Management Programme (EMP) for the Platreef Project in support of a Mining Right, Environmental Authorisation (EA) and Waste Management Licence (WML) application. Since the approval of these applications, Digby Wells has provided ongoing environmental advisory support on the Platreef Project and undertaken further regulatory applications, including an ESIA and EMP Addendum (2016) in support of proposed amendments to the approved EA.

The key environmental and social licences and permits submitted for the Platreef Project are:

- Mining Right,
- Environmental Authorisation,
- Integrated Waste Management Licence,

- Water Use Licence, and
- Heritage Permits.

Ivanplats is engaged with the amendment of some of the existing authorisations and licences to facilitate the aspects of the Platreef Project such as the:

- Changes to the infrastructure plan, such as the inclusion of the Masodi WWTW pipeline, change of the Waste Rock Dump to a dry stack tailing facility and the change of the upstream TSF to a dry stacking tailings facility. Additional specialist studies and public participation have been undertaken to inform the applications for these amendments.
- The proposed backfilling of residue material into the mine voids. This operation does require a full ESIA process which will need to be granted prior to commencing with the activity of backfilling.

The possible future applications that Ivanplats may need to undertake based on the nature of the Platreef Project and/or amendments of approved activities include the following:

- Ivanplats is considering options regarding the construction and operation of a Solar Photovoltaic Plant. Power generation that exceeds 10 MW requires environmental approval and triggers a Basic Assessment Process.
- Further amendments to the infrastructure plan, such as the re-alignment of the TSF pipeline route, will require amendment application processes to be undertaken on the EA.

Various Public Participation Processes (PPPs) have been undertaken for the Platreef Project from the initial ESIA process and the subsequent EA amendment processes. Comments and issues raised by stakeholders were incorporated in a Comments and Response Report (CRR) as part of all ESIA processes undertaken. The key issues and concerns which were raised during the various PPP included:

- Impact of the Platreef Project on both ground and surface water (reduction in water quality and quantity).
- The increase in dust due to mining activities such as hauling on dirt roads and dust from tailings storage facilities.
- Potential damage to houses and infrastructure of surrounding communities as a result of blasting.
- Mistrust in Ivanplats management.
- Unmet expectations regarding benefits from the mine to the community.
- Surface lease agreements and negotiations.
- Enquiries as to how the mine will benefit people and communities.
- Employment of unskilled labour, disabled, women and local persons as first priority.
- Skills training and requirements for employment.
- The absence of government representation and traditional leadership at meetings.
- Following protocol before Public Meetings.

- Additional meetings for stakeholders who live in town and on farms.

The key environmental and social sensitivities that have been identified for the Platreef Project are:

- Surface Water
- Groundwater
- Wetlands
- Cultural Heritage
- Communities
- Noise
- Visual
- Dust

The Platreef Project will contribute to the local economy through both direct and indirect employment opportunities and will result in a substantial injection of cash into the local economy of the Mogalakwena local municipal area. In addition, there will be an increase in opportunities for local suppliers of goods and services to the operation. In general, the socio-economic conditions in the area will be uplifted through better infrastructure, Local Economic Development (LED) projects, Enterprise Development (ED), Broad-Based Black Economic Empowerment (B-BBEE) ownership and projects and other company Corporate Social Responsibility (CSR) initiatives.

The development of entrepreneurs is one of the most effective ways of stimulating economic growth, transformation and the creation of jobs in the communities. Ivanplats' Economic and Enterprise Development function was established to ensure focused and integrated delivery of programmes aimed at contributing to the socio-economic development of the communities and the small, medium and micro enterprise (SMME) sector.

Development will focus on sustainability and job creation. Enterprise and supplier development aims to nurture, grow and sustain SMMEs by providing technical and business development support, through mentoring and coaching. In addition, loan funding will be provided to SMME suppliers through the Ivanplats Lefa Trust ("Lefa Trust"). Economic inclusion and development will be a joint effort between Ivanplats, contracting companies, preferred suppliers and government agencies.

The Local Economic Development (LED) projects in the Social and Labour Plan (SLP), as well as some additional projects, aim in the first five years to construct infrastructure at strategic points in the host villages and to provide appropriate support and training in an effort to make these projects sustainable. The infrastructure addresses urgent issues such as sanitation, a need for educational facilities, a need for pre-school facilities and access to a variety of services including information services, social services, financial services, training and entrepreneurial development. The second five-year plan will change its focus somewhat to include education infrastructure, support of health facilities and critical municipal infrastructure projects.

The potential impacts associated with the Platreef Project, including their pre-mitigation and post-mitigation significance, as well as mitigation management measures were identified. A monitoring programme has been developed to monitor various environmental aspects associated with the Platreef Project. The main potential impacts associated with the Platreef Project include, but are not limited to:

- Increased sediment and salts reporting to the drainage channels and streams from the mine site.
- Increased fugitive dust generation.
- Loss of flora and fauna Species of Special Concern (SSC).
- Soil erosion and soil compaction.
- Increased surface water run-off resulting in decreased infiltration which will affect downstream users.
- Deteriorating surface water quality.
- Dewatering in upper aquifer resulting in negative groundwater quantity impacts.
- Groundwater quality impacts as a result of seepage from TSF, waste rock dumps, stockpile areas and hydrocarbon spills.
- Construction activities causing potential disturbances in wetlands will result in the loss of ecological services in these areas.
- Negative visual impacts due to site clearance and construction of noticeable infrastructure.
- Physical changes to burial grounds and graves due to site clearing.
- Noise impacts emanating from machinery and vehicles.

The findings of the ESIA and subsequent assessments undertaken have shown that the Platreef Project may result in certain negative impacts to the environment; however, adequate mitigation measures have been included into the EMP Report to reduce the significance of all the identified negative impacts. Most negative impacts (minor and moderate) can be reduced through the implementation of mitigation and management measures.

The main potential social impacts associated with the Platreef 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. Further to this, there are social risks due to the social environment under which the Platreef 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. The various stakeholder engagement processes revealed a reoccurrence of issues raised by stakeholders regarding the Platreef Project. Stakeholder engagement is an ongoing process and a grievance mechanism has been developed to manage stakeholder concerns.

Continuous monitoring according to the EMP will be undertaken throughout the Life-of-Mine (LOM) to ensure correct implementation of the mitigation measures. Furthermore, internal and external audits of compliance to the EA, WML and WUL conditions will be undertaken in accordance with the authorisations and submitted to the relevant authorities.

1.19 Platreef 2022 Feasibility Study

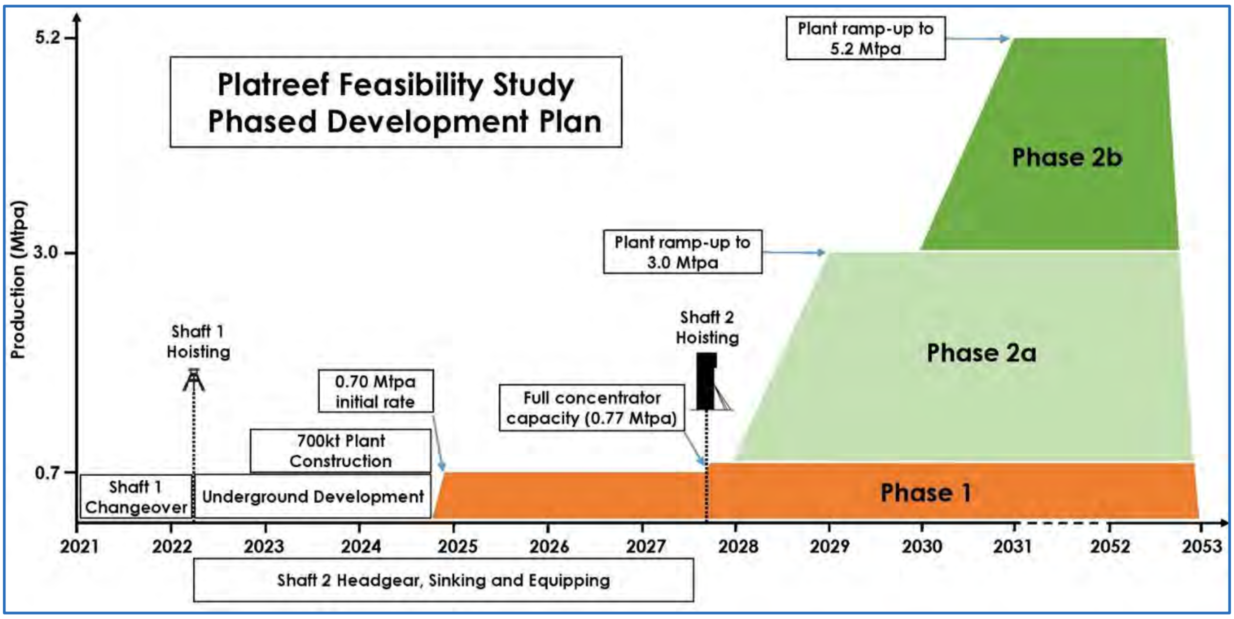
1.19.1 Platreef 2022 FS Overview

The Platreef 2022 FS is a Phased Development Plan based on continued development and earlier production from Shaft 1 which starts with 700 ktpa production (2024–2027) and then two 2.2 Mtpa concentrator streams will be added in 2028 and 2030, increasing the production rate to 5.2 Mtpa. The Platreef 2022 FS describes a change in production rate for the project that will require separate capital costs and infrastructure.

Compared to the Platreef 2017 FS, the tailings storage method has been modified to a dry-stack tailings facility – a sustainable and water-efficient method wherein tailings are placed and compacted in a mound that is concurrently reclaimed with soil and vegetation during the initial development phase.

The Platreef 2022 FS development and production timeline schematic is shown in Figure 1.17.

Figure 1.17 Platreef 2022 FS Development and Production Timeline



OreWin, 2021

The Platreef 2022 FS has an average annual production rate of 522,379 oz of platinum, palladium, rhodium and gold (3PE+Au), plus 23 Mlb of nickel and 14 Mlb of copper, at a cash cost of $514/oz of 3PE+Au, net of by-products, and including expansion and sustaining capital costs.

The Platreef 2022 FS evaluates a phased development of Platreef, with an initial 700 ktpa underground mine and a 770 ktpa capacity concentrator, targeting high-grade mining areas close to Shaft 1, with a significantly lower initial capital cost of $488M. With the sinking of Shaft 2 recommencing in 2023, first concentrate production is targeted in 2024 to coincide with the construction of two 2.2 Mtpa concentrators to be completed in 2028 and 2030. This would increase the steady production to 5.2 Mtpa by using Shaft 2 as the primary production shaft. The Platreef 2022 FS describes a change in production rate for the project that will require separate capital costs and infrastructure.

Initial capital cost of $488M and expansion capital expenditure of 1.48 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of $1.69 billion and an internal rate of return (IRR) of 18.5%.

Key steps involved in preparing the Platreef 2022 FS are as follows:

- Shaft 1 changeover completed for permanent hoisting in February 2022.
- Start development from the bottom of Shaft 1 in April 2022.
- Shaft 1 capacity is limited to ~700 ktpa ore, plus waste development.
- Initial development focus from Shaft 1 is a ventilation raise, completed by February 2024.
- Reduced initial development, focusing on the nearest, highest-grade stopes.
- Shaft 2 sinking recommences in September 2023. This is a discrete decision, and can be started at any point in time, depending on funding.
- Base case is a 770 ktpa concentrator on site.
- Assumes dry stacked tailings dam (for on-site concentrator).

The key dates for the Platreef 2022 FS are summarised in Table 1.5.

Table 1.5 Platreef 2022 FS Key Dates

Activity Name	Start	Finish
Shaft 1 commissioning		Q1'22
Restart Development from Shaft 1	Q2'22	
Ventilation Raise 1 (750 m Level to Surface)	Q1'23	Q4'23
Ventilation Raise 1 (950 m Level to 750 m Level)	Q4'23	Q1'24
Shaft 2 Sinking to –60 m Level	Q3'23	Q4'24
First Concentrator	Q3'24	
Shaft 2 Sinking to –114 m Level	Q4'24	Q4'24
Shaft 2 Sinking to –750 m Level	Q4'24	Q4'25
Shaft 2 Sinking to –850 m Level	Q4'25	Q1'26
Shaft 2 Sinking to –950 m Level	Q1'26	Q1'26
Shaft 2 Sinking to –1,050 m Level	Q1'26	Q2'26
Shaft 2 Sinking to –1,100 m Level	Q2'26	Q3'26
Shaft 2 Equipping Complete		Q3'27
Start of mining ramp up for first 2.2 Mtpa concentrator	Q1'28	
Start of mining ramp up for second 2.2 Mtpa concentrator	Q1'30	
Mine Production Steady State (5.2 Mtpa)	Q4'30	

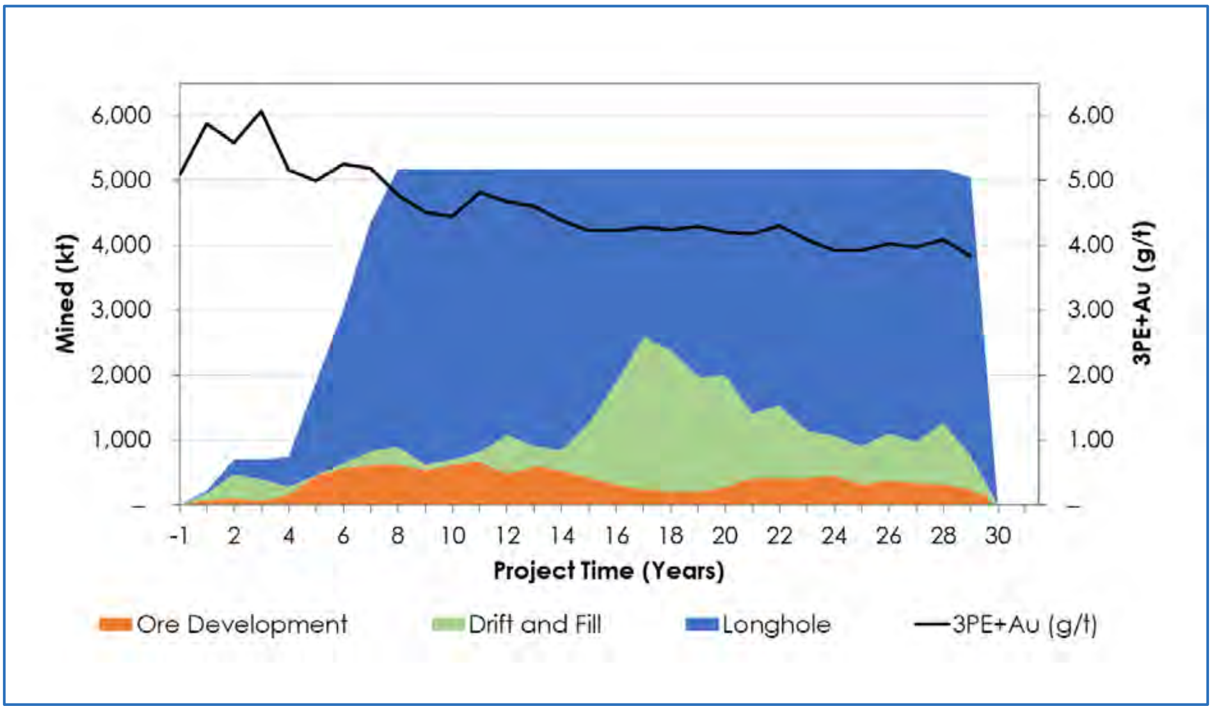
1.19.2 Platreef 2022 FS Production Summary

The production plan for the Platreef 2022 FS focused on maximising higher-grade areas. The ore body was targeted to recover approximately 125 Mt at the highest NSR. This resulted in using declining cut-off grades and decreasing the NSR cut-off from $155/t–$80/t NSR.

A further focus on optimising net present value targeted the higher-grade stopes in the early years. An optimisation was performed based on stope locations, stope grades, mining method, and zone productivities. Through this analysis, the grades in the first 10 years had a significantly higher 3PE+Au.

The mine production forecast is shown in Figure 1.18 and the key average annual production results over the planned life of the mine are shown in Table 1.6. Concentrator feed and estimated concentrator produced along with grades for the life of mine are depicted in Figure 1.19 and Figure 1.20.

Figure 1.18 Platreef 2022 FS Mining Production



OreWin, 2021

Table 1.6 Platreef 2022 FS Average Annual Production Forecast

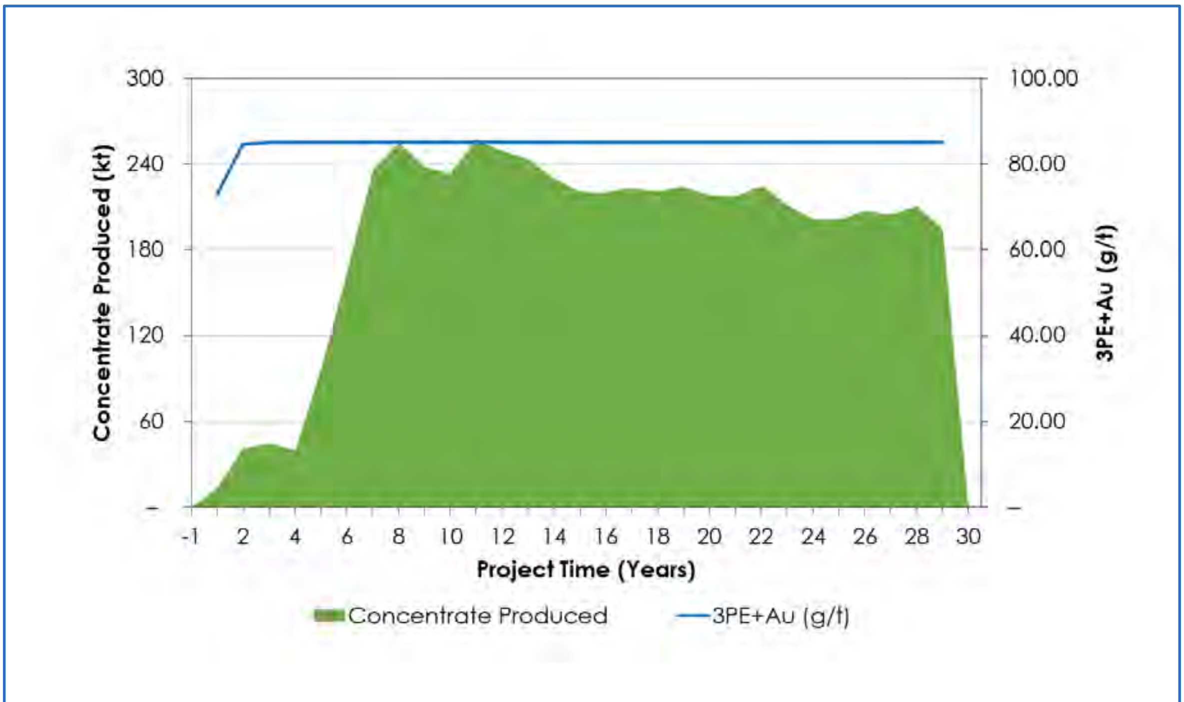
Item	Unit	LOM Average
Mined and Processed	Mtpa	4.32
Platinum	g/t	1.94
Palladium	g/t	1.99
Gold	g/t	0.30
Rhodium	g/t	0.13
3PE+Au	g/t	4.37
Copper	% Cu	0.16
Nickel	% Ni	0.34
Concentrator Recoveries		
Platinum	%	87.2
Palladium	%	86.8
Gold	%	78.5
Rhodium	%	80.3
Copper	%	87.7
Nickel	%	71.6
Concentrate Produced	ktpa	191
Platinum	g/t	38.2
Palladium	g/t	39.0
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE+Au	g/t	85.0
Copper	% Cu	3.3
Nickel	% Ni	5.4
Recovered Metal		
Platinum	kozpa	235
Palladium	kozpa	240
Gold	kozpa	33
Rhodium	kozpa	15
3PE+Au	kozpa	522
Copper	Mlbpa	14
Nickel	Mlbpa	23

1. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.
2. Production over 29 years.

Figure 1.19 Platreef 2022 FS Concentrator Production

OreWin, 2021

Figure 1.20 Platreef 2022 FS Estimated Concentrate Produced and 3PE+Au Grade



OreWin, 2021

1.19.3 Platreef 2022 FS Capital and Operating Cost Summary

The total initial (pre-production) expansion and sustaining capital costs required, including contingency, sourced from the Platreef 2022 FS, are shown in Table 1.7.

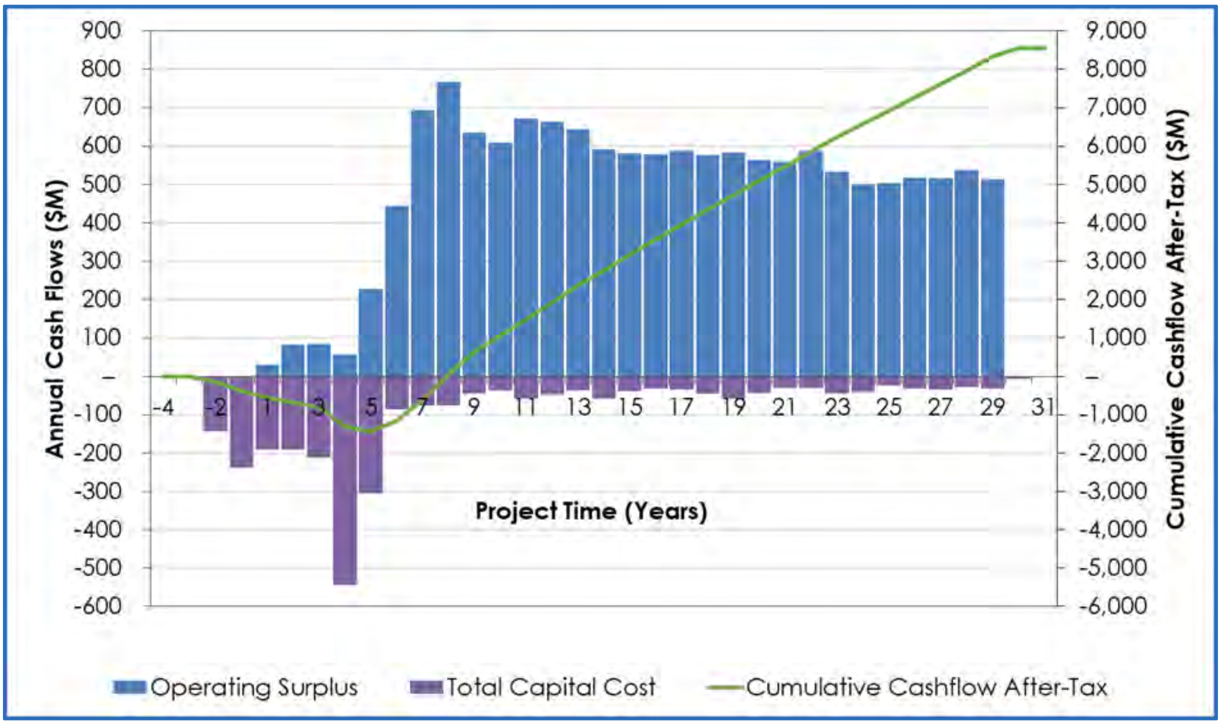
Table 1.7 Platreef 2022 FS Total Projection Capital Cost

Description	Initial ($M)	Expansion ($M)	Sustaining ($M)	Total ($M)
Mining				
Geology	9	31	32	72
Mining	187	697	861	1,744
Capitalised Operating Costs	–	–	–	–
Subtotal	195	728	893	1,816
Concentrator and Tailings				
Concentrator	73	273	2	349
Capitalised Operating Costs	–	–	–	–
Subtotal	73	273	2	349
Infrastructure				
Infrastructure	87	251	25	363
Site Costs	7	0	0	7
Capitalised Operating Costs	–	–	–	–
Subtotal	95	251	25	371
Owners Cost				
Owners Cost	93	126	2	222
Closure	–	–	11	11
Capitalised Operating Costs	–	–	–	–
Subtotal	93	126	13	233
Capex Before Contingency	456	1,378	933	2,768
Contingency	32	101	1	134
Capex After Contingency	488	1,480	934	2,902

1. Initial Capital for the preproduction time including $50M in Shaft 2.
2. Totals vary due to rounding.

The Net Cash Flow After-Tax and the Cumulative Cash Flow After-Tax is shown in Figure 1.21.

Figure 1.21 Platreef 2022 FS Cumulative Cash Flow After-Tax



OreWin, 2022

Mine site cash costs are summarised in Table 1.8. The revenues and operating costs are presented in Table 1.9 along with the net sales revenue value attributable to each key period of operation.

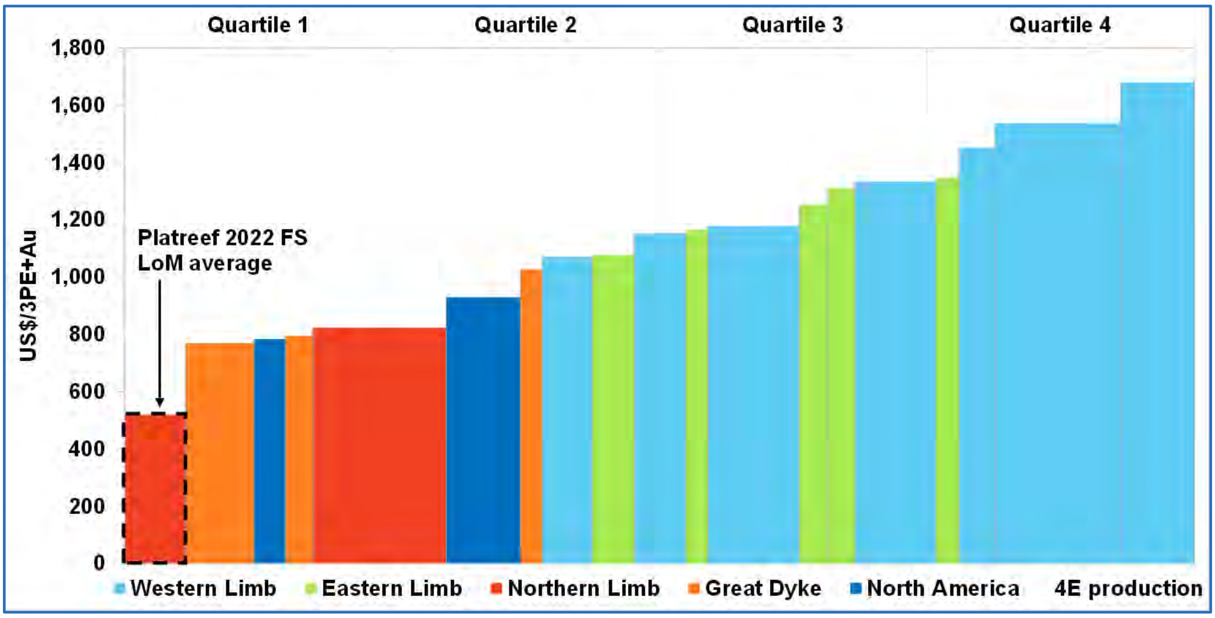
Higher nickel and copper grades contribute to lower cash costs for operations on the Northern Limb as illustrated by Figure 1.22. Among global primary platinum-group-metals producers, Platreef's estimated net total cash cost of $514/oz 3PE+Au, net of copper and nickel by-product credits and including stay-in-business (SIB) capital costs, ranks at the bottom of the cash-cost curve.

Table 1.8 Platreef 2022 FS Cash Costs After Credits

	$/oz Recovered 3PE+Au		
	Years 1–4 Average	Years 5–29 Average	LOM Average
Mine Site Cash Costs	822	419	429
Transport	13	13	13
Treatment & Refining	369	366	366
Royalties	8	90	88
Total Cash Costs Before Credits	1,212	887	895
Nickel Credits	334	351	351
Copper Credits	84	92	92
Total Cash Costs After Credits	794	443	452
Sustaining Capital Costs	–	63	62
Total Cash Costs After Credits & Sustaining Capital	794	506	514

Totals may vary due to rounding.

Figure 1.22 Global Primary Producers' Net Total Cash Cost + Sustaining Capital (2021E) US$/3PE+Au oz



SFA (Oxford), Ivanplats. Notes: Cost and production data for the Platreef project is based on the Platreef 2022 FS parameters, applying payabilities and smelting and refining charges as agreed with purchase of concentrate partners for Platreef concentrate (this is not representative of SFA's standard methodology). SFA's peer group cost and production data follows a methodology to provide a level playing field for smelting and refining costs on a pro-rata basis from the producer processing entity. Net total cash costs have been calculated using Ivanplats' long term price assumptions of 16:1 ZAR:USD, US$1,100/oz platinum, US$1,450/oz palladium, US$5,000/oz rhodium, US$1,600/oz gold, US$8.00/lb nickel and US$3.50/lb copper.

Table 1.9 Platreef 2022 FS Operating Costs and Revenues

Description	LOM Total ($M)	($/t) Milled		
		Years 1–4 Average	Years 5–29 Average	LOM Average
Gross Sales Revenue	28,002	305.48	222.09	223.64
Less: Realisation Costs				
Transport	195	2.12	1.55	1.56
Treatment and Refining	5,540	59.76	43.95	44.25
Royalties	1,337	1.23	10.85	10.67
Total Realisation Costs	7,072	63.10	56.36	56.48
Net Sales Revenue	20,930	242.37	165.74	167.16
Site Operating Costs				
Mining	4,005	72.12	31.22	31.98
Processing and Tailings	1,593	21.38	12.56	12.72
Infrastructure	289	9.44	2.17	2.30
Site Cost	160	3.94	1.23	1.28
General and Administration	447	26.11	3.15	3.57
Escalation and Contingency	–	–	–	–
Total	6,493	132.99	50.32	51.86
Operating Margin	14,437	109.38	115.41	115.30
Operating Margin	52%	36%	52%	52%

Totals may vary due to rounding.

1.20 Model Assumptions

1.20.1 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2022. It is presented in 2022 constant dollars, cash flows are assumed to occur evenly during each year, and a mid year discounting approach is taken. The base case real discount factor applied to the analyses is 8%. No allowance for inflation has been made in the analyses.

The prices in the economic analysis for the Platreef 2022 FS are based on a review of consensus price forecasts from financial institutions and similar studies that have recently been published. The commodity price assumptions for the Platreef 2022 FS are shown in Table 1.10.

Table 1.10 Platreef 2022 FS Commodity Price Assumptions

Parameter	Unit	Financial Analysis Assumptions
Platinum	$/oz	1,100
Palladium	$/oz	1,450
Gold	$/oz	1,600
Rhodium	$/oz	5,000
Copper	$/lb	3.50
Nickel	$/lb	8.00

1.20.2 Treatment Charges and Refining Charges

In the Platreef 2022 FS, payables have been assumed on the basis of the two offtake arrangements and expectations for the life-of-mine concentrate production. Refining charges are shown in the Table 1.11.

Table 1.11 Platreef 2022 FS Refining Charges (%Gross Sales)

	LOM
Copper	27.0%
Nickel	30.0%
Platinum	16.5%
Palladium	16.5%
Gold	16.5%
Rhodium	17.5%

1.20.3 Concentrate Transport Costs

In the Platreef 2022 FS, concentrate transport cost based on the distance of 270 km is ZAR 1.23/t/km and the average distance of the smelters is 277 km. The transport cost applied to the financial model is $0.08 per wet tonne concentrate per km.

1.20.4 Royalties and Taxes

The majority of taxes and fees payable to the government under Republic of South Africa legislation are the Corporate Income Tax (28%) and a production royalty. The royalty rate for refined minerals is a percentage determined as per Section 4 of the Republic of South Africa Royalty Act 28 (2008; Government Gazette No. 31635), and the Mineral and Petroleum Resources Royalty (Administration) Act No. 29 (2008; Government Gazette No. 31642).

Royalty % = 0.5 + [EBIT/ (Gross Sales x 9)] x 100, with a maximum of 7%, for production of unrefined minerals.

Assumptions for the royalties and taxes are shown in Table 1.12.

Table 1.12 Platreef 2022 FS Royalties and Taxes

Royalties		
Base Factor	%	0.50
Unrefined Mineral Factor		9.00
Pct Factor Not to be Exceeded	%	7.00
Taxes		
Corporate Income Tax Rate	%	28.00
Opening Tax Losses	millionZAR	305
Opening Depreciation	millionZAR	7,468
Working Capital		
Receivables	weeks	15.00
Payables	weeks	4.00

1.20.5 Exchange Rates

Costs estimated in ZAR have been converted to US dollars at an exchange rate of 16 ZAR/USD. A comparison between the exchange rates used in the Platreef 2022 FS and the 2017 FS is shown in Table 1.13.

Table 1.13 Exchange Rates

Exchange Rates	Forex Amount	
	2022 FS	2017 FS
ZAR	16.00	13.00
EUR	0.71	0.79
AUD	1.18	1.18
CNY	5.70	5.91
GBP	0.63	0.67
JPY	100.00	92.86
NOK	7.00	7.39
SEK	7.00	7.56

1.21 Project Results

The results of the financial analysis show an After-Tax NPV8% of $1,690M. The Platreef 2022 FS exhibits an after-tax IRR of 18.5% and a payback period of approximately eight years. The estimates of cash flows have been prepared on a real basis as at 1 January 2022 and a mid year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 1.14.

Table 1.14 Platreef 2022 FS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	11,535	8,543
	5.0%	4,242	3,098
	8.0%	2,369	1,690
	10.0%	1,594	1,104
	12.0%	1,051	692
	15.0%	513	283
	20.0%	33	-83
Internal Rate of Return	–	20.54%	18.48%
Project Payback Period (Years)	–	7.93	7.93

1.21.1 Summary of Platreef 2022 FS

The key features of the Platreef 2022 FS included:

- Development of a large, mechanised, underground mine is planned at an initial 700 ktpa and expansion to 5.2 Mtpa.
- Planned average annual production rate of 522 koz of platinum, palladium, rhodium and gold (3PE+Au).
- Estimated pre-production capital requirement of approximately $488M, including contingencies.
- After-tax NPV of $1,690M, at an 8% discount rate.
- After-tax IRR of 18.48%.
- The Platreef 2022 FS maintains options available to accelerate expansions, to the 8 Mtpa or the 12 Mtpa scenarios, as the market dictates.

A summary of the key project physical and financial metrics is shown in Table 1.15.

Table 1.15 Platreef 2022 FS Key Production and Financial Results

Item	Unit	Total
Mined and Processed	Mt	125
Platinum	g/t	1.94
Palladium	g/t	1.99
Gold	g/t	0.30
Rhodium	g/t	0.13
3PE+Au	g/t	4.37
Copper	% Cu	0.16
Nickel	% Ni	0.34
Concentrate Produced	kt	5,545
Platinum	g/t	38.2
Palladium	g/t	39.0
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE+Au	g/t	85.0
Copper	% Cu	3.3
Nickel	% Ni	5.4
Recovered Metal		
Platinum	koz	6,813
Palladium	koz	6,954
Gold	koz	948
Rhodium	koz	433
3PE+Au	koz	15,149
Copper	Mlb	399
Nickel	Mlb	665
Key Financial Results		
Life-of-Mine	Years	29
Initial (Pre-Production) Capital	$M	488
Expansion Capital	$M	1,480
Sustaining Capital	$M	934
Mine-Site Cash Cost	$/oz Rec. 3PE+Au	429
Total Cash Costs After Credits	$/oz Rec. 3PE+Au	452
Site Operating Costs	$/t Milled	52
After-Tax NPV8%	$M	1,690
After-Tax IRR	%	18.48
Project Payback Period	Years	7.93

1. Initial Capital including $50M in Shaft 2 and $32M in contingencies.
2. Totals may not add due to rounding.
3. 3PE+Au = platinum, palladium, rhodium and gold.
4. Economic analysis metal price assumptions: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

1.22 Conclusions

1.22.1 Platreef 2022 Feasibility Study

The Platreef 2022 FS is the current development scenario for the Platreef project. It has advanced the development plan for the Platreef Project and increased the confidence in the Mineral Reserve to a feasibility study level of accuracy. This will provide the technical basis for Ivanplats to continue the project financing and to continue marketing negotiations that have been undertaken to date. The Platreef 2022 FS provides the basis for detailed planning of the project execution and to update the long-term development plans for Platreef.

1.22.2 Geology and Mineral Resource Estimates

Mr Kuhl is of the opinion that the Mineral Resources for the Platreef Project, which have been estimated using diamond core drillhole data, have been performed to industry best practices (CIM, 2003), and conform to the requirements of the 2014 CIM Definition Standards.

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction including:
  - Long-term commodity price assumptions.
  - Long-term exchange rate assumptions.
  - Assumed mining method.
  - Availability of water and power.
  - Operating and capital cost assumptions.
  - Metal recovery assumptions.
  - Concentrate grade and smelting and refining terms.
- Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. The current practice of using grade shells in the area drilled in detail may underestimate the variability of the grades within and in the vicinity of the T1MZ and the T2MZ. Stope boundaries that are laid out along the 2 g/t 2PE+AU grade shell surface will likely not, in practice, be able to follow the exact actual surface. The consequence would be that the effects of contact dilution and ore loss could be more than is currently projected.
- The continuity of FW mineralisation has been modelled based on limited drill data, as not all of the UMT drillholes extended into the FW. For this reason, estimation of Mineral Resources has been restricted to the north-western area of the Platreef Project where drill spacing is in the order of 100–200 m. Similar mineralisation has been seen in drillholes across the entire Platreef Project, but the current drill spacing is insufficient to define Mineral Resources amenable to selective mining methods in these areas.

This represents exploration upside for the Platreef Project. Drill intercepts ≥2.0 g/t 3PE+Au in the FW domains are narrow and suggest selective mining would be required. Grade continuity is best observed at a 1.0–1.5 g/t 3PE+Au cut-off. Discontinuous pods of mineralisation at a 2.0 g/t 3PE+Au cut-off are present but are not well defined at the current drill spacing, and additional drilling is required. The FW cpx domain includes thicker zones of low-grade mineralisation that may permit mass mining methods at a lower cut-off (1 g/t 3PE+Au).

1.22.3 Mining

The mine plan and expenditure schedule presented herein is reasonable. The plan is based on Platreef 2022 FS data and established mining practice. The resource model and geotechnical parameters provided to OreWin are suitable for the design of a large-scale and highly mechanised underground mine at a feasibility-level of confidence.

The proposed plan uses well-established mining technology. No unproven equipment or methods are contained in the plan; however, there is potential to take advantage of currently available and future technology gains.

1.22.4 Metallurgy

The metallurgical testwork programme has yielded sufficient information to develop a definitive metallurgical flow sheet, and the testwork concluded is considered to be adequate. Detailed mineralogical analysis of selected ores, and the tailing samples have contributed to the understanding of the mode of occurrence and liberation characteristics of the valuable minerals.

The testwork programmes have been conducted by parties well versed in the processing of ores from the Bushveld Igneous Complex. The necessary checks and balances have been applied to ensure that the testwork and chemical analysis has been conducted with the necessary diligence and accuracy. The selection of samples, done in conjunction with the mining and geological teams, submitted for the metallurgical testwork for the purposes of the Platreef 2017 FS, and used in Platreef 2022 FS, are deemed to be sufficient.

Open circuit bench scale flotation test work has indicated that filtered site water, expected to be representative of the expected water quality from the Masodi supply system, results in a similar metallurgical performance per Mintek test work using municipal drinking water.

A mini pilot plant campaign was conducted, primarily, to produce bulk concentrate samples for downstream hydrometallurgical refining test work and concentrate de-watering test work. The added objective of deriving additional design data from the pilot runs was only partially achieved due to a number of operational challenges at Mintek. These runs and are thus considered to reflect preliminary commissioning results. These commissioning runs, successfully, allowed for generation of concentrate samples for Kell test work and concentrate de-watering test work but did not provide comprehensive data to fully confirm the metallurgical performance as achieved in bench scale locked cycle test work. The locked cycle test results as derived during the 2017 FS are considered adequate for deriving metallurgical performance projections.

The proposed circuit is considered to be the preferred option for the concentrator. The use of a multi-stage crusher circuit followed by a single stage milling circuit is considered to be the option of least risk to the project. Preliminary assessments have indicated that the inclusion of an HPGR circuit as an alternative to the tertiary crushing circuit for the Phase 2, 4.4 Mtpa concentrator could potentially provide additional operating cost saving opportunities.

The proposed flotation circuit is based on interpretation of the results obtained from the bench scale flotation testwork. The design and specification of the various flotation stages is considered adequate for the level of study and flexibility required during commercial production. The proposed phased concentrator capacity ramp-up and modular approach used for Phase 2 is considered to be appropriate for this level of study.

1.22.5 Infrastructure

A number of mining projects are in the development phase on the BIC that all require water, power and road access. This will place significant strain on the existing infrastructure, as well as further pressure on the approval and/or completion of major infrastructure projects.

The project team has addressed the supply-demand requirements of bulk power and water to a sufficient level of detail for this study.

Bulk water availability seems to be sufficient based on the level of accuracy of the study performed, however the timing of when the water will be available is of concern, and the delivery of the alternative bulk water supply options must not be taken for granted.

Substantive engagement with Eskom has resulted in the commencement of the construction of the overhead lines to supply bulk power to the mine. The supply application has been revised a to a NMD of 100 MVA.

The availability of skilled labour resources for both construction and operational phases of the Platreef Project is limited, and the training programme will have to be closely monitored to ensure that the correct skills development is done at the correct time, depending on the phase of the project.

1.23 Recommendations

1.23.1 Platreef 2022 Feasibility Study

The Platreef 2022 FS will provide the technical basis for Ivanplats to continue the project financing and to continue marketing negotiations that have been undertaken to date. Ivanplats should continue to prepare for the execution activities and to update the long-term development plans for Platreef. Continued development of Shaft 1 will progress the project and this can be used for further defining the execution plans.

The Platreef 2022 FS describes a phased development plan with a different initial production rate of 700 ktpa and then an expansion to 5.2 Mtpa.

1.23.2 Delineation Drilling

Ivanhoe has proposed stope positions and stope boundaries that should be delineated in more detail as underground development approaches mineralisation. The drill spacing should be approximately 25 m for the first 11 years of mining and 50 m thereafter. Holes would intersect the reef at angles >30°, and their maximum length would be limited to 250 m.

1.23.3 Geology and Mineral Resource Estimates

Ivanhoe plans to focus on the development of the Platreef underground mine, and no additional drilling is expected within the next few years. MTS recommends the FW mineralisation be further evaluated, but priority should be given to delineation of the TCU to support underground mining.

The data on positions of grade shell boundaries should be examined to the extent possible to estimate their short-scale variability; the likely accuracies of down-hole surveys should be taken into account, and it is recognised a definitive answer may have to await exposures in underground workings.

1.23.4 Mining Recommendations

The mining plan is reasonable based on the current block model with no additional drilling expected within the next few years no additional work is required. Delineation drilling to be completed as prioritised in the current schedule to better define the contacts for proper infrastructure placement.

1.23.5 Metallurgical

The metallurgical testwork programme has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes.

Preliminary open circuit bench scale flotation test work to evaluate the potential inclusion of Jameson cells in the cleaner flotation circuit showed reduced metallurgical performance but were deemed inconclusive due to the use of an outdated test procedure. It is recommended that these tests are repeated using the updated vendor procedure in order to confirm these findings.

Preliminary mini pilot plant test work was conducted during the Platreef FS, however, the plant was not fully commissioned, stabilized and optimized. Additionally, the majority of the runs reflect commissioning runs on low grade samples with a 3PE+Au head grade of 2.9 to 3.8g/t. To further evaluate optimisation opportunities and confirm additional detail design parameters, additional pilot plant test work on high grade samples aligned to the early years of mining (> 5g/t 3PE+Au) is proposed as part of the project implementation phase.

The mini-pilot testwork included trials of an SIBX reagent suite with preliminary data indicating this to be a viable alternative to the copper collector reagent suite. Additional testwork should be conducted to confirm this result and the inclusion of an SIBX make-up and dosing system should be undertaken during project implementation.

Additional pilot scale column test work is recommended to confirm the concentrate upgrade potential in a column cell as aligned to the Platreef design flowsheet.

The potential for cost savings should be evaluated further during the project implementation phase as follows:

- The 0.77 Mtpa concentrate de-watering equipment, as sized based on the 2017 FS benchmarked flux information, is considered adequate for the required duty. It is however noted that there is the potential to reduce the size of the 0.77 Mtpa concentrate dewatering equipment based on the findings from concentrate dewatering test work conducted during the Platreef FS.
- The installation of a tailings vacuum disk filter circuit to replace the vacuum belt filter as currently allowed for in the 0.77 Mtpa concentrator design for the Platreef FS.
- The potential inclusion of an HPGR circuit as an alternative to the tertiary crushing circuit for the 4.4 Mtpa concentrator (Phase 2) should be evaluated further as part of the phased implementation programme.

1.23.6 Infrastructure

Regular interfacing with the project teams of Eskom, JWF, and SANRAL to understand the status of external infrastructure projects, directly affecting the Platreef Project, must be pursued.

Further investigations into alternative bulk water sources to continue and suitable memorandums of understanding to be negotiated with regards to already identified alternative water sources.

Continuing interfacing with Eskom to ensure alignment on the electrical supply requirements and load ramp up, especially in light proposals to make use of electrical mobile equipment underground and develop a supplementary power plant. Acceleration of the schedule may change the load ramp and Eskom will need to be apprised.

2 INTRODUCTION

Ivanhoe is a mineral exploration and development company, with a portfolio of properties located in Africa. The Ivanhoe strategy is to build a global, commodity-diversified mining and exploration company. Ivanhoe has focused on exploration within the Central African Copperbelt and the Bushveld Complex. In addition, Ivanhoe holds interests in prospective mineral properties in the Democratic Republic of the Congo (DRC), Gabon, and Australia. Ivanhoe currently has three key assets: (i) the Kamoa Project; (ii) the Platreef Project, and (iii) the Kipushi Project. In 2013 Ivanhoe changed its name from Ivanplats Ltd. to Ivanhoe Mines Ltd.

Ivanhoe holds a 64% interest in South African Mining Right LP30/5/2/2/1/10067MR, while a Japanese consortium (the Japanese Consortium), comprising Itochu Corporation (Itochu); Japan Oil, Gas and Metals National Corporation (JOGMEC); and Japan Gas Corporation (JGC), holds a 10% interest, and local communities, local entrepreneurs, and employees hold the remaining 26% as a result of the Broad-Based Black Economic Empowerment (B-BBEE) transaction (the B-BBEE Partners), implemented on 26 June 2014. The Japanese Consortium's interest in the Platreef Project was acquired in two tranches for a total investment of $290M.

For the purposes of the Platreef 2022 FS, the name Ivanhoe refers interchangeably to, Ivanhoe Mines Ltd., the predecessor company named Ivanhoe Nickel and Platinum Ltd, and to Ivanplats. Ivanplats was formerly named Platreef Resources and African Minerals.

2.1 Terms of Reference

The Platreef 2022 FS is an Independent Technical Report (the Report) for the wholly-owned Platreef nickel–copper–gold–platinum group element (PGE) project (the Platreef Project) located near Mokopane, in the Limpopo Province of the Republic of South Africa.

The Platreef 2022 FS has been prepared using the Canadian National Instrument (NI) 43-101 Standards of Disclosure for Mineral Projects.

The following companies have undertaken work in preparation of the Platreef 2022 FS:

- OreWin Pty Ltd (OreWin): Overall report preparation, underground development and production costs, and financial model, underground mine plan and Mineral Reserve
- Mine Technical Services (MTS): Geology and Mineral Resource estimation.
- SRK Consulting (South Africa) (Pty) Ltd. (SRK): Mine geotechnical recommendations.
- DRA Projects SA (Pty) Ltd (DRA): Process engineering and infrastructure.
- Golder Associates Africa (Pty) Ltd: Water use License Application in terms of section 21 of the National Water Act, 1998 (Act 36 of 1998)(NWA).

The Platreef 2022 FS uses metric units of measure. The currency used is Q4’21 United States dollars, unless otherwise mentioned.

2.2 Qualified Persons

The following people served as Qualified Persons (QPs) as defined in NI 43-101 Standards of Disclosure for Mineral Projects:

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for Sections 1.1, 1.2, 1.17, 1.22.1, 1.23.1; Section 2; Section 3; Section 4; Section 19; Section 21; Sections 25.1, 25.3 and 25.4; Sections 26.1, 26.4 and 26.5; and Section 27.
- Timothy Kuhl, SME Registered Member (1802300), employed by Mine Technical Services as a Principal Geologist, was responsible for: Sections 1.3 to 1.10, 1.22.2, 1.23.2 and 1.23.3; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8 to 10.10.1; Section 11; Section 12; Section 14; Section 25.2; Sections 26.2 to 26.3; and Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd. as Corporate Consultant, was responsible for: Section 1.12; Section 2; Section 3; Section 10.10.3; Section 16.1; and Section 27.
- Curtis Smith B. Eng. (Mining), MAusIMM (CP) (311458), employed by OreWin Pty Ltd as Principal Mining Engineer, was responsible for: Sections 1.11, 1.13, 1.19 to 1.21; 1.22.1, 1.22.3; 1.23.1, 1.23.4; Section 2; Section 3; Section 15; Sections 16.2 to 16.8; Section 18.12; Sections 21.1 to 21.4.2, 21.4.6 to 21.9; Section 22; Section 23; Section 24; Sections 25.1, 25.3 and 25.4; Sections 26.1, 26.4 and 26.5; and Section 27.
- Val Coetzee, B.Eng. (Chemical), M.Eng. (Mineral Economics), Senior Vice President - Process, DRA Projects (Pty) Ltd, was responsible for: Sections 1.14 to 1.15, 1.16.1 to 1.16.3, 1.22.4 to 1.22.5, 1.23.5 to 1.22.6; Section 2; Section 3; Sections 5; Sections 10.7, 10.10.2; Section 13; Section 17; Sections 18.1 to 18.4, 18.6 to 18.11, 18.13 to 18.15; Sections 21.4.3 to 21.4.5; Sections 25.5 to 25.6; Sections 26.6 to 26.7; and Section 27.
- Riaan Thysse, employed as Business Unit Lead by Golder Associates Africa (Pty) Ltd, B.Eng., Pr Eng ECSA, was responsible for Sections 1.16.4, 1.18; Section 2, Section 3, Sections 4.2.5, 4.7 to 4.10; Sections 18.5; Section 20; Section 26.8; and Section 27.

2.3 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Bernard Peters visited the property for two days in February 2010 and for one day in April 2010, on 8 November 2012, on 9 October 2014, on 17 April 2017 and on 3 April 2019. The site visits included briefings from Ivanhoe geology and exploration personnel, site inspections of potential areas for mining, plant and infrastructure, inspections of surface operations and underground 950 m Level, meeting with Ivanhoe's management, discussions with other QPs and review of the existing infrastructure and facilities in the local area around the Platreef Project site. Bernard Peters has also visited the Ivanhoe office in Sandton South Africa on several other occasions for meetings with Ivanhoe personnel and consultants working on the Platreef Project.

- Mr Timothy Kuhl visited the site from 26 March–9 April 2010, 19 July–3 August 2011, 25 January–3 February 2012, and again from 27 November–12 December 2012. Most recently, Mr. Kuhl was at site 13 May–25 June 2015 and 8 July–3 August 2015. During these trips, he audited drill data obtained since the 2007 database audit (DaSilva, 2007), obtained QA/QC data, field checked drill collars, and collected witness samples for check assays. He also inspected drill core, surface outcrops, and sample cutting and logging areas. Discussions were held with Ivanhoe's staff about project geology and mineralisation; geological interpretations were reviewed, and potential locations of major infrastructure were viewed.
- Mr William Joughin has visited the site for one day during 2011, 23–24 May 2013, 21–22 January 2015 and 9 June 2015 to inspect drill core and to plan the geotechnical investigations. He also visited the site on 27 September 2016 to assess the rock conditions and support in Shaft 1. Additional site visits were conducted by SRK staff for quality assurance of the Ivanplats geotechnical logging.
- Mr Curtis Smith visited the property on 3 April 2019. The site visits included briefings from Ivanhoe geology and exploration personnel, site inspections of potential areas for mining, plant and infrastructure, inspections of surface operations and underground 950 m Level, meeting with Ivanhoe's management, discussions with other QPs and review of the existing infrastructure and facilities in the local area around the Platreef Project site. Curtis Smith has also visited the Ivanhoe office in Sandton South Africa on several other occasions for meetings with Ivanhoe personnel and consultants working on the Platreef Project.
- Mr Val Coetzee visited the site during October 2014 for a general site inspection and visited the Mintek laboratory where the current metallurgical testwork is underway.
- Riaan Thysse has not visited the site.

2.4 Effective Dates

There are a number of effective dates for the information included in the Report, as follows:

Effective Date of Technical Report:	28 February 2022
Mineral Resource Amenable to Selective Underground Mining Methods:	28 January 2022
Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	28 January 2022
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	28 January 2022
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Validation of resource models as current using current economic inputs:	28 January 2022
Effective Date of Mineral Resources:	28 January 2022
Effective Date of Mineral Reserves:	26 January 2022

2.5 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of the Platreef 2022 FS were used to support preparation of the Report. Additional information was provided by Ivanhoe as supporting information for the QPs.

Supplemental information was also provided to the QPs by third-party consultants retained by Ivanhoe in their areas of expertise.

Other supporting information was sourced from Ivanhoe.

Metric units of measurement have been used in the Platreef 2022 FS except where noted, and currency is expressed in US dollars unless stated otherwise.

3 RELIANCE ON OTHER EXPERTS

3.1 Project Ownership, Mineral Tenure, Permits and Agreements

The legal status of the mineral tenure, ownership of the Project area, and underlying property agreements or permits has not been independently verified.

QPs Bernard Peters and Tim Kuhl have fully relied upon, and disclaim responsibility for, information derived from the report:

- Ivanplats 6 December 2020: Platreef 2020 Feasibility Study Property Description and Ownership Report.

This information is used in Section 1.2, 1.3.1 and Section 4 of the Platreef 2022 FS and in support of the Mineral Resource estimate in Section 14.

3.2 Legal, Political, Royalties and Taxes

The assumptions for legal, political, royalties and taxes have been provided by Ivanhoe and are based on the following:

- Ivanplats 6 December 2020: Platreef 2020 Feasibility Study Property Description and Ownership Report.
- KPMG memorandum: subject: Updated commentary on specific tax consequences applicable to an operating mine in the Republic of South Africa 10 May 2017.
- Platreef 4 Mtpa Feasibility Study, Section 15 –Title and Legal, DRA Report Number: DRA-J0283-STU-REP-909, Ivanplats Report Number: 1051-EV-00-209Ivanplats (Pty) Ltd, July 2017.
- Ivanplats Email: Fwd.: Platreef – tax assets, 26 June 2020.

QPs Bernard Peters and Tim Kuhl have fully relied upon, and disclaim responsibility for the assumptions and work relating to royalties and taxes presented in Sections 1, 4, and 22 and in support of the Mineral Resource estimate in Section 14.

3.3 Marketing

Ivanhoe Mines Ltd., provided the following document relating to marketing that has been used:

- Ivanhoe 1 December 2020: Platreef 2021 FS Marketing Report.

Bernard Peters, the QP for the marketing assumptions, has relied on Ivanhoe and disclaims responsibility for the marketing assumptions in Section 19 and Section 22. Tim Kuhl has also fully relied upon and disclaim responsibility for this information in support of the Mineral Resource estimate in Section 14.

3.4 Environmental

Riann Thysse, the QP for the environmental assumptions, has relied on Ivanhoe and disclaims responsibility for these assumptions and the work presented in Section 20. Ivanhoe provided the following documents that have been used:

- Ivanplats 21 November 2014: Environmental Studies, Permitting and Social or Community Impact.
- Els, M., 2003: Interim Environmental Baseline Report for the Platreef Project: WSP Walmsley, Volume 1 Main Report W603/2, Sandton, Republic of South Africa, and Update to the Executive Summary of the August 2003 Environmental Baseline Report for the Platreef Project S0242, September 2007: unpublished report prepared by WSP Walmsley, Sandton, South Africa for Ivanplats.
- Wessels, B., 2013: Platreef Updated Technical Report: email from Barbara Wessels, Digby Wells Consultant to Wood providing updates on ongoing environmental studies.
- Field D, 2014: Platreef Hydrogeology Report, 26 March 2014, provided by Ivanplats.
- Van Wyk & Veermak 2014: Platreef Project: Summary of Progress on Golder Water and Waste Studies, February 2014, by Golder Associates.
- Mr. Kuhl has also fully relied upon and disclaim responsibility for this information in support of the Mineral Resource estimate in Section 14.

3.5 Infrastructure

Val Coetzee the QP for the Infrastructure, has relied on Ivanhoe and DRA Projects disclaims responsibility for these assumptions and the work presented in Section 18.

Ivanhoe provided the following information that has been used:

- Amended Water Use Licence of 7th September 2021 Licence No. 07/A61G/GCJAIBF/6975

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Summary Introduction

The Platreef platinum–palladium–rhodium–nickel–gold–copper (3PGE+Ni+Au+Cu) project on the Northern Limb of the Bushveld Complex in the Republic of South Africa (the Project), owned by Ivanplats (Pty) Ltd (Ivanplats), comprises a new 4 Mtpa vertical-shaft accessed underground platinum mine, platinum group elements (PGE) concentrator, tailings storage facility (TSF) and supporting infrastructure, located approximately 280 km northeast of Johannesburg and 8 km from the town of Mokopane in Limpopo Province of South Africa.

Corporate Ownership

Up to 2010, Ivanhoe held a 100% interest in the Project. In late 2010, and 2011, Itochu Corporation and the Japanese Consortium obtained an effective 10% interest in the Project for a total consideration of approximately US$290 million. In June 2014 and September 2014, Ivanplats concluded its B-BBEE transaction in 2 phases, achieving a target of 26% Historically Disadvantaged South Africans (HDSA) ownership of Ivanplats.

During June 2014, as part of implementing the B-BBEE transaction, Ivanplats entered into the Consolidated Investors' Agreement with Ivanplats Holding Sàrl, ITC Platinum, Itochu, Ivanhoe and BEE Co ("Investors"). The Consolidated Investors' Agreement regulates the Investors' commercial relationships with each other in respect of Ivanplats and the Project in a consolidated framework, irrespective of whether an Investor holds its ultimate and effective ownership of the Project directly (i.e. Ivanplats Holding, ITC and BEE Co, as shareholders of Ivanplats) or indirectly (i.e. Ivanhoe and Itochu, as shareholders of Ivanplats Holding). The Consolidated Investors' Agreement, among other things, regulates the following aspects of the relationship of the direct and indirect Investors of Ivanplats:

- Appointment of the Ivanplats Board of Directors
- Establishment of the Technical Committee
- Establishment of the Management Committee
- Continued funding of the Project
- Restrictions on disposals of shares or "participating interest" in the Project.

Property Description and Location

The Mining Right Area where the Project is located, comprises the farms Macalacaskop (243 KR) and Turfspruit (241 KR). There are a number of informal settlements located on or near the Mining Right Area.

The farms have been legally surveyed in the past, and the original surveys are on file at the Office of the Surveyor-General of the Limpopo Province (formerly Northern Province) of South Africa. Macalacaskop is filed at that location under reference SG Number 1496/1894. Turfspruit is filed at the same location as reference SG Number A44/1963. Plot surveys and land area calculations were performed by the Surveyor-General.

Ivanplats currently envisages using a portion of the farm Rietfontein (2 KS) for a tailings storage facility and related infrastructure. Rietfontein does not currently form part of the Mining Right Area, but Ivanplats submitted an application for a prospecting right in respect of Rietfontein, which Ivanplats intends to incorporate into the Mining Right Area in due course.

Legal and Titles

The Mining Right was granted to Ivanplats on 30 May 2014 and notarially executed on 4 November 2014. By virtue of the Mining Right, Ivanplats is the sole and exclusive holder of the mining title in, and to, platinum group metals (PGM), gold, silver, nickel, copper, iron, vanadium, cobalt and chrome in respect of the Mining Right Area. The Mining Right commenced on 04 November 2014 and, unless cancelled or suspended in terms of clause 13 of the Mining Right and/or section 47 of the Mineral and Petroleum Resources Development Act, 2002 (MPRDA), will continue in force for a period of 30 years, ending on 03 November 2044, renewable for further periods, each of which may not exceed 30 years at a time. The Mining Right was successfully registered at the Mineral and Petroleum Titles Registration Office (MPTRO) on 03 February 2017.

A number of internal appeals (in terms of section 96 of the MPRDA) against the granting of the Mining Right to Ivanplats were lodged with the Department of Mineral Resources (now known as the Department of Mineral Resources and Energy – DMRE) by the following persons/entities on the following dates:

- Lawyers for Human Rights on behalf of Mokopane Interested and Affected Communities Committee (MIACC), dated 03 July 2014
- Mokopane Interested and Affected Communities Development Forum (MIACDF) dated 01 December 2014
- Mr Aubrey Langa, on behalf of the Kopano Formation Committee, dated 04 December 2014.

The appeal by the Kopano Formation Committee was dismissed by the DMRE. The appeals by MIACC and MIACDF have not yet been decided by the DMRE; however, neither MIACC nor MIACDF have taken any steps to advance these appeals since May 2015. Ivanplats accordingly regards these appeals as having been abandoned by MIACC and MIACDF, respectively. According to legal advice received by Ivanplats, these appeals are without legal or factual basis and are highly unlikely to succeed. In terms of South African law, an administrative action (including, in this instance, the grant of a Mining Right) remains in force pending an appeal, in accordance with the rule of law that administrative actions by organs of state are deemed to be valid until such time as they are reversed, declared invalid and/or set aside by a competent authority or a court of law.

Ivanplats desires various portions of the relevant properties to be zoned more specifically for mining, industrial, unspecified and "special" land use, in order to ensure that the appropriate permanent land use authorisations are in place, and that the property rates and taxes payable in respect of the land is commensurate with its actual use. This process is on-going.

Ivanplats entered into Surface Use and Cooperation Agreements (SUCAs) with the leadership of the following communities, on whose land Ivanplats's mining activities are currently taking place:

- Ga-Magongoa
- Ga-Kgobudi
- Ga-Madiba
- Tshamahansi (comprised of Baloyi, Matjeke and Hlongwane).

The SUCAs were initially concluded when Ivanplats was conducting prospecting activities on the relevant land. The principal purpose of the SUCAs at the time was to provide for adequate compensation for persons whose use of their land (used mainly for subsistence farming) was adversely impacted by Ivanplats's prospecting activities, and provided for seasonal payments to each beneficial landowner in lieu of the use of the productive capacity of the land. With the transition from prospecting activities to mining activities, Ivanplats recognises the need to agree with the affected communities on a long-term solution. The SUCAs currently represent an interim measure to ensure that subsistence farmers continue to receive compensation for the use of their land, pending the conclusion of long-term surface leases with the communities concerned.

Consumer Agreements

Following discussions between Ivanplats and the Mogalakwena Local Municipality (MLM) regarding the construction of the Masodi Waste Water Treatment Works (WWTW), which discussions commenced in 2016, Ivanplats and the MLM concluded a final agreement regarding the supply of treated municipal effluent (grey water) for purposes of supplying Ivanplats' water requirements at the Platreef Mine. This agreement follows two previous agreements, which afforded Ivanplats rights of first refusal in respect of the grey water, in exchange for financial assistance by Ivanplats to the MLM. In terms of the final agreement, Ivanplats undertakes to complete the WWTW, at a cost of approximately ZAR 214 million, and the MLM undertakes to supply grey water from the WWTW to Ivanplats at a cost of R5 per kilolitre (escalated annually for inflation). The duration of the offtake agreement is 32 years from the date on which the Platreef Mine reaches steady state production.

Ivanplats is currently in discussions with Eskom SOC Limited ("Eskom") to agree on the terms and conditions on which Eskom is willing to supply electricity (100 MVA) to Ivanplats. Ivanplats accepted the Budget Quotation, which secures the 100 MVA power supply to the mine, including cost for the design and land acquisition. Discussions for the negotiation of final agreements are underway, and Ivanplats anticipates that the agreements will be concluded by 19 April 2022. Ivanplats will enter into a "Self-Build Agreement" with Eskom in terms of which Ivanplats will construct the infrastructure itself, based on the plans and designs prepared by Eskom (rather than wait for Eskom to construct the powerlines in accordance with Eskom's time lines). Ivanplats will further enter into an "Electricity Supply Agreement" which regulates the terms and conditions on which it will be supplied with electricity. These draft agreements are currently being reviewed by both parties.

Royalties and Encumbrances

In terms of section 25(2)(g) of the MPRDA, the holder of a mining right must pay a royalty to the State in terms of any relevant law. The royalty payable to the State is determined in terms of the Mineral and Petroleum Royalty Act, 2008. In terms of section 2 of the Royalty Act, a royalty is payable for the benefit of the National Revenue Fund in respect of the transfer of a mineral resource extracted from within the Republic of South Africa.

Section 3 of the Royalty Act distinguishes between a "refined mineral resource" and an "unrefined mineral resource", and different formulae for calculating royalties are prescribed in respect of refined and unrefined mineral resources, respectively. The royalties are calculated by multiplying the gross sales of minerals by the mining right holder during the year of assessment by the percentages determined in sections 4(1) and 4(2) of the Royalty Act in respect of refined and unrefined mineral resources. The obligation to pay the State a royalty, and interest on any late payments, is also recorded as a condition of Ivanplats's Mining Right (clauses 5 and 6 thereof). The obligation to pay royalties will arise as and when Ivanplats commences the sale of minerals obtained from its mining and processing operations.

In terms of section 11(1) of the MPRDA, a prospecting right or mining right or an interest in any such right, or a controlling interest in a company or close corporation, may not be ceded, transferred, let, sublet, assigned, alienated or otherwise disposed of without the written consent of the Minister, except in the case of change of controlling interest in listed companies. This provision constitutes an effective restriction on the transfer or securitisation of any shares in Ivanplats itself, as well as, among other things, the cession, transfer, letting or subletting of the mining right or any share or interest in the mining right, unless ministerial consent is obtained.

The Mining Right contains the following further restrictions:

- Mining operations in the mining area must be conducted in accordance with the Mining Work Programme and any amendment to such Mining Work Programme and an approved Environmental Management Plan.
- Ivanplats shall not trespass or enter into any homestead, house or its curtilage nor interfere with or prejudice the interests of the occupiers and/or owners of the surface of the Mining Area except to the extent to which such interference or prejudice is necessary for purposes of enabling the Holder to properly exercise the Holder's rights under this mining right (clause 7.3).
- It is a condition of the granting of the Mining Right that the Holder shall dispose of all minerals and/or products derived from the exploitation of the minerals at competitive market prices which shall mean in all cases, non-discriminatory prices or non-export parity prices. If the minerals are sold to any entity that is an affiliate or non-affiliated agent or subsidiary of the Holder, or is directly or indirectly controlled by the Holder, such purchaser must unconditionally undertake in writing to dispose of the minerals and any products produced from the minerals, at competitive market prices (clause 8).

This notice is an integral component of the Platreef 2022 Feasibility Study (Platreef 2022 FS) and should be read in its entirety and must accompany every copy made of the Platreef 2022 FS. The Platreef 2022 FS has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Platreef 2022 FS has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by principal consultant DRA Global, with economic analysis led by OreWin, and specialised sub-consultants, including Stantec Consulting, Amec Foster Wheeler, Murray & Roberts Cementation, SRK Consulting, Golder Associates and Digby Wells Environmental as the Report Contributors.

The Platreef 2022 FS is based on information and data supplied to the report contributors by Ivanhoe and other parties. The quality of information, conclusions and estimates contained herein are consistent with the level of effort involved in the services of the report contributors, based on (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions and qualifications set forth in this report. Each portion of the report is intended for use by Ivanhoe subject to the terms and conditions of its contract with the report contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the report, by any third party, is at that party's sole risk.

The Platreef 2022 FS is intended to be used by Ivanhoe, subject to the terms and conditions of its contract with the report contributors. Recognising that Ivanhoe has legal and regulatory obligations, the report contributors have consented to the filing of the Platreef 2022 FS with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR).

The results of the Platreef 2022 FS represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Platreef 2022 FS. Readers are cautioned that actual results may vary from those presented.

The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report in each relevant section.

The conclusions and estimates stated in the Platreef 2022 FS are to the accuracy stated in the Platreef 2022 FS only and rely on assumptions stated in the Platreef 2022 FS. The results of further work may indicate that the conclusions, estimates and assumptions in the Platreef 2022 FS need to be revised or reviewed.

The report contributors have used their experience and industry expertise to produce the estimates and approximations in the Platreef 2022 FS. Where the report contributors have made those estimates and approximations, they are subject to qualifications and assumptions, and it should also be noted that all estimates and approximations contained in the Platreef 2022 FS will be prone to fluctuations with time and changing industry circumstances.

The Platreef 2022 FS should be construed in the light of the methods, procedures, and techniques used to prepare the Platreef 2022 FS. Sections or parts of the Platreef 2022 FS should not be read in isolation of, or removed from, their original context.

4.2 Title and Legal

4.2.1 Corporate Ownership

Ivanplats' shareholding is as follows:

- Ivanplats Holding Sàrl: 66%
- K2014089596 (South Africa) (RF) Proprietary Limited, ("BEE Co"): 26%
- ITC Platinum Development Limited: 8%

The shareholding in Ivanplats Holding Sàrl is as follows:

- Ivanhoe Mines Limited: 97.3% of total shares in issue
- Itochu Corporation: 2.7% of total shares in issue

The shareholding in BEE Co is as follows:

- K2014043822 (South Africa) (RF) Proprietary Limited ("Community Trust Co"): 76.92%
- Sekgomantsha (RF) Proprietary Limited ("Employee Trust Co"): 11.54%
- K2014043815 (South Africa) (RF) Proprietary Limited ("Entrepreneur Co"): 11.54%

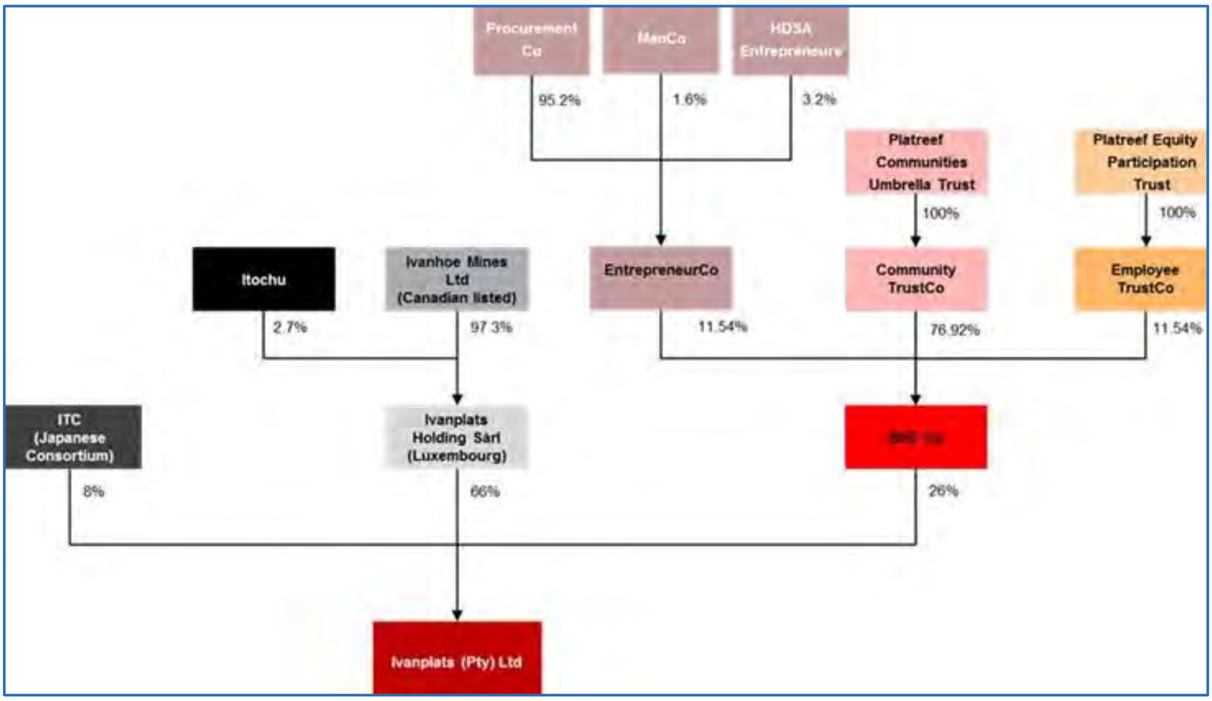
Community Trust Co is 100% owned by the Bonega Communities Trust, a trust established for the benefit of the inhabitants of the 20 villages comprising the Mokopane Community, a recognised traditional community under the jurisdiction of the Mokopane Traditional Council.

Employee Trust Co is 100% owned by the Sekgomantsha Trust, a trust established for the benefit of historically disadvantaged, non-managerial employees of Ivanplats.

The shares in Entrepreneur Co are held by a consortium of historically disadvantaged entrepreneurs residing in and around the town of Mokopane, where Ivanplats' mining activities take place, and historically disadvantaged senior managerial employees of Ivanplats.

The shareholding structure of Ivanplats and its shareholders is graphically represented in Figure 4.1 below.

Figure 4.1 Ivanplats' Ownership Structure



Ivanplats, 2022

The final B-BBEE structure was the product of a thorough process of engagement between the Ivanhoe group, representatives of the local villages, local entrepreneurs, Ivanplats employees and the DMRE as regulator.

The B-BBEE structure has the following key features:

- The main empowerment vehicle (BEE Co) is 100% owned by HDSAs having the following effective indirect ownership interests in Ivanplats:
- 20% is held by the trust established for the benefit of 20 local villages (or communities) in the vicinity of the Platreef Project. Through the implementation of projects to be identified by the trustees of the trust, it is envisaged that all members of these 20 communities will benefit from the funds flowing to the Bonega Communities Trust;
- 3% is held by the Sekgomantsha Trust, the trust established for the benefit of historically disadvantaged, non-managerial South African employees of Ivanplats; and
- 3% is held by a consortium of local HDSA entrepreneurs (including Ivanplats managerial employees who elected to participate in this consortium). The majority of this consortium consists of local HDSA entrepreneurs who are registered on Ivanplats' procurement database.
- The effective B-BBEE ownership interests are held through a structure designed with the primary objectives of separately housing the interests of HDSAs and facilitating the funding of the acquisition of their respective shares, while shielding the ultimate beneficiaries (individuals) from liability for repayment of the loans.

- The acquisition of these ownership interests are, except in the case of a limited number of local entrepreneurs, fully vendor-funded in such a manner as to impose no risk on the HDSAs who are the ultimate beneficiaries of the structure. The vendor loans provided to fund the acquisition of these ownership interests are to be repaid from portions of the dividends declared by Ivanplats from time to time.
- In the case of the limited number of local entrepreneurs who had the means to fund the acquisition of their ownership interests themselves, they take the normal risks inherent in an investment of this nature, but they also enjoy the increased benefit of the full dividends paid to them in cash from time to time, without deductions made towards repayment of vendor funding.

4.2.2 Property Description and Location

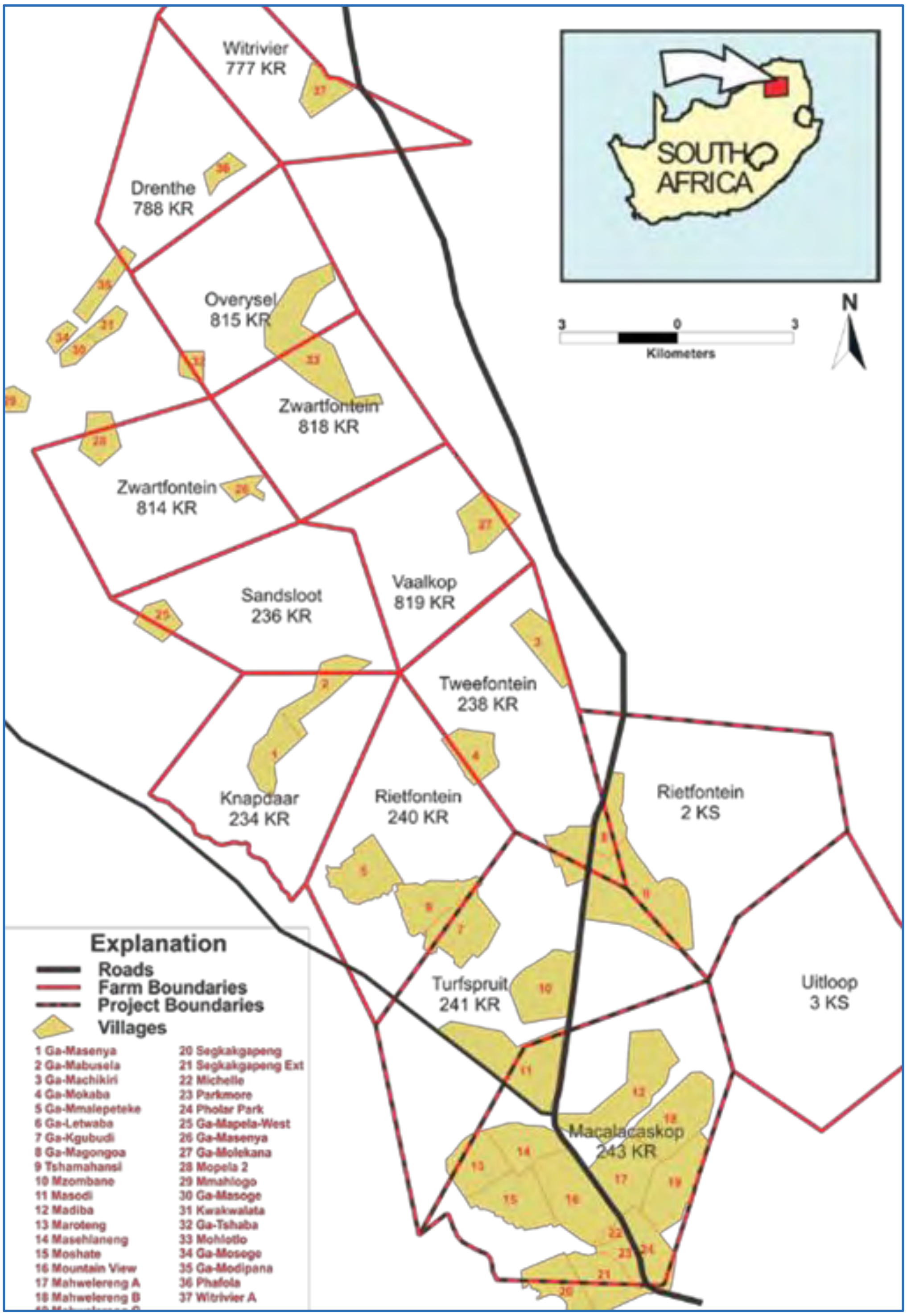
The Platreef Project is located at about 24º05'S and 28º59'E. The Project is located in the Limpopo Province of the Republic of South Africa on two farms: Turfspruit (3,561 ha), and Macalacaskop (4,281 ha). Ivanplats intends to extend the Platreef Project to Rietfontein (2,878 ha) in order to accommodate a TSF with related infrastructure.

Mining Right LP30/5/2/2/1/10067MR boundaries correspond to the perimeter boundaries of the Macalacaskop (243 KR) and Turfspruit (241 KR) farms. Khaki areas on the plan depicted in Figure 4.2 are the main settlements and townships. The Ivanhoe-controlled farms, Macalacaskop 243 KR and Turfspruit 241 KR, are contiguous, sharing a common boundary along the north-west border of Macalacaskop and the south-eastern border of Turfspruit. Macalacaskop contains 4,281 ha of land. Turfspruit contains 3,561 ha of land. The combined total is 7,842 ha.

The farms have been legally surveyed in the past, and the original surveys are on file at the Office of the Surveyor-General of the Limpopo Province (formerly Northern Province) of South Africa. Macalacaskop is filed at that location under reference SG Number 1496/1894. Turfspruit is filed at the same location as reference SG Number A44/1963. Plot surveys and land area calculations were performed by the Surveyor General as indicated on the registered diagrams: SG Diagram No. A 44/63 (Turfspruit 241 KR) and No. A 45/63 (Macalacaskop 243 KR).

Rietfontein Farm 2 KS has a contiguous border with Turfspruit 241 KR, sharing a common boundary along the south-western border of Rietfontein and the north-eastern border of Turfspruit. Rietfontein Farm has an area of 2,878 ha.

Figure 4.2 Project Location and Farm Boundaries



Ivanplats, 2022

4.3 Legal and Titles

4.3.1 Mining Titles

4.3.1.1 Turfspruit 241 KR and Macalacaskop 243 KR

Ivanplats Proprietary Limited (which was formerly known as Platreef Resources Proprietary Limited) was the holder of a prospecting right, which was notarially executed on 2 February 2006 and registered in the Mineral and Petroleum Titles Registration Office (MPTRO) on 9 February 2006 under registration number 55/2006 PR, which prospecting right entitled Ivanplats to prospect, for its own account, for base minerals and precious metals in, on and under the farm Turfspruit 241 KR and the farm Macalacaskop 243 KR, for a period of five years commencing on 2 February 2006 and ending on 1 February 2011 ("the Prospecting Right"). The Prospecting Right was subsequently renewed on 1 June 2011, for a period of three years, commencing on 1 June 2011 and ending on 31 May 2014.

On 6 June 2013, Ivanplats lodged an application for a mining right in terms of section 22 of the Mineral and Petroleum Resources Development Act, 2002 (MPRDA) in respect of Platinum Group Metals (PGM): Platinum (Pt), Rhodium (Rh) Palladium (Pd), Iridium (Ir), Ruthenium (Ru) and Osmium (Os), and all associated metals and minerals mined out of necessity and convenience together with the platinum group metals including but not limited to Gold (Au), Silver (Ag), Copper (Cu), Cobalt (Co), Iron (Fe), Vanadium (V) and Chrome (Cr), over the farms Macalacaskop 243 KR and Turfspruit 241 KR.

The DMR informed Ivanplats of the successful outcome of its mining right application by means of a letter dated 30 May 2014, in terms of which the Director-General, in terms of the powers delegated to him by the Minister in terms of section 103(1) of the MPRDA, informed Ivanplats that its application for a mining right in respect of Platinum Group Metals, Au, Ag, Ni, Cu, Fe, V, Co and Cr in respect of the farms Macalacaskop 243 KR and Turfspruit 241 KR (excluding areas comprising graveyards, built-up areas and protected areas) had been granted in terms of section 22(1) (sic) of the MPRDA.

Ivanplats and the Regional Manager: Limpopo Region of the DMR notarially executed a mining right on 4 November 2014 with DMR reference number LP 30/5/2/2/1/10067 MR ("the Mining Right"). By virtue of the Mining Right, Ivanplats is the sole and exclusive holder of the mining title in and to Platinum Group Metals, Gold, Silver, Nickel, Copper, Iron, Vanadium, Cobalt and Chrome in respect of the farms Macalacaskop 243 KR and Turfspruit 241 KR (excluding areas comprising graveyards, built-up areas and protected areas). The Mining Right commenced on 4 November 2014 and, unless cancelled or suspended in terms of clause 13 of the Mining Right and/or section 47 of the MPRDA, will continue in force for a period of 30 years ending on 3 November 2044, renewable for further periods, each of which may not exceed 30 years at time.

In terms of section 25(2)(a) of the MPRDA, the holder of a mining right must lodge the right for registration at the MPTRO within 60 days of the right becoming effective (i.e. the date of notarial execution of the Mining Right). The MPRDA does not require that a right must be registered within a certain period of time; only that it must be lodged for registration. The Mining Right was lodged for registration on 3 December 2014 – well before the expiry of the prescribed 60 day period. Following a number of queries from the MPTRO, the mining right was successfully registered 3 February 2017.

The version of the Mining Right which was executed on 4 November 2014 and registered by the MPTRO on 3 February 2017 contained the following rider in clause 17 thereof (quoted verbatim):

"In the furthering the objects of this Act, the Holder is bound by the provisions of an agreement or arrangement dated 31st October 2014 entered into between the Holder/empowering partner and Community Trust Co (K2014043822) 20%, Employee Trust Co (K2014043829) 3%, Entrepreneur Co (K201408959596) 3% and 74% is held by Itochu and Ivanplats Holdings Sàrl (The participation interest of the HDSA shall be restructured through the increase of the entrepreneurs' stake to a minimum of 5%, so as to enable emerging black entrepreneurs to acquire a stake as part of enhancing their meaningful participation in the mining industry). The restructuring shall be done by no later than 30 June 2016." ("the BEE Restructuring Condition")

Ivanplats lodged an appeal (in terms of section 96 of the MPRDA) against the BEE Restructuring Condition on 17 December 2014. The South African Minister of Mineral Resources upheld Ivanplats' appeal, and issued a decision on 12 May 2015 (but only received via post by Ivanplats legal advisors on 15 September 2015) in terms of which the BEE Restructuring Condition was withdrawn and replaced with the following wording:

"In furthering the objects of this Act, the Holder is bound by the provisions of a suite of black economic empowerment agreements implemented on 26 June 2014 and 3 September 2014, in terms of which K2014408996 (South Africa) (RF) Proprietary Limited, ("BEE Co") holds 26% of the issued shares in the Holder, and through their shareholding in BEE Co, the following entities hold the following indirect ownership interest in the Holder, namely K201404822 (South Africa) (RF) Proprietary Limited ("Community Trust Co") – 20%, K201404815 (South Africa) (RF) Proprietary Limited ("Entrepreneur Co") – 3%, and K201404829 (South Africa) (RF) Proprietary Limited ("Employee Trust Co") – 3%, which suite of agreements was taken into consideration for purposes of compliance with the requirements of the Act and or Broad Based Socio-Economic Empowerment Charter for the South African Mining and Minerals Industry developed in terms of the Act and such suite of agreements shall form part of this right."

The following internal appeals (in terms of section 96 of the MPRDA) against the grant of the mining right to Ivanplats were lodged with the DMR by the following persons/entities on the following dates:

- Lawyers for Human Rights on behalf of the Mokopane Interested and Affected Communities Committee (MIACC), dated 3 July 2014;
- the Mokopane Interested and Affected Communities Development Forum (MIACDF) dated 1 December 2014; and
- Mr Aubrey Langa, on behalf of the Kopano Formation Committee, dated 4 December 2014.

The DMR has formally notified Ivanplats of the appeals lodged by MIACDF (on 18 August 2015) and the Kopano Formation Committee (on 10 December 2014), but not the appeal lodged by MIACC.

Ivanplats responded to the MIACDF appeal on 30 September 2015, and to the Kopano Formation Committee appeal on 27 January 2015.

The DMRE dismissed the appeal by the Kopano Formation Committee.

As far as the MIACC appeal is concerned, Ivanplats is entitled to wait until it receives a formal notification by the DMR before it responds to the appeal. However, Lawyers for Human Rights have subsequently withdrawn its representation of MIACC (the community-based organisation on behalf of which the appeal was lodged) and MIACC have not subsequently taken any steps to bring the appeal to finalisation. Ivanplats is on good terms with MIACC and MIACDF, and is of the view that these appeals have been abandoned. In terms of legal advice received by Ivanplats, the appeals are without legal or factual basis and are very unlikely to succeed.

In terms of South African law, an administrative action (including, in this instance, the grant of a mining right) remains in force pending an appeal, in accordance with the rule of law that administrative actions by organs of state are deemed to be valid until such time as they are reversed, declared invalid and/or set aside by a competent authority or a court of law.

As the holder of a mining right, Ivanplats has, inter alia, the following rights and obligations with effect from the notarial execution of the mining right (i.e. from 4 November 2014):

- In terms of section 5 of the MPRDA, the holder of a mining right has the right to:
  - enter the land to which such right relates together with its employees, and bring onto that land any plant, machinery or equipment and build, construct or lay down any surface, underground or under sea infrastructure which may be required for the purpose of mining;
  - prospect or mine for its own account on or under that land for the mineral for which such right has been granted;
  - remove and dispose of any such mineral found during the course of prospecting or mining;
  - subject to the National Water Act, 1998 (Act No. 36 of 1998), use water from any natural spring, lake, river or stream, situated on, or flowing through, such land or from any excavation previously made and used for prospecting or mining purposes, or sink a well or borehole required for use relating to prospecting or mining, on such land; and
  - carry out any other activity incidental to prospecting or mining operations, which activity does not contravene the provisions of the MPRDA;
- In addition to the rights referred to in section 5, the holder of a mining right has, subject to section 24 of the MPRDA, the exclusive right to apply for and be granted a renewal of the mining right in respect of the mineral and mining area in question.

4.3.1.2 Rietfontein 2 KS

Historically, Atlatsa Resources Corporation (formerly Anooraq Resources Corporation) through its South African subsidiary, Plateau Resources Limited, held an exclusive prospecting right (No. MPT 76/2007 PR) to prospect for base and precious metals on the farm Rietfontein 2 KS, over a total area of 2 878ha. Ivanhoe had a joint venture agreement with Atlatsa in respect of Rietfontein.

The prospecting right was finally expired in 2019.

Ivanplats submitted an application for a new prospecting right in respect of Rietfontein in September 2019. As required by law, the application was accompanied by an application for an environmental authorisation (EA). During December 2020, the DMRE refused the EA application, citing insufficient public participation as reason for the refusal. The difficulty was that the public participation process was undertaken during 2020, when restrictions on large gatherings applied in terms of the COVID-19 regulations issued under the Disaster Management Act, 2002.

Although the EA application was refused, the prospecting right application itself was not refused. Ivanplats therefore submitted a new EA application early in 2021, and undertook a new public participation process, addressing the shortcomings identified by the DMRE in respect of the first public participation process.

The DMRE has subsequently advised Ivanplats that its system cannot link the new EA application to a pre-existing prospecting right application, and requested Ivanplats to resubmit the prospecting right application as well. Ivanplats submitted new prospecting right application and a new EA application in respect of Rietfontein on 16 February 2022.

4.3.2 Land Use Planning

The Spatial Planning and Land Use Management Act, 2013 (SPLUMA) came into force on 1 July 2015, and replaced the various systems of provincial and municipal town planning regulation, including the Town-planning and Townships Ordinance, 1986 ("the Ordinance"), which was applicable in Limpopo Province until 30 June 2015, with a uniform system applicable throughout South Africa.

In terms of section 60(1) of the SPLUMA, the repeal of laws in relation to provincial or municipal planning does not affect the validity of anything done in terms of that legislation. Accordingly, town planning schemes in Limpopo Province adopted under the Ordinance remain in force until such time as they are replaced with new land use planning schemes to be developed by municipalities in terms of the SPLUMA.

Prior to the commencement of the SPLUMA, the Mogalakwena Municipality adopted a town planning scheme, known as the Mogalakwena Land Use Management Scheme, 2008 ("the Scheme"), in terms of section 39(1) of the Town Planning and Townships Ordinance, 1986 (Ordinance 15 of 1986). The Scheme came into force on 30 October 2009 by proclamation under General Notice No. 371 of 2009 in the Limpopo Provincial Gazette No. 1697.

In terms of the Scheme and the relevant maps forming part thereof, the farms Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS were zoned "Mining 2".

In terms of section 1.2.46 of the Scheme, "Mining 2" is defined as meaning "land with ore bodies and/or mineral potential/occurrences with or without mining rights in terms of existing mining and mineral legislation. The minerals are therefore likely to be extracted in future". In terms of Table 1 on page 29 of the Scheme, there are no prohibited land uses listed in respect of land zoned "Mining 2".

In terms of section 26(2) of SPLUMA, land may be used for purposes permitted under a town planning scheme (adopted under the Ordinance), until such scheme is replaced by a land use scheme adopted under SPLUMA.

On 24 June 2016, the Mogalakwena Municipality adopted a new Land Use Management By-Law, (Proclamation 19 of 2016, Notice No. 2723, Limpopo Provincial Gazette Vol. 23) in order to give effect to the provisions of SPLUMA. In terms of the transitional arrangements set out in Item 3(3) and 3(4) of the Mogalakwena Land Use Management By-Law, land uses which were undertaken lawfully prior to the effective date of the Mogalakwena Land Use Management By-Law, may continue for a period of 15 years even if the land is not zoned for that purpose. Ivanplats is currently undertaking its mining- and related operations in accordance with these transitional arrangements.

Ivanplats desires various portions of the relevant properties to be zoned more specifically for mining, industrial, unspecified and "special" land uses under the new land use management scheme, in order to ensure that the property rates and taxes payable in respect of the land is commensurate with its actual use. To this end, an application for re-zoning of the land in terms of section 41(2) of the SPLUMA, read with the provisions of the Mogalakwena Land Use Management By-law, 2016, has been prepared and is substantially ready for submission to the relevant municipal planning tribunal established under the SPLUMA. The only outstanding document which is required before the re-zoning application can be submitted is a special power of attorney on behalf of the nominal landowner of the farms Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS, being the Minister of Rural Development and Land Reform, on behalf of the Republic of South Africa.

The Minister of Rural Development and Land Reform has indicated that the special power of attorney can only be furnished when Ivanplats has entered into long-term surface lease agreements with the relevant communities on whose land the mining activities are taking place. The status of the surface lease process is dealt with in Paragraph 4.3.3.2 below.

4.3.3 Surface Rights

4.3.3.1 Surface Use and Cooperation Agreements

Ivanplats entered into Surface Use and Cooperation Agreements (SUCAs) with the leadership of the following communities, on whose land Ivanplats' mining activities are taking place:

- Ga-Magongoa;
- Ga-Kgobudi;
- Ga-Madiba; and
- Tshamahansi (comprised of Baloyi, Matjeke and Hlongwane).

The SUCAs were initially concluded when Ivanplats (known at the time as Platreef Resources) was conducting prospecting activities on the relevant land. The chief purpose of the SUCAs at the time was to provide for adequate compensation for persons whose use of their land (used mainly for subsistence farming) was adversely impacted by Ivanplats' prospecting activities, and provided for seasonal payments to each beneficial landowner in lieu of the use of the productive capacity of the land.

With the transition from prospecting activities to mining activities, Ivanplats recognises the need to agree with the affected communities on a long-term solution, in terms of which beneficial landowners are compensated with alternative land, or another suitable form of compensation, as part of a broader livelihood restoration programme. The desired outcome of this engagement process will be twofold: individual subsistence farmers will receive alternative means to make a living (preferably in the form of alternative land with similar or better agricultural properties), and Ivanplats will enter into long-term surface leases with the communities whose land is utilised for mining activities, in terms of which the communities will be collectively compensated for the use of their land.

The SUCAs presently represent an interim measure to ensure that subsistence farmers continue to receive compensation for the use of their land, pending the conclusion of long-term surface leases with the relevant communities.

Ivanplats has made significant progress in terms of developing and implementing a livelihood restoration programme to address the need of subsistence farmers to find alternative means to make a living. In this regard, Ivanplats appointed a consultant, Synergy, to undertake a consultation process and specialist studies and to develop a livelihood restoration programme. As part of the livelihood restoration programme, and in conjunction with other initiatives to develop new economic activities in the affected area, Synergy proposed a once-off compensation payment to the beneficial landowners. As at the time of this feasibility study, Ivanplats has concluded compensation agreements with a number of the affected beneficial owners and foresees that it will be in a position to conclude such agreements with the vast majority, if not all, of the beneficial landowners.

4.3.3.2 Surface Leases

In South African law, surface lease agreements which will endure for longer than ten years ought to be notarially executed and registered in the applicable deeds registry. The long-term lease agreement is then noted on the title deed in respect of the land as a condition which affects the landowner's use and enjoyment of the land and which will be binding on the landowner's successors-in-title.

In order to achieve the notarial execution of a long-term lease agreement, and the subsequent registration thereof against the title deed of the land, the co-operation of the registered landowner (as reflected in the deeds registry) is required.

The registered landowner in respect of the farms Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS, is the Republic of South Africa, represented by the Minister of Rural Development and Land Reform.

Notwithstanding the State's formal ownership of the land in question, the members of the abovementioned communities are recognised as holders of "informal land rights" in terms of the Interim Protection of Informal Land Rights Act, 1996 (IPILRA), and are for all practical purposes to be treated as the de facto landowners. In this regard, section 2 of the IPILRA contains the following relevant provisions:

"2(1) Subject to the provisions of subsection (4), and the provisions of the Expropriation Act, 1975 (Act No. 63 of 1975), or any other law which provides for the expropriation of land or rights in land, no person may be deprived of any informal right to land without his or her consent.

(2) Where land is held on a communal basis, a person may, subject to subsection (4), be deprived of such land or right in land in accordance with the custom and usage of that community.

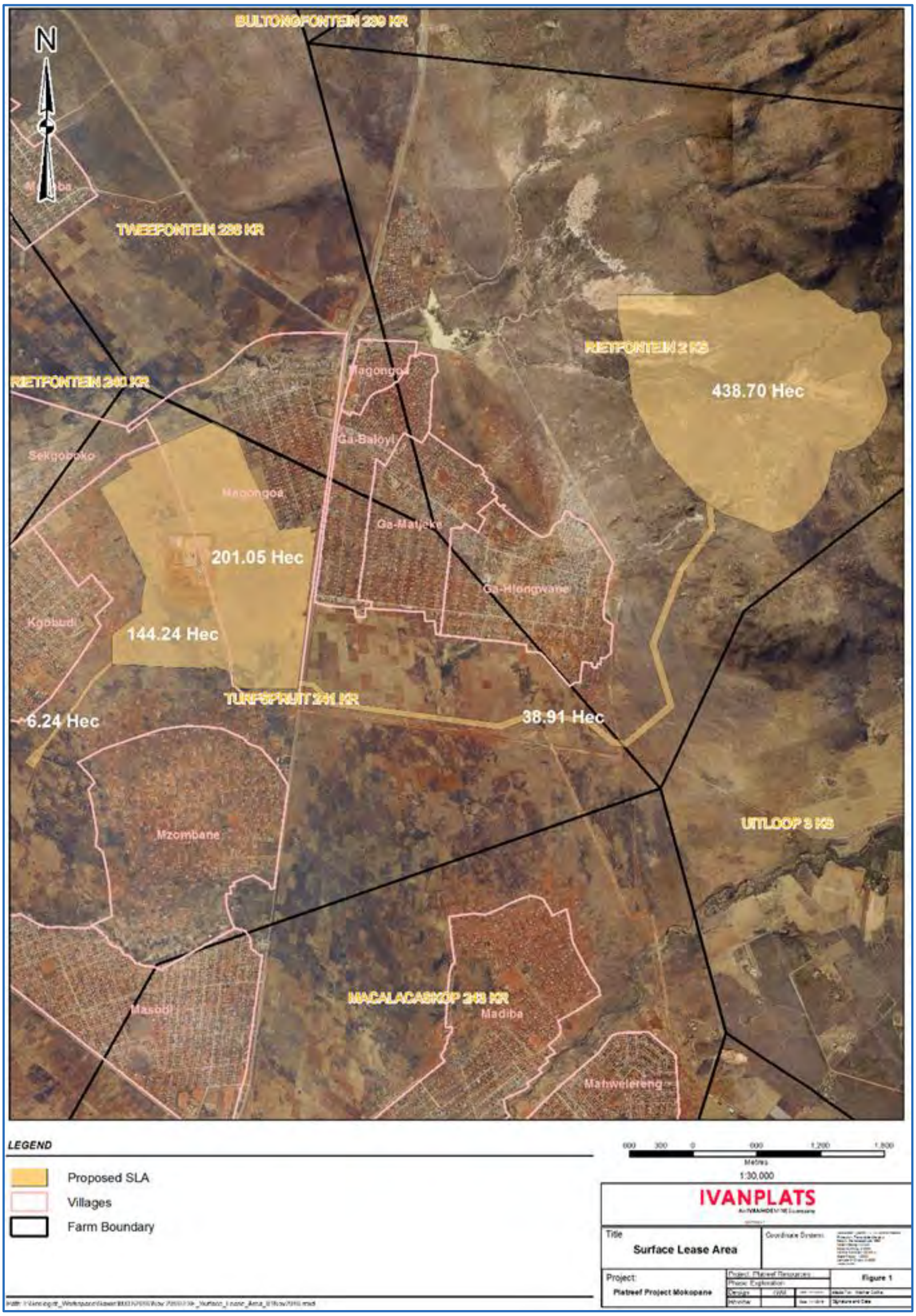
(4) For the purposes of this section the custom and usage of a community shall be deemed to include the principle that a decision to dispose of any such right may only be taken by a majority of the holders of such rights present or represented at a meeting convened for the purpose of considering such disposal and of which they have been given sufficient notice, and in which they have had a reasonable opportunity to participate."

The cumulative effect of the provisions of the IPILRA, as well as the fact that the Republic of South Africa is the registered landowner in respect of the relevant properties, is that the Minister of Rural Development and Land Reform (as duly authorised representative of the registered landowner) will only execute a notarial lease agreement, and co-operate with the registration thereof in the deeds registry, if the lease agreement is approved by the relevant community in accordance with its usage and custom. By law, this usage and custom includes the principle that a decision to enter into such a lease may only be taken by a majority of the members of the relevant community present or represented at a meeting convened for the purpose of considering such lease and of which they have been given sufficient notice, and in which they have had a reasonable opportunity to participate.

Ivanplats is currently in discussions with the various communities, as well as the Department of Rural Development and Land Reform, to arrange for such meetings to be finalised, so that the surface lease agreements can be approved by the relevant communities. Once this has been done, the Minister of Rural Development and Land Reform will execute the lease agreements on behalf of the State. The process of holding consultation meetings and meetings where the resolutions can be adopted by the relevant communities was delayed by legal restrictions on large gatherings forming part of the South African Government's response to the COVID-19 pandemic. In circumstances where the restrictions are being eased progressively, Ivanplats is hopeful that this process can be concluded in the near future.

As mentioned above, the grant of a special power of attorney by the Minister of Rural Development and Land Reform to authorise the re-zoning application under section 41 of the SPLUMA is also subject to conclusion of the relevant surface lease agreements. See Figure 4.3 below for a depiction of surface lease and servitude areas.

Figure 4.3 Surface Lease and ServitudeAreas



Ivanplats, 2022

4.3.3.3 Land Claims

In terms of the Restitution of Land Rights Act, 1994 ("Restitution Act"), persons or communities who were dispossessed of rights in land after 19 June 1913 as a result of past racially discriminatory laws or practices, are entitled to lodge claims for restitution of such land rights (colloquially referred to as "land claims").

A land claim has been lodged by the Mamashela family in respect of the farm Rietfontein 2 KS. In terms of documents issued by the Office of the Regional Land Claims Commissioner: Limpopo Province, the land claim has been found to be "prima facie valid" and has been "approved" by the Regional Land Claims Commissioner on 11 July 2013.

Ivanplats has received legal advice to the effect that the land claim may be finalised in a number of different manners, namely:

- The landowner (in this case, the State) and the claimant, under mediation of the Regional Land Claims Commissioner, may enter into an agreement as to how the land rights may be restored to the claimant. The agreement could provide for an alternative method of settling the claim, such as partial restitution, monetary compensation or alternative land;
- Failing agreement, the Land Claims Commission may refer the claim to the Land Claims Court for final determination;
- The Land Claims Court may, inter alia:
  - dismiss the claim; or
  - order restitution of all or part of the land (potentially in conjunction with other claimants); or
  - order the award of alternative land; or
  - order payment of monetary compensation by the State.

Ivanplats is aware that the Mokopane Community challenged the approval of the Mamashela family's land claim and questions how the land claim could be granted under circumstances where the Mokopane Community as a whole asserts historical ownership in respect of the farm Rietfontein 2 KS.

Section 11(7) of the Restitution Act imposes the following restrictions in relation to land in respect of which a land claim has been gazetted:

"(7) Once a notice has been published in respect of any land–

(a) no person may in an improper manner obstruct the passage of the claim;

(aA) no person may sell, exchange, donate, lease, subdivide, rezone or develop the land in question without having given the regional land claims commissioner one month's written notice of his or her intention to do so, and, where such notice was not given in respect of–

(i) any sale, exchange, donation, lease, subdivision or rezoning of land and the Court is satisfied that such sale, exchange, donation, lease, subdivision or rezoning was not done in good faith, the Court may set aside such sale, exchange, donation, lease, subdivision or rezoning or grant any other order it deems fit;

(ii) any development of land and the Court is satisfied that such development was not done in good faith, the court may grant any order it deems fit;

(b) no claimant who occupied the land in question at the date of commencement of this Act may be evicted from the said land without the written authority of the Chief Land Claims Commissioner;

(c) no person shall in any manner whatsoever remove or cause to be removed, destroy or cause to be destroyed or damage or cause to be damaged, any improvements upon the land without the written authority of the Chief Land Claims Commissioner;

(d) no claimant or other person may enter upon and occupy the land without the permission of the owner or lawful occupier."

The provisions of section 11(7) of the Restitution Act impact on the processes which must be followed in relation to the conclusion of a long-term surface lease in respect of Rietfontein 2 KS, as well as the development of the property, in that the regional land claims commissioner must be notified of any intention to develop the land and/or to enter into any lease.

4.3.3.4 Heritage

As part of the environmental impact assessment process and in support of Ivanplats's mining right application, a Heritage Impact Assessment (HIA) Report was compiled by Digby Wells. The HIA Report identified 55 burial grounds which would be impacted by the mining operations and would require relocation if mitigation is not possible. According to the HIA Report, grave relocation will be necessary in respect of the 42 burial grounds which are located within the Operational Area. According to the HIA Report, the age of these graves is "approximately 100 years to present." The age of the other graves on the properties is unknown. The HIA Report indicates that present day settlements of Tshamahansi, Mahwereleng, Ga-Madiba, Maroteng and Masodi are situated on the mining properties, and the burial grounds are possibly affiliated with the local community.

The HIA Report was made available to the South African Heritage Resources Agency (SAHRA) for review. On 8 November 2013, SAHRA issued a 'Final Comment', setting out a number of recommendations, in particular that a permit in terms of section 36 of the National Heritage Resources Act 25 of 1999 (NHRA) for the relocation of graves must be obtained from SAHRA.

Ivanplats is in the process of undertaking grave relocations and other mitigation measures in respect of the identified heritage resources. The grave relocations have been divided into three phases, of which Phase 1 and Phase 2 have been completed to date. Ivanplats is currently in the process of obtaining the necessary permits and consents to complete the third and final phase of grave relocations.

4.3.3.5 Servitudes, Wayleaves and Permits

In South African law, a "servitude" constitutes a limited real right in land which entitles the holder of the servitude to use the servitude area for one or more specific purpose(s), despite the fact that the ownership of the land vests in another. A servitude is enforceable against the landowner and against third parties, and constitutes a subtraction from the dominium (ownership) of the land.

Typical examples of servitudes are:

- aqueduct (the right to convey water over the property of another)
- usufruct (the right to enjoy the fruits/economic potential of the land of another); and
- roads, power lines, pipes, conveyor belts and other infrastructure over the land of another.

Ivanplats may require a servitude to convey water and tailings (by means of pipes) from its mining operations (situated on the farms Turfspruit 241 KR and Macalacaskop 243 KR) to its proposed tailings storage facility (TSF) situated on the farm Rietfontein 2 KS. This would be done by means of a notarial deed of servitude, to be concluded between Ivanplats, the Minister of Rural Development and Land Reform (as registered owner of the land), and the Tshamahansi community as the holder of informal land rights over the land in question.

An alternative option would be to include the proposed servitude area of the TSF line (depicted in Figure 4.3 above) in the relevant surface lease agreement to be concluded with the Tshamahansi community and the Minister of Rural Development and Land Reform (as registered landowner). In this case, the surface lease itself will embody Ivanplats' right to convey water and tailings over the relevant land, and a separate servitude will not be required.

Upon the conclusion of a long-term electricity supply agreement between Ivanplats and Eskom, it may also be necessary to conclude and register one or more servitudes over the land in respect of the power lines which will convey the electricity to Ivanplats' operations. Again, this will require the co-operation of the community and the Minister of Rural Development and Land Reform (as registered landowner). However, Ivanplats will potentially not be a party to such agreement, as electricity servitudes usually vest in Eskom, which in turn owes a contractual obligation to the customer (Ivanplats) in respect of the supply of electricity. Ivanplats has been informed by Eskom that the land rights for 132 kv servitude and substation have been confirmed.

At this stage, no other servitudes or wayleaves are presently required for purposes of the proposed mining operations.

A description of the permits required for Ivanplats' proposed mining operations appears in Section 20.

4.3.4 Consolidated Investors Agreement

During June 2014, as part of implementing the B-BBEE transaction, Ivanplats entered into the Consolidated Investors' Agreement with Ivanplats Holding Sàrl ("Ivanplats Holding"), ITC Platinum Development Limited ("ITC"), Itochu Corporation ("Itochu"), Ivanhoe Mines Limited ("Ivanhoe") and BEE Co ("Consolidated Investors' Agreement"). The Consolidated Investors' Agreement sets out the terms and conditions in a consolidated framework in terms of which, irrespective of whether an investor (collectively, Ivanplats Holding, ITC, Itochu, Ivanhoe and BEE Co) hold their ultimate and effective ownership of the Project directly (being Ivanplats Holding, ITC and BEE Co, as shareholders of Ivanplats) or indirectly (being Ivanhoe and Itochu, as shareholders of Ivanplats Holding).

The Consolidated Investors' Agreement (amongst other things), regulates the following aspects of the relationship of the direct and indirect investors of Ivanplats:

Ivanplats Board of Directors

- The board of directors of Ivanplats shall be limited to nine directors, nominated as follows: Ivanplats Holding may nominate for appointment three directors, ITC may nominate for appointment one director and BEE Co may nominate for appointment one director.
- The board shall seek to appoint not less than two independent directors to serve on the board of directors of Ivanplats.

Establishment of the Technical Committee

- A technical committee is established through which the Investors consult and discuss with one another in good faith the planning and execution of all work programmes for the Project. This committee may not be disestablished for so long as ITC and Itochu hold in aggregate not less than a 2% "participating interest" in the Project.
- The Technical Committee consists of not more than nine members and is comprised as follows: for so long as ITC and Itochu hold in aggregate not less than a 2% "participating interest" in the Project, ITC and/or Itochu may appoint two members, Ivanhoe may appoint five members and BEE Co may appoint two members.

Establishment of the Management Committee

- A management committee is established, which committee is responsible for the supervision, control, management and operation of the Project, is responsible to the Investors for the conduct of the Project and has the authority to make decisions and to exercise powers necessary for the supervision, control, management, development, construction, financing and operation of the Project.
- The Management Committee consists of not more than nine members and is comprised as follows: for so long as ITC and Itochu hold at least a 2% "participating interest" in the Project, ITC and/or Itochu may appoint two members, Ivanhoe may appoint five members and BEE Co may appoint two members to the Management Committee.

Continued Funding of the Project

- Certain exploration, development and production activities (the Phase II Work Programmes) are funded from existing funds of Ivanplats. However, if the Management Committee determines (in accordance with the rules set out in the Consolidated Investors' Agreement) that further funding is required by Ivanplats for the Project, Ivanplats Holding and Ivanplats shall each deliver a notice to their respective shareholders stipulating the aggregate amount of funding required from each shareholder (calculated on a pro rata basis of their direct and indirect interest in Ivanplats), which shall be advanced as a loan to Ivanplats Holding or Ivanplats, as the case may be. If BEE Co does not provide the funding it is called upon to provide (or provides less than its share of the funding), Ivanhoe shall pay such amount or shortfall on behalf of BEE Co. Should any other investor fund less than its proportionate share, the other investors may fund some or all of that shortfall.
- Ivanplats shall not make any distributions or repay any of the funding advanced by investor before it has not first repaid the funding advanced to it or Ivanplats Holding at the date on which the B-BEE transaction was implemented.

Restrictions on Disposals of Shares/"Participating Interest"

- The Consolidated Investors' Agreement provides for certain restrictions on the investors to dispose of their shares/"participating interest" in the Project.
- Ivanhoe and Ivanplats Holding granted to Itochu and ITC a right of first offer to purchase all or any portion of their shares in Ivanplats Holding or Ivanplats, as appropriate, provided that Ivanhoe's written consent will be required if, as a result of the acceptance by Itochu and/or ITC of all or a portion of such offered shares, Itochu and/or ITC hold "participating interests" in the Project in excess of 12%.
- Each of Itochu and ITC granted to Ivanhoe a right of first refusal to purchase all or any portion of their shares in Ivanplats Holding or Ivanplats (if they wish to dispose of such shares), at the same price and on the same terms and conditions as those offered by such third party.

BEE Co may not, without the prior written consent of Ivanhoe, dispose of any of these shares in Ivanplats (or loans to Ivanplats) until it has discharged all of its obligations pursuant to the B-BEE transaction finance agreements and the eighth anniversary of the date on which the B-BEE transaction was implemented (being 26 June 2014). Unless Ivanhoe provides its prior written consent, BEE Co may not dispose of its shares in Ivanplats unless it is to an HDSA and the disposal will not prejudice the on-going validity of any mineral right of Ivanplats.

4.4 Consumer Agreements

4.4.1 Industrial Water Use Agreements

During February 2016, the Mogalakwena Local Municipality published a request for expressions of interest to be submitted to it by interested parties to participate in the development of a new wastewater treatment works and to participate in the re-use of the treated effluent volumes from the Northern Drainage Zone. The Municipality indicated in its call for expressions of interest that its intention is to enter into agreements with a number of water users to support the capital funding of the project and to facilitate the effective re-use of the treated effluent. During April 2016, Ivanplats submitted its expression of interest to the Mogalakwena Local Municipality, indicating its interest in participating in the project. The Municipality identified Ivanplats as a preferred bidder to provide financial assistance for the project, and to take up the treated effluent generated by the WWTW.

Over the period from 2016 – 2018, the Municipality commenced construction of the WWTW, but did not timeously conclude definitive agreements with Ivanplats to access the funding Ivanplats was willing to advance. This occurred despite the fact that the Municipality experienced significant financial difficulties over this time period, and eventually had to cease construction of the WWTW during 2018.

Over the period from 2018 – January 2022, Ivanplats and the Municipality engaged in further discussions to negotiate the necessary agreements. These included conclusion of the following preliminary agreements, namely:

- a Memorandum of Agreement in May 2018, in terms of which Ivanplats agreed to advance financial assistance to the Municipality in an amount of approximately R37 million, and the Municipality granted Ivanplats a right of first refusal in respect of the effluent generated by the WWTW;
- an Off-take Agreement in December 2018, in terms of which Ivanplats would contribute approximately R214 million to fund the completion of the WWTW, in exchange for which the Municipality undertook to supply Ivanplats with up to 10 megalitres per day of treated effluent, at a price of R5 per kilolitre. However, this agreement was subject to certain suspensive conditions, which were not fulfilled timeously due to a number of factors.

Eventually, in January 2022, Ivanplats and the Municipality concluded two new, unconditional agreements, namely:

- a Sponsorship Agreement, in terms of which Ivanplats undertook to complete the partially constructed WWTW for and on behalf of the Municipality, at no cost to the Municipality. In terms of the agreement,
  - Ivanplats will now take over both financial and technical responsibility for construction of the facility, including appointing contractors to finish the works and supplying the necessary materials;
  - Ivanplats anticipates spending approximately ZAR 215 million (USD 14 million) to complete the works, but this figure may be increased if necessary;

  - Ivanplats' financial contribution takes the form of a sponsorship in favour of the Municipality, so Ivanplats will not record it as a loan or require repayments in respect of this amount;
  - The Municipality grants Ivanplats a right of first refusal in respect of the treated effluent generated by the facility, up to a volume of 10 megalitres per day;
- The second agreement is an off-take agreement, in terms of which:
  - Ivanplats will be entitled to take off at least 3 megalitres per day of treated effluent from the facility, up to a maximum of 10 megalitres per day. The exact volume of water is dependent on Ivanplats' requirements, as well as the availability of fresh water to be supplied to households by the Municipality (which, in turn, is the source of the effluent).
  - The price at which Ivanplats will purchase the treated effluent is equal to R5 per kiloliter (escalated annually by the inflation rate), or the reasonable, audited cost of managing and maintaining the facility, whichever is the higher;
  - The duration of the off-take agreement is 32 years from the date on which the Platreef mine reaches steady state production (which date will be certified by our engineers); and
  - Ivanplats will assist the Municipality to build its own capacity to maintain and operate the facility.

The agreements were designed so as to impose no financial obligations on the Municipality and to avoid the need for following government procurement processes, which have in the past caused a variety of delays and difficulties. It also affords Ivanplats the ability to de-risk the project by controlling the technical aspects of the construction. However, Ivanplats will seek to set off its financial contributions against taxes and/or its obligations under its Social and Labour Plan.

4.4.2 Electricity Supply Agreements

Ivanplats is currently in discussions with Eskom SOC Limited ("Eskom") to agree on the terms and conditions on which Eskom is willing to supply electricity (100 MVA) to Ivanplats. Ivanplats accepted the Budget Quotation, which secures the 100 MVA power supply to the mine, including cost for the design and land acquisition. Discussions for the negotiation of final agreements are underway, and Ivanplats anticipates that the agreements will be concluded by 19 April 2022. Ivanplats will enter into a "Self-Build Agreement" with Eskom in terms of which Ivanplats will construct the infrastructure itself, based on the plans and designs prepared by Eskom (rather than wait for Eskom to construct the powerlines in accordance with Eskom's time lines). Ivanplats will further enter into an "Electricity Supply Agreement" which regulates the terms and conditions on which it will be supplied with electricity. These draft agreements are currently being reviewed by both parties.

4.5 Royalties and Encumberances

Royalty

In terms of section 25(2)(g) of the MPRDA, the holder of a mining right must pay a royalty to the State in terms of any relevant law. The royalty payable to the State is determined in terms of the Mineral and Petroleum Resources Royalty Act, 2008 ("Royalty Act"). In terms of section 2 of the Royalty Act, a royalty is payable for the benefit of the National Revenue Fund in respect of the transfer of a mineral resource extracted from within the Republic of South Africa.

Section 3 of the Royalty Act distinguishes between a "refined mineral resource" and an "unrefined mineral resource", and different formulae for calculating royalties are prescribed in respect of refined and unrefined mineral resources, respectively. The royalties are calculated by multiplying the gross sales of minerals by the mining right holder during the year of assessment by the percentages determined in section 4(1) and 4(2) of the Royalty Act in respect of refined and unrefined mineral resources. The percentages are calculated as follows:

Refined Mineral Resources

0.5+(earnings before interest and taxes)/(gross sales in respect of refined mineral resources×12.5)×100

subject to a maximum of 5%

Unrefined Mineral Resources

0.5+(earnings before interest and taxes)/(gross sales in respect of refined mineral resources×9)×100

subject to a maximum of 7%

The Royalty Act contains detailed, anti-avoidance provisions which stipulate how the variables in the abovementioned formulae should be calculated and/or adjusted, for example, in respect of transactions not taking place at arm's length, and exchange transactions (or donations) where no purchase price is paid in respect of the transfer of a mineral. Schedules 1 and 2 to the Royalty Act further contain provisions which determine when a mineral will be considered to be "refined" or "unrefined" for purposes of the calculation of the royalty.

The obligation to pay the State royalty, and interest on any late payments, is also recorded as a condition of Ivanplats' mining right (clauses 5 and 6 thereof).

Restriction on Transfer

In terms of section 11(1) of the MPRDA, a prospecting right or mining right or an interest in any such right, or a controlling interest in a company or close corporation, may not be ceded, transferred, let, sublet, assigned, alienated or otherwise disposed of without the written consent of the Minister, except in the case of change of controlling interest in listed companies. This provision constitutes an effective restriction on the transfer or securitisation of any shares in Ivanplats itself, as well as the cession, transfer, letting, subletting etc. of the mining right or any share/interest in the mining right, unless ministerial consent is obtained. The purpose of this section is to prevent circumvention of the requirements of the MPRDA with regard to the ability of transferees to comply with the financial, technical, environmental and empowerment obligations of a mining right holder.

The statutory restriction contained in section 11(1) of the MPRDA is repeated clause 9.1 of the Ivanplats mining right, and clause 9.2 of the Ivanplats mining right further provides that any transfer, encumbrance, cession, letting, sub-letting, assignment, alienation or disposal of the mining right or any interest therein or any share or any interest in Ivanplats, without the consent of the Minister of Mineral Resources, is of no force, no effect and is invalid.

Other Restrictions

The Ivanplats mining right contain the following further restrictions:

- Mining operations in the mining area must be conducted in accordance with the Mining Work Programme and any amendment to such Mining Work Programme and an approved Environmental Management Plan (clause 7.2);
- The Holder shall not trespass or enter into any homestead, house or its curtilage nor interfere with or prejudice the interests of the occupiers and/or owners of the surface of the Mining Area except to the extent to which such interference or prejudice is necessary for purposes of enabling the Holder to properly exercise the Holder's rights under this mining right (clause 7.3); and
- It is a condition of the granting of the mining right that the Holder shall dispose of all minerals and/or products derived from the exploitation of the minerals at competitive market prices which shall mean in all cases, non-discriminatory prices or non-export parity prices. If the minerals are sold to any entity, which is an affiliate or non-affiliated agent or subsidiary of the Holder, or is directly or indirectly controlled by the Holder, such purchaser must unconditionally undertake in writing to dispose of the minerals and any products produced from the minerals, at competitive market prices (clause 8).

4.6 Key Points and Significant Risk Factors

Ivanplats has identified the following key points regarding the property:

- Rietfontein 2 KS is critical as some of the necessary infrastructure to support underground mining activities on Turfspruit 241 KR and Macalacaskop 243 KR, such as a tailings dam, may be located within the Rietfontein 2 KS area.

- The Right MR was granted in favour of Ivanplats on 30 June 2014, and notarially executed on 4 November 2014, signifying the formal activation of the MR. The MR will continue to be in force until 3 November 2044. The MR allows a company to mine and process minerals optimally from the mining area for a maximum period of 30-years, which may be extended upon application for further periods, each of which may not exceed 30-years at a time.
- Surface rights within the areas of the Rietfontein 2 KS, Macalacaskop 243 KR, and Turfspruit 241 KR farms belongs to the national government and holders of informal land rights. There is a reasonable expectation that land access and provision of land for infrastructure development for any proposed mining activity will be achievable following appropriate negotiation and compensation payments.
- Other than the known claim by the Mamashela Community, no additional information was provided to confirm what other communities may lawfully occupy the Rietfontein 2 KS farm. Should infrastructure related to future mining operations be sited in the farm area, studies will be required to identify such communities.
- A royalty will be payable to the South African Government on production; this will be determined on whether the mined product will be classified as either a refined (capped at 5%), or unrefined (capped at 7%) material.
- Exploration and mining activities to date have been conducted within the regulatory framework required by the South African Government.
- Based on information discussed in Section 20 of the Report, collection of baseline environmental data has commenced. The current state of knowledge on environmental and permit status for the Platreef Project supports the declaration of Mineral Resources. Additional permits will be required for project development.
- A gazetted land claim has been lodged over the Rietfontein 2 KS farm; information provided to Ivanhoe by the DRDLR indicates a non-gazetted claim by the Mokopane Tribe over the area covered by PR MPT76/2007PR.
- Through its actions to date, Ivanhoe has shown its understanding of, and accepts the importance of, proactive community relations, and is continuing to liaise with representatives of the local communities.
- To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

There are two significant permitting risks to project development: finalisation of long-term surface lease agreements with the South African Government and affected communities, and the finalisation of the new prospecting right application in respect of Rietfontein.

4.7 Comments on Section 4

Ivanplats manages the ownership and title for Platreef with full time personnel including in-house legal expertise and this assists in providing important understanding of the project and in mitigating the risks around these issues. in the opinion of the QP Bernard Peters, the information discussed in this Section supports the Platreef 2022 FS and it is reasonable to rely on the information supplied by Ivanplats.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Platreef Project site is located approximately 280 km north-east of Johannesburg. Year-round access to the site is by paved, all-weather national N1 highway (N1) to Mokopane (formerly Potgietersrus). From Mokopane the access continues as a paved, all-weather national N11 highway (N11). The N11 highway is a two-lane tarmac road suitable for heavy loads year-round.

The closest major international airport is the OR Tambo International Airport, about a three-hour drive from Mokopane, and the regional hub is at Polokwane (formerly Pietersburg) 30 km to the north of Mokopane.

The Limpopo Province has a developed rail network, connecting with lines that lead to Zimbabwe in the north, Maputo in Mozambique to the east and south to Gauteng. The closest railhead to the Platreef Project is in Mokopane.

5.2 Climate

The climate is semi-arid, with precipitation occurring as rain. Average annual rainfall is around 500 mm. Over 90% of the annual rainfall occurs between the months of October and March. The highest monthly averages typically occur in November and December. Golder Associates (Golder, 2014) noted the highest monthly rainfall was 127 mm in November 1977.

High daily temperatures occur throughout the year; the average maximum monthly temperatures range from 21–30°C, with a maximum recorded temperature of 39°C. During the winter months, the temperature may drop to around 0°C, although freezing is extremely rare. The average minimum monthly temperature ranges from 6–18°C.

Golder, 2014 noted that at Mokopane winds originate from the north (17.5% of the time) and from the north–north-west (14.5% of the time). Wind speeds are low to moderate, with a low percentage (19.46%) of calm conditions (<1 m/s).

It is expected that any future mining operations will be able to be conducted year-round.

5.3 Local Resources

Electrical power, potable water, fuel supply, accommodation, communication services, and other infrastructure components are available in Mokopane. The Mokopane town centre is approximately 11 km from the Platreef Project site. The main line of the national railroad system passes approximately 6 km east of the site.

A business survey showed that a larger number of businesses are located near the Platreef Project area. Most of these businesses specialise in building and construction (20%), providing services (12%), and catering (10%). Typical to the area, most businesses are very small, employing less than five people. Just less than a third of all business enterprises indicated that they provide some kind of engineering service; of these, the majority (59%) provide civil engineering services such as construction and earthworks. The Ivanplats Social and Labour Plan (SLP) provides clear guidelines on how these businesses are to be incorporated in the overall project both during construction and LOM.

5.4 Local Labour

Mining activity is moderately prevalent within a 100 km radius. A large potential labour force resides within close proximity of the site. A skills survey was conducted, and a database of available labour was developed. The majority of individuals who registered on the local labour database are unemployed, although most of them were previously employed and have some workplace experience. During the skills survey, it was determined that only a small number of individuals interviewed, were or still employed in the mining sector. The Ivanplats SLP makes provision for extensive training programmes to train the local communities to develop the necessary skills. Skilled trade positions and professional staff will have to be recruited from outside the area.

5.5 Infrastructure

5.5.1 Power Supply

This Section of the report discusses the current status of power supply to the area.

The South African electricity utility Eskom Holdings SOC Limited (Eskom) continues to implement rotational load shedding due to breakdowns in it's generating fleet. However, the Medupi power station was brought into full commercial operation in July 2021. The Kuslie power plant currently under construction has connected 4 of its 6 x 800 MW generating units to the national grid.

During 2011, Ivanplats submitted an application to Eskom for the supply of bulk power to the Platreef Project. The power application was for a 3 Mtpa underground mine and the maximum demand was estimated at 70 MVA. The Eskom desktop feasibility study phase for the Platreef Project was completed.

Ivanplats has continued in discussion and recently requested that Eskom complete the budget quote study for 100 MVA supply based on "self build" option that considers a premium supply. The design package has been completed by Eskom and the construction is being undertaken by the project team. The latest forecast energisation date of the Platreef Eskom incoming substation is Q3' 2023.

The Platreef Project power requirement for a 5.2 Mtpa underground mine has been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats is in the process of notifying and requesting from Eskom the 20 MVA additional power demand for the Platreef Project.

Electrical power is currently supplied to the mine from the Mahwelereng 33/11 kV substation, which is located 7 km from the Platreef Project site.

The agreement for a 5 MVA construction power supply was concluded and is currently in use. Eskom have agreed to increase the NMD on the line to 8 MVA, but this upgrade had not yet occurred at time of writing.

The 8 MVA supply is sufficient for current operation and construction power and any power requirements exceeding the supply of temporary construction power will be supplied by diesel generated sets.

The mine is also now legally permitted to self-generate 100 MW of power to supplement the Eskom supply and options in this regard are being investigated by Ivanplats.

5.5.2 Water Supply

The Limpopo province and the Mokopane area in particular, are considered to be particularly water-poor resource areas, and various studies were commissioned to determine the most likely water supply sources for the project.

This Section of the report discusses the availability of water and the current water supply to the area.

Ivanplats has an approved water use licence that permits the extraction of ground water for construction purposes and for dewatering of the underground workings.

Ivanplats has recently concluded and offtake agreement with the Mogalakwena Local District Municipality which secure a supply of water from the local Masodi Water Treatment Works. The agreement is for an initial supply of 3 Ml/d rising to 10 Ml/day.

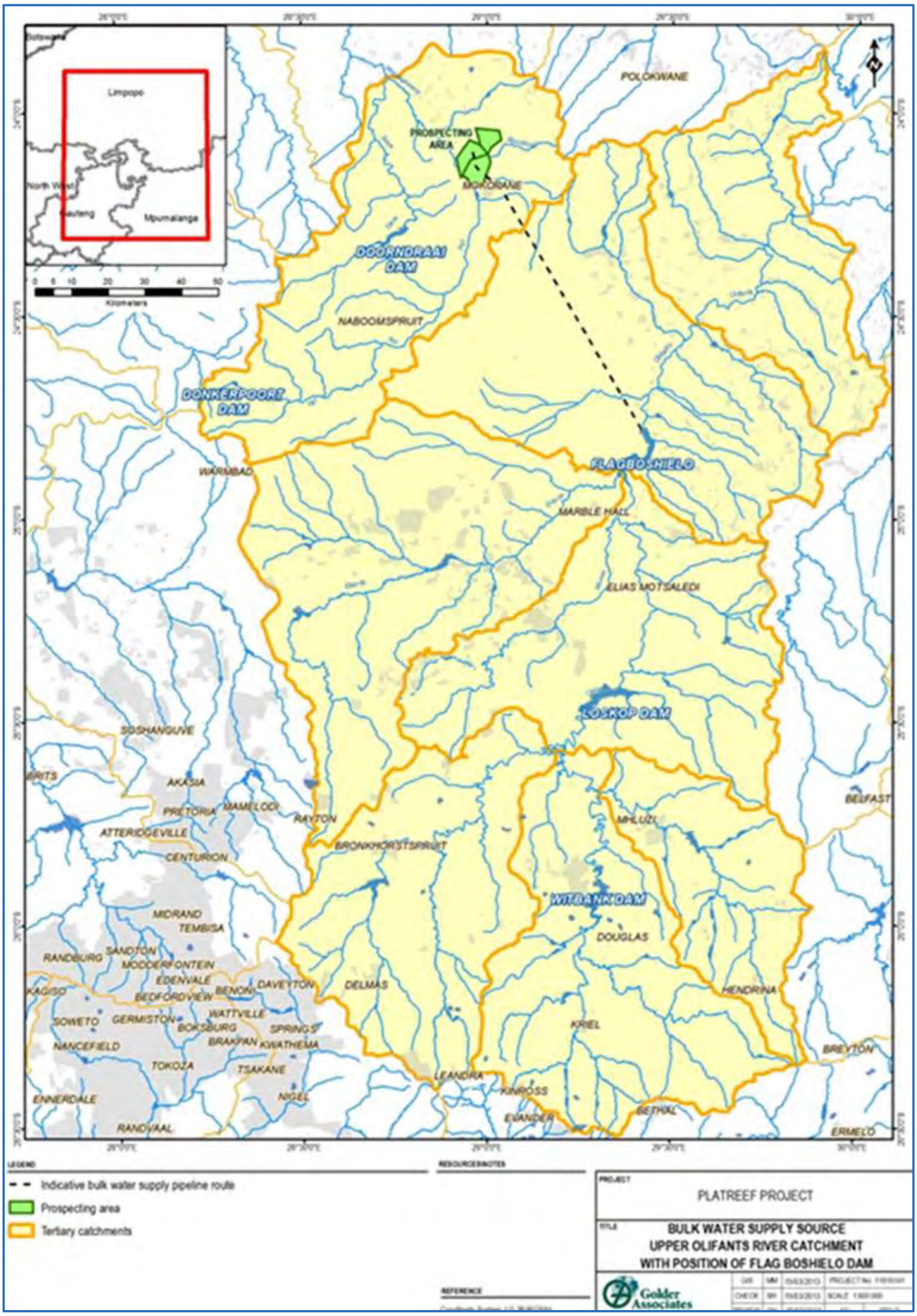
Ivanplats will contribute financially to the construction of the WWTW and is expected to receive water from this source in Q1'2024

Ivanplats is a participant in the Olifants River Water Resource Development Project (ORWRDP), which is designed to deliver water for domestic and industrial (mining) purposes to the Eastern and Northern Limbs of the Bushveld Complex. Ivanplats is also a member of the Joint Water Forum (JWF), which facilitates and coordinates discussions with the various participants in the water scheme.

Under the ORWRDP, a pipeline is to be constructed between Flag Boshielo dam on the Olifants River to Pruissen and from there to the north of Mokopane including the Platreef Project and other projects (Figure 5.1). Ivanplats' continued participation will require contributions to the costs of pipeline construction. These costs will be in relation to the number of participants in the final agreement.

The Department of Water and Sanitation (DWS) has stated that all water for the Northern Limb (including any potential mining operation on the Platreef Project) would be supplied through the ORWRDP. A number of possible water sources to augment the supply system have been investigated. The sources investigated include excess mine water from the Witbank coalfields and water transferred from the Vaal River system.

Figure 5.1 Location Plan Flag Boshielo Dam and Proposed Water Pipeline



Ivanhoe, 2013

5.5.3 Access Roads

Access from Mokopane to Johannesburg, Polokwane, and Rustenburg (for concentrate delivery) is via the newly upgraded N1 highway. The Platreef Project site is located approximately 11 km north–north-east of Mokopane and is accessed via the N11, a single-carriageway public highway with a bitumen surface.

Accelerated mining developments and envisaged further expansions to the north of Mokopane have led to an increase in pressure on existing infrastructure in the area and specifically on the N11 at and through Mokopane. The N11 is also the only feasible road to and from the Platreef Project.

The South African National Roads Agency Limited (SANRAL) is considering two options with regards to the N11 highway:

- Upgrade the existing road through Mokopane, to cater for the increased traffic volumes.
- Build a reroute of the N11, exiting the N1 north of Mokopane and entering the existing N11 approximately 5 km north of the Platreef Project area.

5.6 Physiography

The Rietfontein 2 KS, Macalacaskop 243 KR, and Turfspruit 241 KR farms are located in a broad valley on flat terrain with a gradual westerly slope. There is very little topographic relief on the farms; however, to the east and west of the farms, semi-parallel, north–south trending high ridges flank the valley floor. A portion of the eastern ridge system trends onto the Rietfontein 2 KS farm, adjacent to Turfspruit 241 KR. Figure 5.2 is a photograph taken in the Platreef Project area illustrating the general topography.

The elevation on the farms ranges from a maximum of about 1,140 m above sea level (masl) in northern Turfspruit 241 KR to about 1,060 masl on Macalacaskop 243 KR.

The land on the farms has been disturbed by settlements and farming. Subsistence farming and urban development covers the majority of all the farms. Some land has been allowed to lie fallow and is being reclaimed by bush, comprising shrubs and small trees. There are no remnant forests or other significant vegetation.

Figure 5.2 Platreef Project Physiography

Ivanhoe, 2012; Drill rigs show scale. Rigs are testing Zone 1.

5.7 Sufficiency of Surface Rights

There is sufficient suitable land area available within the MR licences for any future tailings disposal, mine waste disposal, and installations such as a concentrator and related mine infrastructure.

6 HISTORY

6.1 Previous Work

During the 1970s, regional exploration was undertaken over the Platreef mineralised zone (the Platreef) by Rustenberg Platinum Holdings Limited (Rusplats), a wholly owned subsidiary of Anglo-American Platinum Corporation (Amplats). Rusplats reportedly drilled several widely spaced drillholes along the Platreef on Turfspruit 241 KR and Macalacaskop 243 KR farms. This drilling followed-up earlier work by the predecessor of Amplats during the 1960s. No data from either of these programmes were available for the Platreef 2022 FS.

Ivanhoe acquired a prospecting permit for both Turfspruit 241 KR and Macalacaskop 243 KR farms in February 1998, and subsequently Ivanhoe entered into a JV with Atlatsa over the Rietfontein 2 KS farm in 2001.

Work completed by Ivanhoe consists of geological mapping, airborne and ground geophysical surveys, limited trenching, percussion drilling over the Platreef sub-crop, core drilling, petrography, density determinations, geotechnical and hydrogeological investigations, metallurgical testwork, preliminary engineering and design studies. These studies and mineral resource estimates were performed during the period 2001–2015.

The initial exploration focus was on delineation of mineralisation that could support open pit mining. From 2003–2007, Ivanhoe undertook studies involving concentrator and smelter options, metallurgical testwork, and conceptual mining studies that considered open pit scenarios. Results of this work indicated that the mineralisation on the Turfspruit 241 KR and Rietfontein 2 KS farms was more likely to support a mining operation than the mineralisation on the Macalacaskop 243 KR farm.

Following news of AfriOres' success in deep drilling to the north at Akanani (Witley, 2006), Ivanhoe commenced a deep drilling programme in 2007, to test for mineralisation down-dip within the Turfspruit 241 KR farm and to investigate the continuity and grade in an area targeted as having potential to be mined by underground methods. The drill programme identified the area of mineralisation within the UMT deposit currently known as the Flatreef, and supported estimation of mineral resources amenable to underground mining methods.

Mineral Resource estimates for the underground deposit were updated multiple times in internal documentation between 2007–2011, and the 2011 update for mineralisation considered amenable to open pit and underground mining methods was publicly disclosed in the technical report entitled 'Ivanplats Limited, Platreef Project, Limpopo Province, Republic of South Africa, NI 43-101 Technical Report on Mineral Resources', with effective date of 20 August 2012, (www.sedar.com, Parker et al., 2012). A mineral resource estimate update assuming selective and mass mineable underground mining methods was prepared in April 2013 and estimates for the Bikkuri Reef were prepared in May 2013. In March 2014 Ivanhoe completed the Platreef 2014 PEA (www.sedar.com, Peters et al., 2014) and in January 2015 the Platreef 2014 PFS was completed (www.sedar.com, Peters et al., 2015). In June 2016 Ivanhoe issued a Mineral Resource update in the Platreef 2016 Resource Technical Report (www.sedar.com, Peters et al., 2016). The previous Technical Report was the Platreef 2017 Feasibility Study (www.sedar.com, Peters et al., 2017).

6.1.1 Open Pit Resource Models

A resource model supporting Open-Pit Mineral Resources was completed in 2003. Ivanplats is no longer considering the open-pit option as part of their current development plans. A detailed description of the open-pit resource was included in the September 2012 Technical Report (Parker et al., 2012).

6.1.2 Mass Mining Resource Models

A resource model supporting an underground mass mining option was completed in 2011. Significant changes to the geological interpretation have occurred since 2011, and the 2011 UMT-MM model is no longer considered valid. A detailed description of the UMT-MM resource was included in the September 2012 Technical Report (Parker et al., 2012).

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Platreef Project is hosted within the Palaeoproterozoic (2.06 Ga) Bushveld Igneous Complex (BIC), which is the largest of the known layered igneous intrusions, covering an area > 65,000 km². The BIC hosts up to 75% of the world's platinum resources (Naldrett et al, 2009).

The BIC includes an early bimodal volcanic sequence (the Rooiberg Group) that is followed by an intrusive layered series of ultramafic and mafic units known as the Rustenburg Layered Suite (RLS) and the Lebowa Granite and Rashoop Granophyre Suites. The RLS is 7 to 8 km thick and ranges in composition from dunite to diorite.

Hall (1932) divided the RLS into 5 zones in descending order:

- Upper Zone (UZ) — Gabbroic succession.
- Main Zone (MZ) — A succession of gabbronorites with occasional anorthosite and pyroxenite bands.
- Critical Zone (CZ) — The Lower Critical Zone (LCZ) consists of orthopyroxenitic cumulates, and the Upper Critical Zone (UCZ) comprises packages of chromitite, harzburgite, pyroxenite, norite, and anorthosite. The CZ hosts PGE–Au–Ni–Cu and chromite deposits in several different chromitite layers known as reefs. The most significant are the Merensky Reef and the Upper Group 2 (UG2) Reef of the Eastern and Western Limbs. These range on average from 0.4–1.5 m in thickness and the contained PGE (Pt, Pd, Rh, Au) content typically ranges from 4–10 g/t (Cawthorn, 2005).
- Lower Zone (LZ) — Upper and lower peridotites separated by a central harzburgite.
- Marginal Zone (MZN) — Norites with variable proportions of accessory clinopyroxene, quartz, biotite and hornblende, indicating magma contamination from the underlying metasediments. This unit is not always present.

In the East and West Limbs of the BIC, the RLS was intruded into the Magaliesberg Formation of the Proterozoic Transvaal Supergroup. In the North Limb, the RLS intrudes progressively older country rocks northward (Magaliesberg Formation, Malmani Subgroup and Duitschland Formation). Figure 7.3 shows the Project geology projected to surface. The locations of Zones 1 to 5 referred to in the following geologic discussion are shown in Figure 7.4.

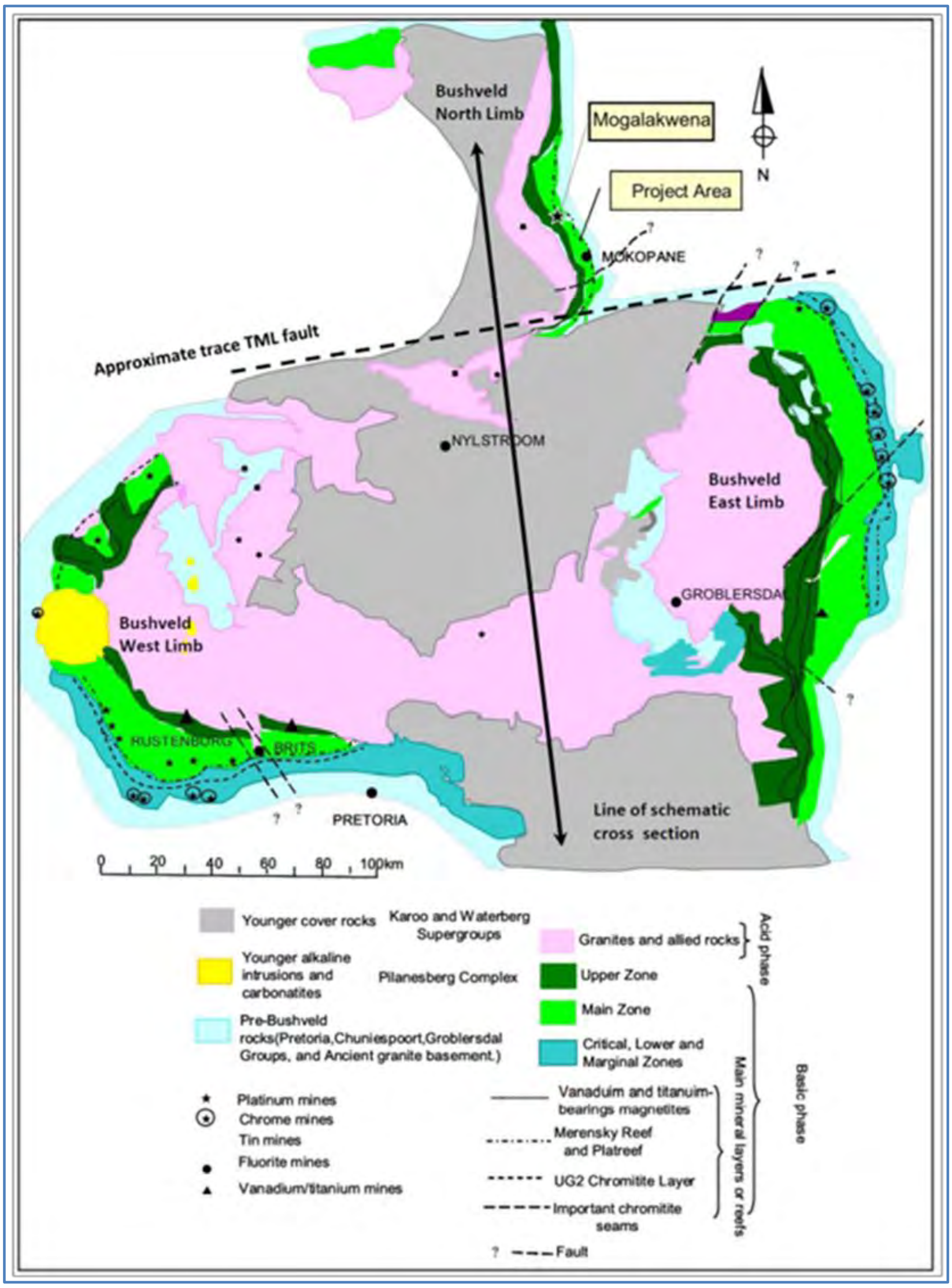
Figure 7.1 provides a location and regional geology map for the BIC. Figure 7.2 provides a diagrammatic cross-section through the BIC.

In the East and West Limbs of the BIC, the CZ includes the Merensky Reef and UG2 chromitite that are exploited for PGE mineralisation. In the North Limb of the BIC, the mineralised horizons have been referred to as the Platreef. The North Limb hosts the Platreef Project.

7.2 Project Geology

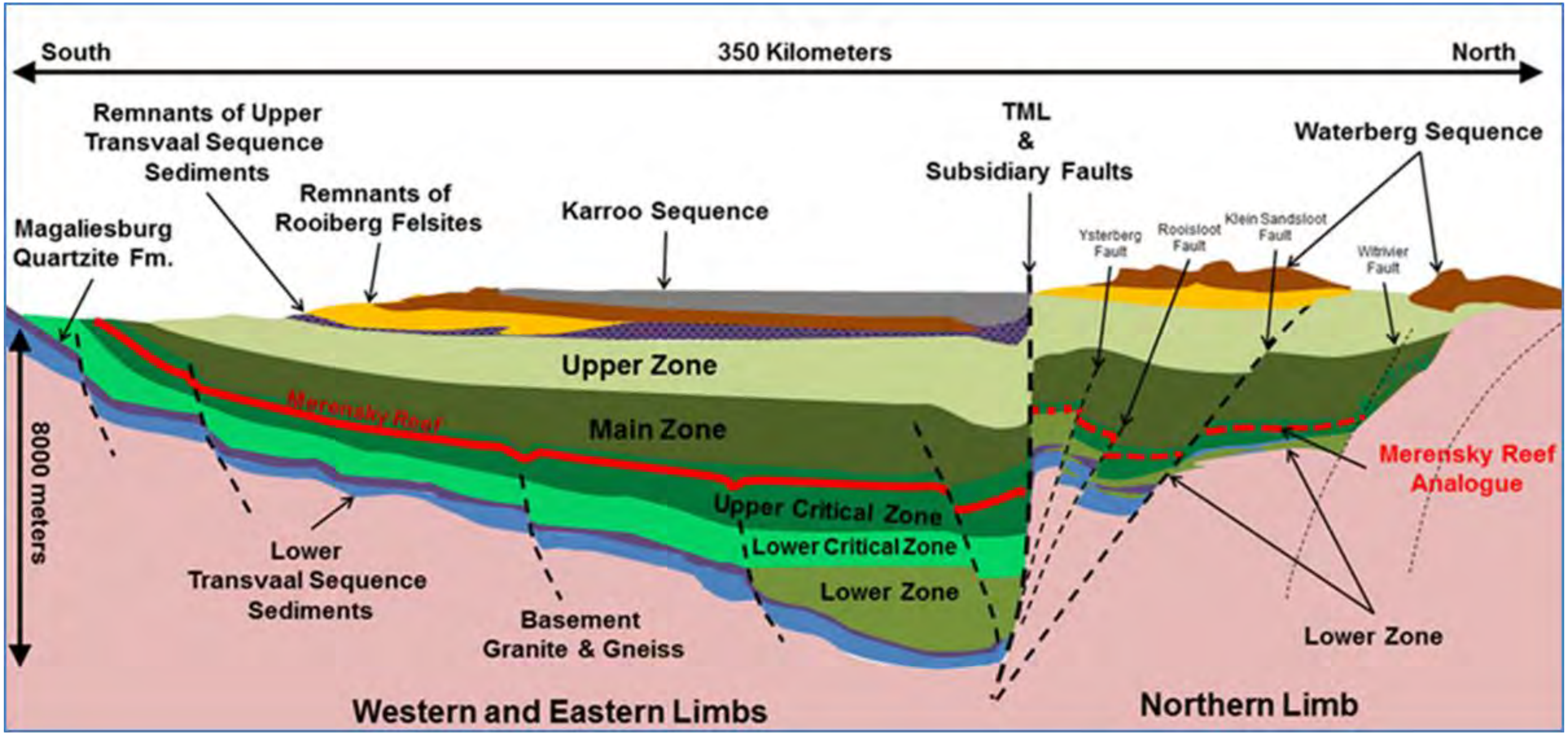
Figure 7.3 shows the Project geology projected to surface. The locations of Zones 1 to 5 referred to in the following geologic discussion are shown in Figure 7.4.

Figure 7.1 Regional Geological Plan of The Bushveld Complex



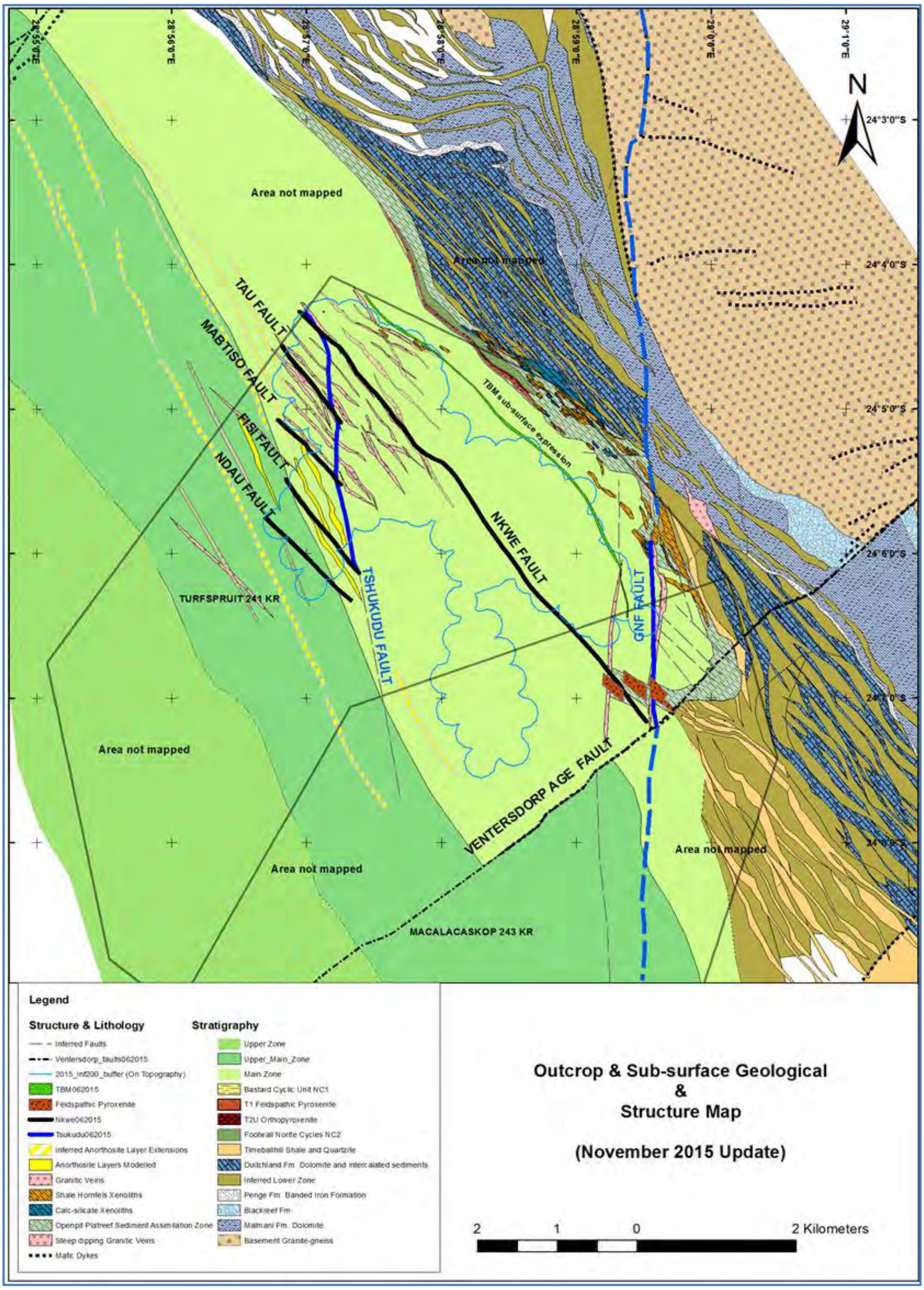
Modified after Viljoen and Schürmann, 1998; section line represents location of section in Figure 7.2

Figure 7.2 Schematic Cross-Section Through the Bushveld Igneous Complex



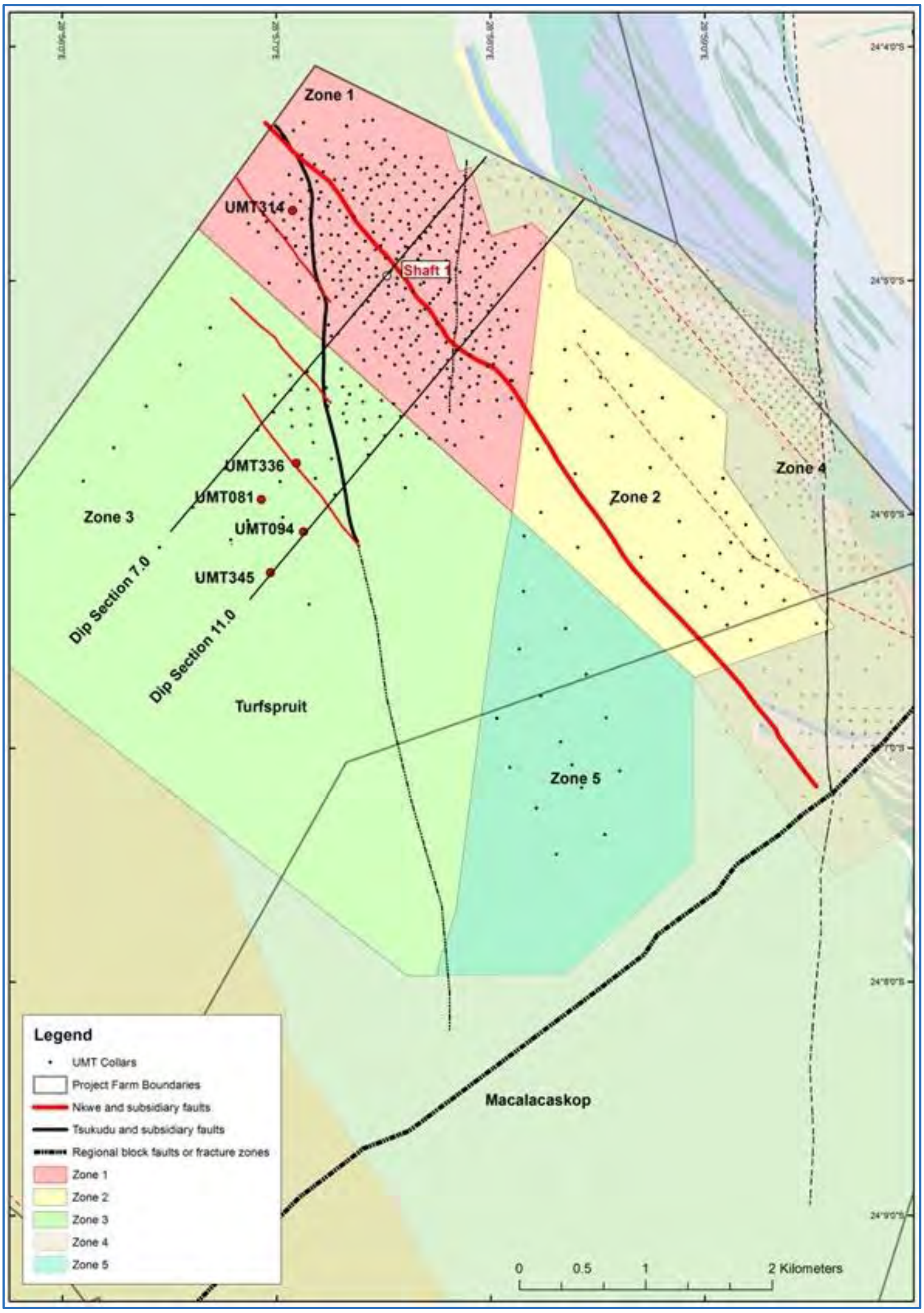
Ivanhoe, 2012; modified after Kruger, 2005. Figure is schematic and not to scale. Section line illustrated is shown in Figure 7.3.

Figure 7.3 Project Geology Plan



Ivanhoe, 2016

Figure 7.4 Project Exploration Zones Plan



Ivanhoe, 2016 (After Brits and Nielsen, 2015).

7.2.1 Introduction

The following is summarised from Grobler et al. (2016). Since 2013, Ivanhoe has modified the stratigraphic framework for the Project and revised geological descriptions and interpretations compared to those presented in previous technical reports. The interpretation is based on drill core interpretations and core relogging, geophysical surveys, and geochemical data.

Detailed re-logging of drill core was completed for intersections of the Turfspruit Cyclic Unit (TCU) and the footwall lithologies found stratigraphically below the T1 and T2 Reefs. The re-logging and structural interpretation enabled the recognition of continuous magmatic layering from within the MZ through the TCU as well as in the footwall. Further investigations focusing on metamorphic and metasomatic processes associated with the magma-sediment interaction zones are currently in progress at academic institutions.

7.2.2 Stratigraphic Correlations and Nomenclature

The magmatic strata of the Upper Critical Zone (UCZ) on the Project has locally been subdivided into different major magmatic cyclic units similar to what has been done for the eastern and western Bushveld. This is a refinement of the first attempt made in 2013 in correlating the TCU with the Upper CZ. The down-dip Zone 3 (Figure 7.4) on the farm Turfspruit 241KR is one of the few areas on the Northern Limb where undisturbed magmatic stratigraphy has been intersected, since the 1924 discovery of the Northern Limb by Dr Hans Merensky. The magmatic strata of the Upper Critical Zone (UCZ) on the Project has locally been subdivided into different major magmatic cyclic units.

- Norite Cyclic Unit 1 (NC1), uppermost cyclic unit includes the Bastard Reef equivalent;
- Turfspruit Cyclic Unit (TCU), includes the Merensky Reef equivalent;
- Norite Cyclic Unit 2 (NC2), Repetitive magmatic cyclical layering in footwall to TCU;
- UG2, Cyclic Unit (includes hanging wall pyroxenite/chromitite and footwall harzburgite);
- PNZ (Pyroxenite-Norite-Zone), homogenous medium-grained pyroxenite/norite with intermittent chromitite bands possibly representing part of the Lower Critical Zone (LCZ); this can include assimilated floor as clinopyroxenites or hornfels lenses;
- Lower Zone (LZ), Mafic and ultramafic magmatic units correlated with the Lower Zone of BIC.

Each cyclic unit consists of a regular sequence of norite to pyroxenite and olivine cumulates with sub-horizontal rhythmic layering (cycles) developed with varying degrees of cyclicity. Layering can be spectacularly developed with regular cyclic layers (going upward) of chromite-orthopyroxenite-norite-anorthosite (commonly in that order). It is within this sequence of cyclic stratigraphy that correlations of the UG2, Merensky (TCU) and Bastard cyclic units (NC1) were identified from core intercepts. Although these stratified layers are laterally contiguous, they display significant lateral facies variations. The facies variation can be attributed to magmatic processes and magma interaction with sedimentary xenoliths (Grobler et al., 2016).

In addition to the above described magmatic stratigraphy, undifferentiated lithologies have also been recognised on the project. The major occurrence is the Footwall Assimilated Zone (FAZ) that occupies a similar stratigraphic position as the NC2 in the well drilled Zone 1 area. The FAZ is a zone of intense magma-sediment interaction, which can also include the basal part of the T2 pegmatoid (part of the TCU). This unit can be well mineralised, but commonly displays irregular continuity of grades across the Project area (Grobler et al, 2016). Also rocks with similarities to Marginal Zone norites and pyroxenites have been identified on the project area by Yudovskaya et al (2013). A description of the major units on the Platreef Project follows.

Figure 7.5 shows a comparison between a clean magmatic stratigraphy (left column) and a magmatic stratigraphy including significant sediment interaction (right column).

The four major magmatic cyclic units are shown in Figure 7.6. The major cyclic units consist of a series of alternating leucocratic to mafic to ultramafic lithologies within the interval from the base of the UG2 Cyclic Unit to the contact of the Main Zone gabbronorite. The cyclicity is most recognisable within the down dip extensions of the Upper CZ located in Zone 3 on the Turfspruit farm (Figure 7.4).

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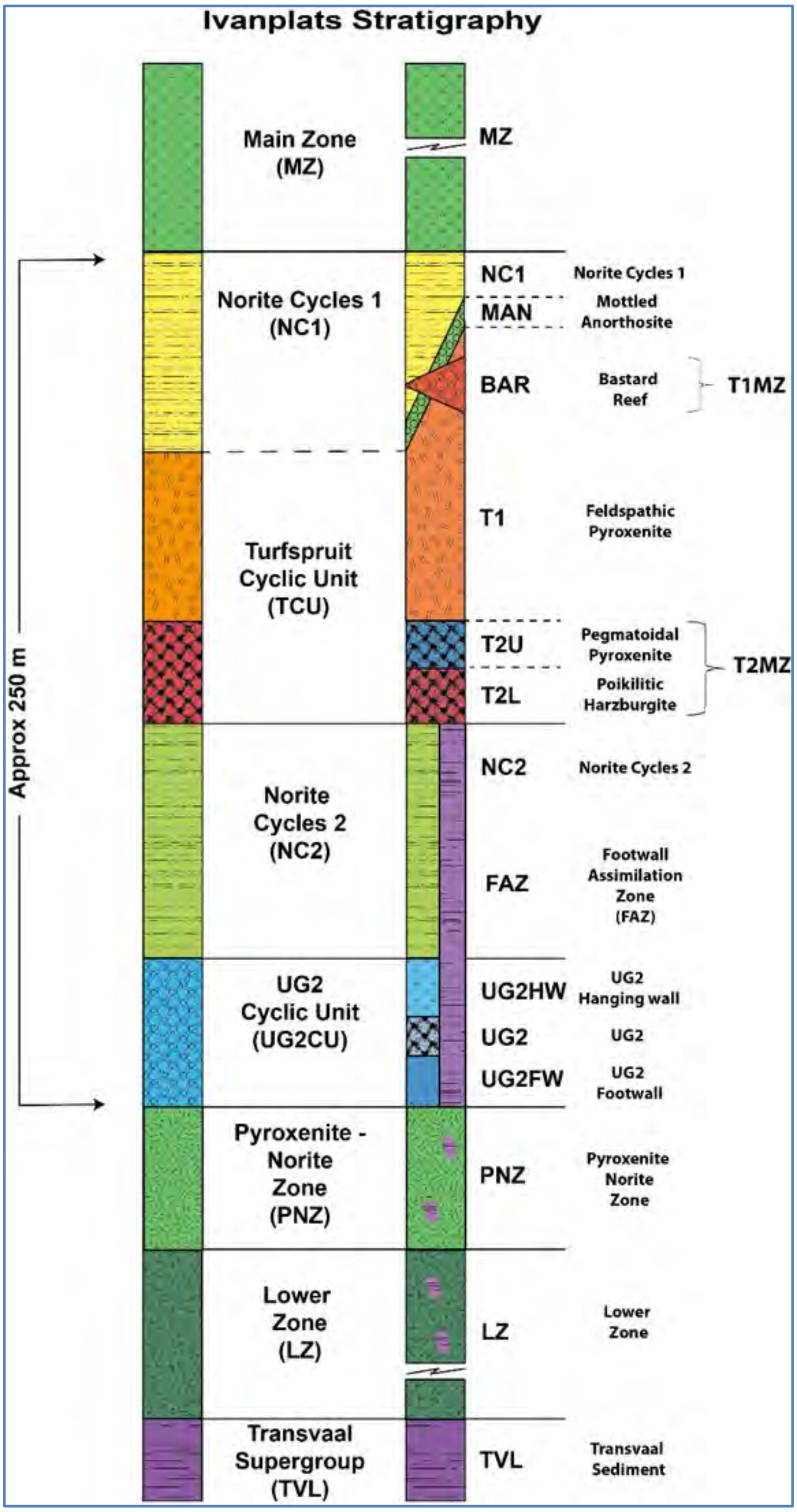
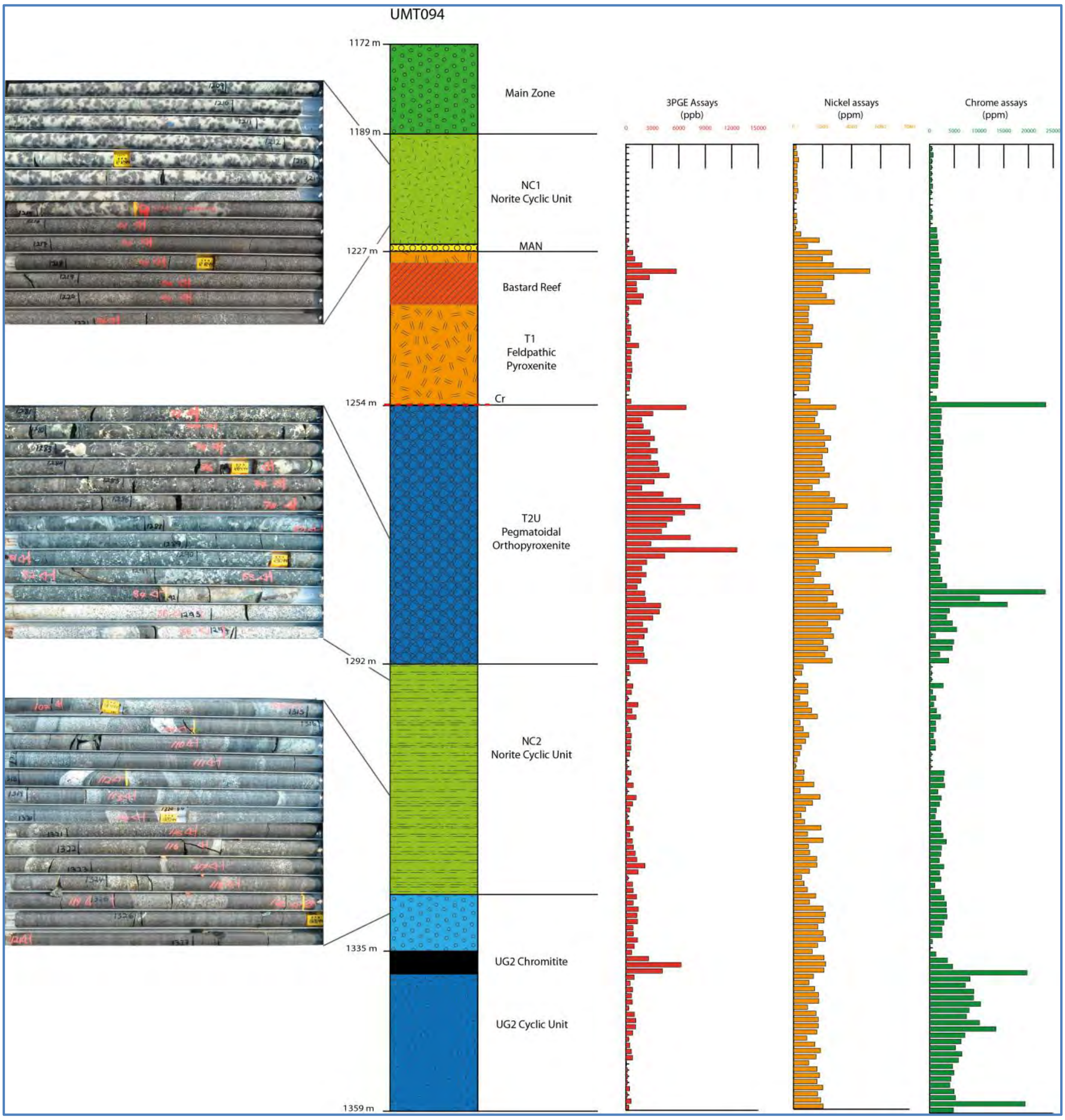
Figure 7.5 Platreef Stratigraphic Column



Ivanhoe, 2016

Figure 7.6 Example of Magmatic Cyclic Units from UMT094



Ivanhoe, 2016; See Figure 7.4 for location of UMT094; tickmarks on grid scale are at 3,000 ppb intervals for 3PGE, 2,000 ppm for nickel, 5,000 ppm for chromium.

7.2.3 Transvaal Supergroup

On the Turfspruit farm, the RLS intrudes shallow-marine to shelf-clastic metasedimentary rocks of the Duitschland Formation at the base of the Pretoria Group. The floor of the RLS appears to be close to the unconformity with platform carbonates of the Chuniespoort Group (Bekker, 2001).

7.2.4 Lower Zone (LZ)

The LZ consists of mafic and ultramafic magmatic units situated stratigraphically at the base of the Bushveld Complex. The LZ has been intersected by UMT drillholes in In the Project area, but the base of the LZ has not been observed. In the eastern extents of the Turfspruit area (see Figure 7.3) the LZ can be inferred as being intruded as inter-fingered sills, sub-parallel or transgressive to bedding and controlled by cross-cutting tectonics. The sills appear to have varying thickness along dip and strike, with the variability of mafic units ascribed to assimilation of varying amounts of country rock.

The layered ultramafic sequence predominantly consists of pyroxenite, dunite and harzburgite that form cyclic units, with varying thickness and transitional contacts. Disseminated chromite (of up to 10 vol %) associated with the olivine-bearing sequence generally marks the basal contact of these cycles.

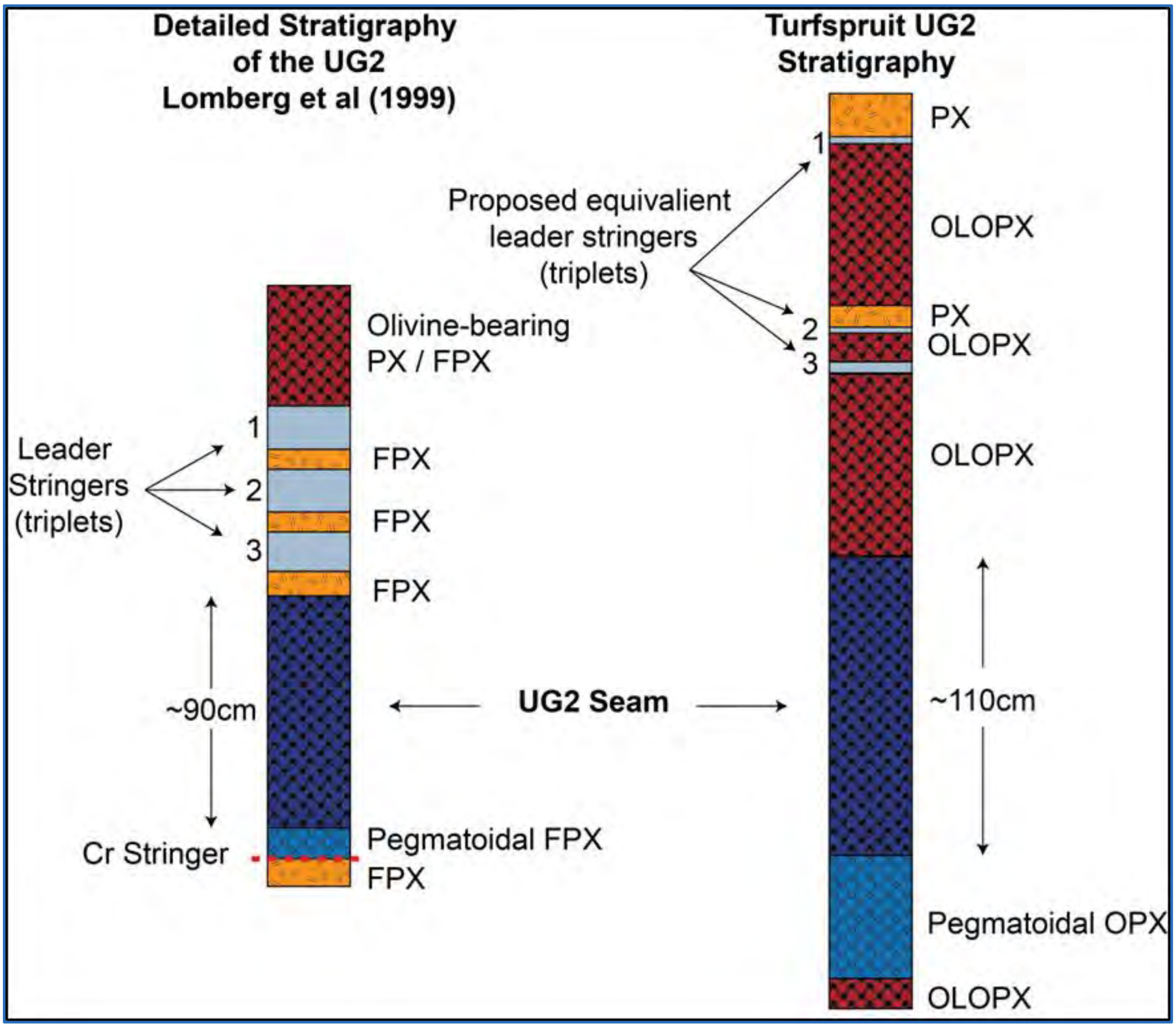
7.2.5 Pyroxenite-Norite-Zone (PNZ)

The PNZ generally occurs below the NC2 and the UG2 and is mostly represented by an undifferentiated fine- to medium-grained pyroxenite/norite with orientated elongated pyroxene crystals. Poorly-developed bands, stringers and disseminated zones of chromitite have been identified within the upper part of the PNZ in areas of low sediment contamination. These chromitite layers may possibly represent stratigraphic equivalents of the Middle Group and Lower Group chromitites found elsewhere in the Lower CZ of the Bushveld Complex.

7.2.6 UG2 Cyclic Unit

Investigations of drill core and assay data from UMT081 and UMT094 (Figure 7.4) in 2011 showed the possible existence of a UG2 reef equivalent below the T2 pegmatoid (Merensky equivalent) also found in these drillholes. The overall appearance, stratigraphic position below the T2 pegmatoid, and occasionally the presence of three thin chromitite stringers (UMT336 and UMT345) in the immediate hanging wall suggests that it may be a UG2 equivalent. Additional deep drilling within the down-dip extent of the property in Zone 3 (Figure 7.4) consistently intersected UG2 like layers in areas where limited sediment assimilation occurred. Figure 7.7 shows a comparison of the UMT336 UG2-like chromitite with published data (Nodder, 2015). Recent unpublished work by University of the Witwatersrand reports that chromitite from the Turfspruit UG2 analogue is poorer in Cr and richer in Ti compared to published UG2 data, but belongs to the same lineage of melt compositions in terms of its Mg/Al ratio. The Turfspruit orthopyroxene is very rich in Cr (Yudovskaya et al., 2013).

Figure 7.7 UG2 Equivalent from UMT336 Compared with Known UG2 Lithologies (Lomberg Et Al, 1999) From Elsewhere in the Bushveld Complex (Diagram Modified After Nodder, 2015)

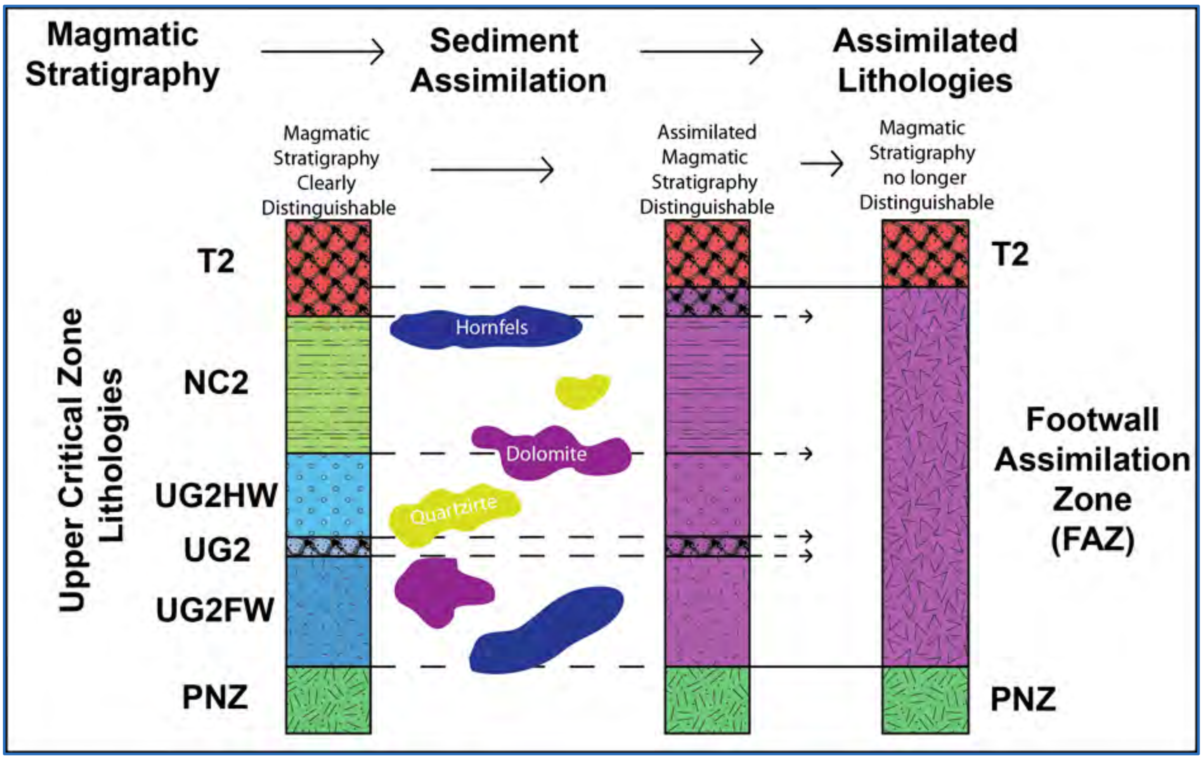


Ivanhoe, 2016

7.2.7 Norite Cyclic Unit 2 (NC2) and Footwall Assimilated Zone (FAZ)

The NC2 and the FAZ are the direct footwall to the mineralised T2 pegmatoid of the TCU. The NC2 is defined as magmatic cyclical layers in unconformable footwall contact with the TCU. This stratigraphic position is shared with the FAZ where the NC2 magmatic unit interacted with metasedimentary xenoliths (Figure 7.8). Interaction also occurs with the base of the T2 pegmatoid.

Figure 7.8 Footwall Assimilated Zone (FAZ)



Ivanhoe modified after Nodder, 2015

Magmatic cyclicity is well developed within the deeper (down-dip) portions of the UCZ and is only sporadically evident within the up-dip (near topographic surface) reaches, thus historically named the Platreef. The main influence controlling the cyclical nature of a unit is the amount of sediment interaction. The ratio of magma to sediment within a particular area, as well as the extent of assimilation (including melting) and nature of the sediments being assimilated all affect the magma, reducing cyclicity. For logging and modelling purposes, FAZ has been used to correlate the zones where the effect of excessive assimilation has made the logging of discrete magmatic strata impossible.

7.2.8 Turfspruit Cyclic Unit (TCU)

The TCU is the best-developed cyclical unit recognised in the Platreef Project and hosts the principal mineralised reefs. The TCU is in general subdivided into the following units in ascending order:

- T2 Lower (T2L) - Mineralised pegmatoidal harzburgite and/or pegmatoidal olivine-bearing pyroxenite locally with a chromitite stringer at its bottom contact;
- T2 Upper (T2U) – Mineralised pegmatoidal orthopyroxenite commonly with a thin (~0.5 cm) chromitite stringer marking its upper contact;
- T1 - Non-mineralised non-pegmatoidal medium-grained feldspathic pyroxenite with a generally non-pegmatoidal mineralised zone near its top (T1MZ).

The distribution of the T2U and T2L pegmatoidal units are controlled by the presence and volume of olivine. Together they form one stratigraphic layer similar to the Merensky Reef described in the main BIC.

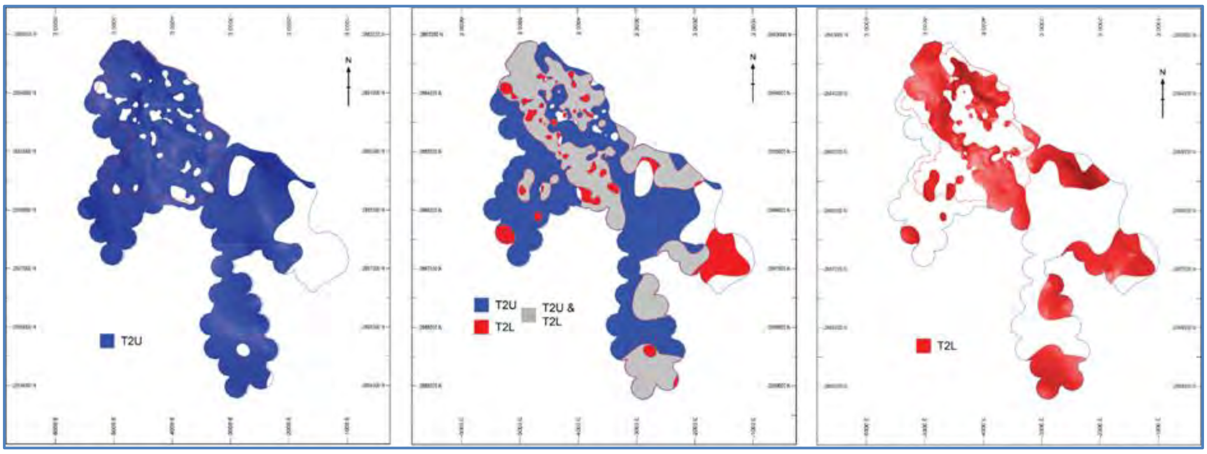
7.2.8.1 T2U and T2L

The T2U forms a contiguous mineralised layer overlying the variably developed T2L harzburgite. A coarse-grained, plagioclase-rich rock is formed where the T2 magma interacts with shale, and an olivine-rich coarse-grained rock is formed where the T2 magma interacts with dolomite or calc-silicate.

The distribution of the T2U and T2L are shown in Figure 7.9.

Higher PGE and Ni-Cu grades (>4 g/t PGE, >0.4% Ni, >0.2% Cu) are commonly associated with the T2 pegmatoid and chromitite. The Pt/Pd ratios also tend to be higher (>1.0) in association with chromitite and pegmatoid. A mineralised zone (T2MZ) is defined based on a 1 g/t 3PE+Au cut-off that exhibits an average thickness of ~ 25 m.

Figure 7.9 Distribution of the T2U and T2L



Ivanhoe, 2016

7.2.8.2 T1

The T1 pyroxenite is medium to coarse-grained, variably feldspathic, and usually comprises the thickest unit within the TCU (~31 m). The T1 can exceed 100 m in thickness locally.

7.2.8.3 T1MZ and T2MZ Mineralised Zones

Near the upper contact, the T1 contains a mineralised zone (T1MZ) that consists of disseminated, medium to coarse-grained sulfides hosted within the typically equigranular feldspathic pyroxenite with local chromitite stringers. The T1MZ contact is gradational with adjacent weakly to un-mineralised T1 pyroxenite. The T1MZ is better developed where the T1 feldspathic pyroxenite is thickened. The average thickness of the T1MZ is 4.5 m using a 2 g/t 3PE+Au cut-off. Table 7.1 summarises the thicknesses of the T1MZ and T2MZ.

Mineralisation associated with the T2 occurs at the base of the T1 feldspathic pyroxenite (directly above the T2 pegmatoid contact). This mineralisation may contain millimetre-thick chromitite leader stringers, and was previously included in the T1MZ (Parker et al., 2013).

The 2015 geological model shows that only the upper mineralised zone found near the top of the T1 feldspathic pyroxenite can be assigned to the T1MZ. The mineralisation situated just above the basal contact of the T1 should be included with the T2MZ.

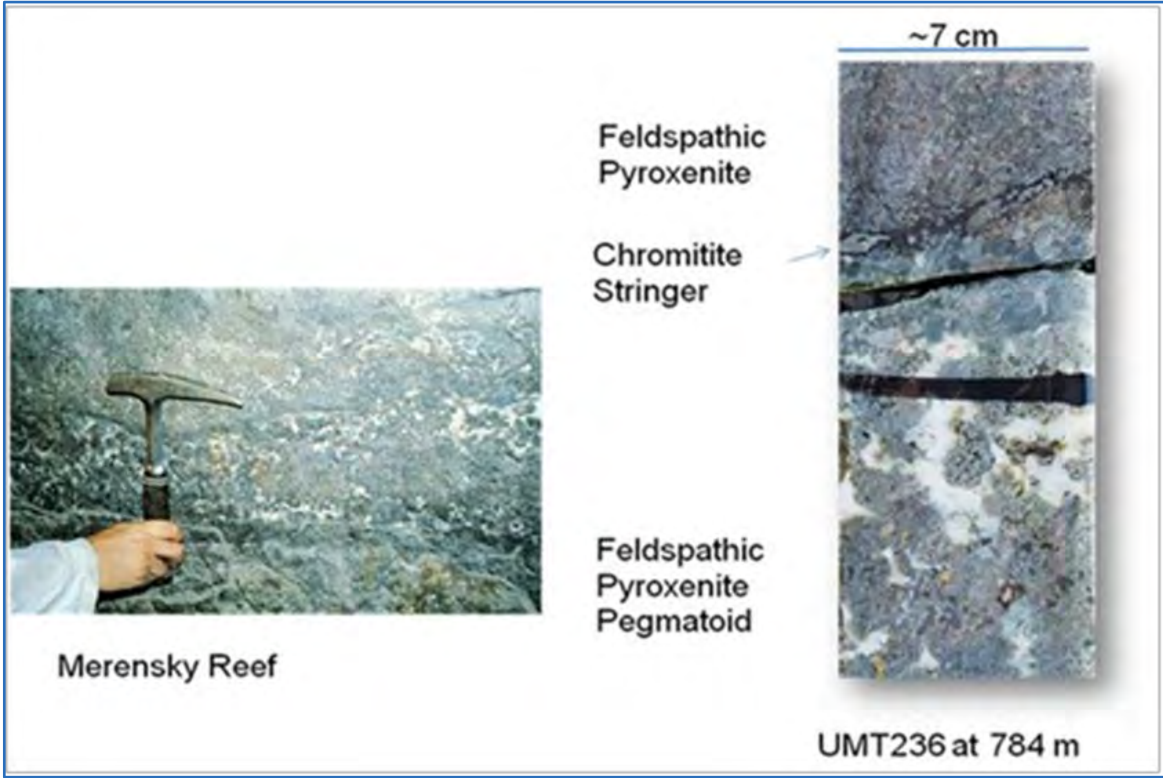
The T1MZ is therefore correlated with the Bastard Reef, and the mineralisation found above the top contact of the T2 pegmatoid, within the feldspathic pyroxenite, is correlated with the M1, as postulated by Davey (1992) and Lea (1996). This implies that the T2 pegmatoid correlates with the M2.

Figure 7.10 is a comparison of the Merensky Reef and the TCU.

Table 7.1 True Thickness of Minzones

Unit	Minzone	Number of Drillholes	Min	Max	Avg
TCU	T1MZ 1g/t	431	1.69	52.31	5.53
	T1MZ 2g/t	431	1.67	17.08	3.84
	T1MZ 3g/t	431	1.28	12.93	3.33
	T2MZ 1g/t	406	2.08	93.41	24.70
	T2MZ 2g/t	406	1.75	66.49	14.99
	T2MZ 3g/t	406	1.18	47.58	8.99
Bikkuri	B1MZ 1g/t	36	2.31	10.17	3.33
	B2MZ 1g/t	75	2.36	40.75	13.78
	B2MZ 2g/t	75	2.25	32.86	6.88
	B2MZ 3g/t	75	1.86	10.47	3.72
CPX	CPX	58	6.45	207.44	84.06
PNZ	AMZ	42	0.74	59.19	12.79
	BMZ	27	0.95	71.70	19.76
	CMZ	20	1.99	40.81	15.10
	DMZ	18	0.84	36.04	15.03
	EMZ	11	2.95	67.89	18.10
	FMZ	5	9.37	54.28	34.34

Figure 7.10 Comparison of Merensky Reef and the TCU



Left photograph taken by Anthony Naldrett of mine face from Rustenburg District, supplied by Ivanhoe 2012. In this photograph, the pegmatoid is shown in white and black, and the chromitite stringers are dark grey. Right photograph by Ivanhoe (2012) of the Platreef within the Platreef Project area. Two dark lines are visible in the Platreef core that are not the chromitite stringer as identified in the core labelling; the top line is a geotechnical break in the core, the basal, thicker line, is a pen line drawn on the core by the logging geologist.

7.2.9 Norite Cyclic Unit 1 (NC1)

The NC1 occurs below the MZ contact and represents the uppermost cyclic unit of the UCZ. The NC1 is laterally extensive with significant changes in thickness. The NC1 consists of a sequence of multiple anorthosite to norite to pyroxenite units with sub-horizontal to horizontal layering. Lateral facies variation from norite cyclic units to feldspathic pyroxenitic units have been observed at this stratigraphic location.

A sporadically developed, well-mineralised pyroxenite unit found as part of the NC1 is now correlated with the T1MZ found in the upper part of the T1 feldspathic pyroxenite. This unit is at the same stratigraphic position as the Bastard Cyclic Unit described from other parts of the BIC (Davey, 1992; Viljoen et al., 1986a, 1986b; Viring & Cowell, 1999).

A laterally extensive mottled anorthosite (MA) occurs between the NC1 and the gabbro-norite of the MZ. The MA occurs at the same stratigraphic level as the Giant Mottled Anorthosite (GMA) of the eastern and western Bushveld Complex. The thickness of the MA ranges from 0 m to several tens of metres.

7.2.10 Main Zone (MZ)

Overlying the magmatic rocks of the CZ is a succession of leucocratic to melanocratic norite and gabbronorite of the MZ. The MZ is the uppermost unit of the RLS observed in drillholes in the Platreef Project area, and MZ forms the hanging wall to the UCZ. Drilling has intersected the MZ up to a vertical depth of 1,450 m.

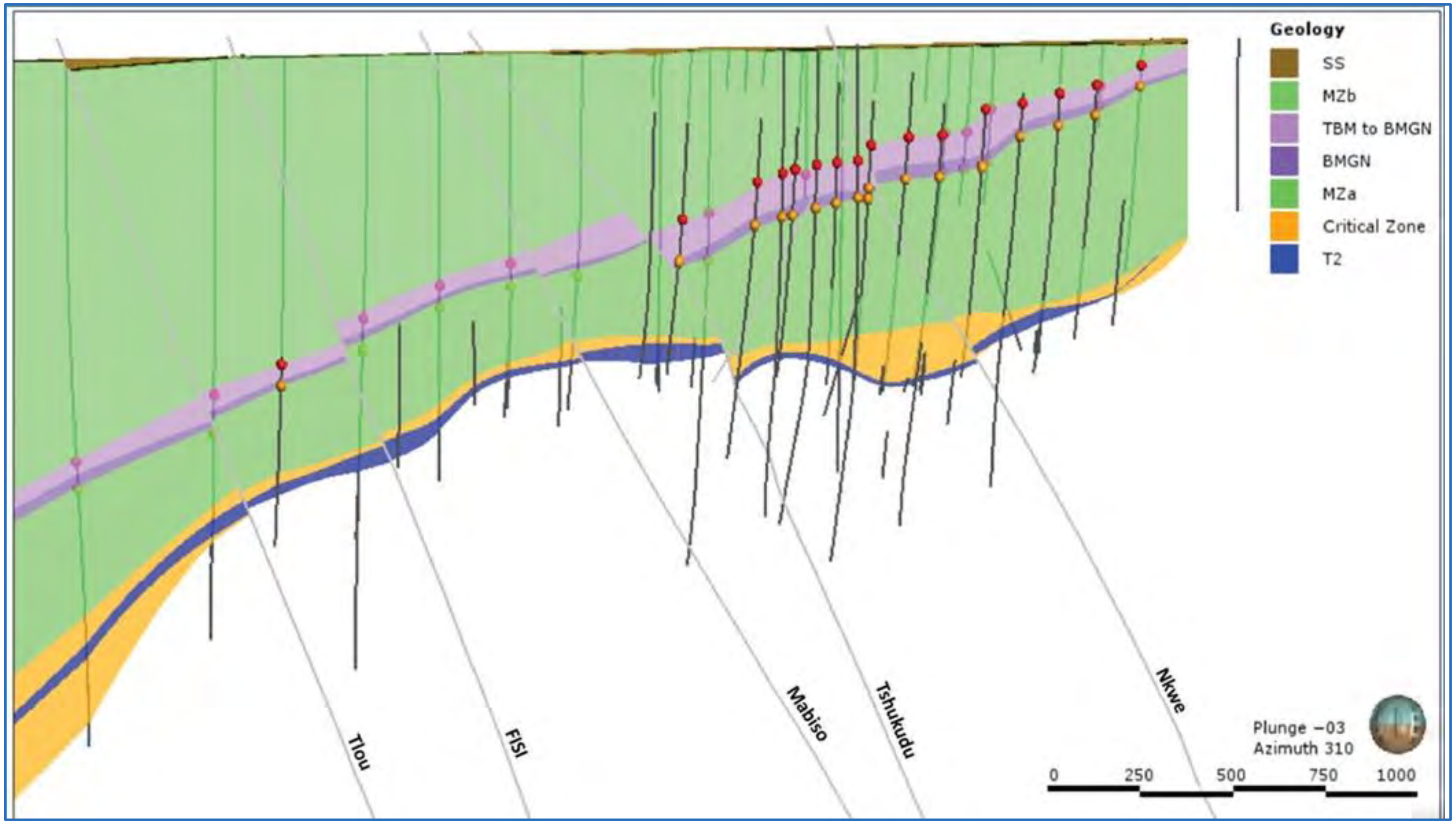
The Main Zone is broken into four units in the Project area (refer to Figure 7.11):

- The interval (MZa) between the bottom of the base of the Main Zone and the base of the 'Basal Melagabbronorite' (BMGN) (bottom green layer in figure);
- The interval between the bottom and top of the BMGN (dark purple in figure);
- The interval between the top of the BMGN and the 'Tennis Ball Marker' (TBM), together shown in the figure in light purple;
- The interval (MZb) between the top of the TBM and the lowermost anorthosite layers (base of the upper Main Zone) (top light green layer).

Within the Main Zone, two units have been informally assigned to 'marker horizon' status, the TBM and the BMGN. These intervals are generally free of metasomatic interaction and thereby demonstrate remarkable continuity as described in other parts of the Bushveld Complex in similar stratigraphic positions (Dunnett, 2015). The fault interpretations in the main Zone are consistent with those made for fault interpretations in the UCZ below, giving credence to the overall structural model.

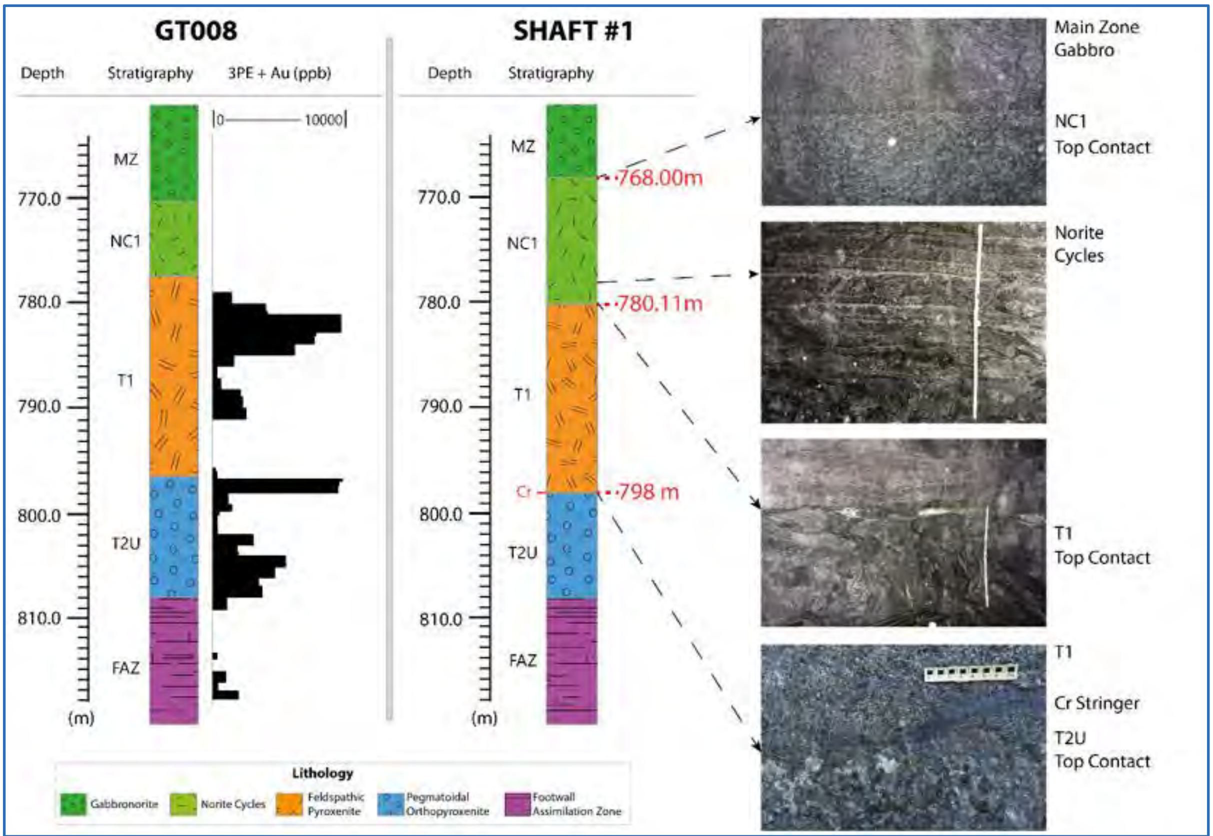
Figure 7.12 compares geology in drill hole GT008 (completed at the location of Shaft 1) and the corresponding intersection in Shaft 1. The top of the T1 was intercepted in Shaft 1 at 780.1 m compared to 777.5 in drill hole GT008. The top of T2U was intercepted at 798 in Shaft 1 compared to 796.5 in drill hole GT008.

Figure 7.11 Main Zone Layers (Section 7.0; see Figure 7.4 for section location)



Ivanhoe, 2016; Faults in grey, Tennis Ball Marker top contact as red points. Basal Melagabbronorite Marker bottom contact as orange points.

Figure 7.12 Lithologies and Grades in Drillhole GT008 and Corresponding Intersection in Shaft 1



Ivanhoe, 2020

7.3 Metamorphism and Metasomatism

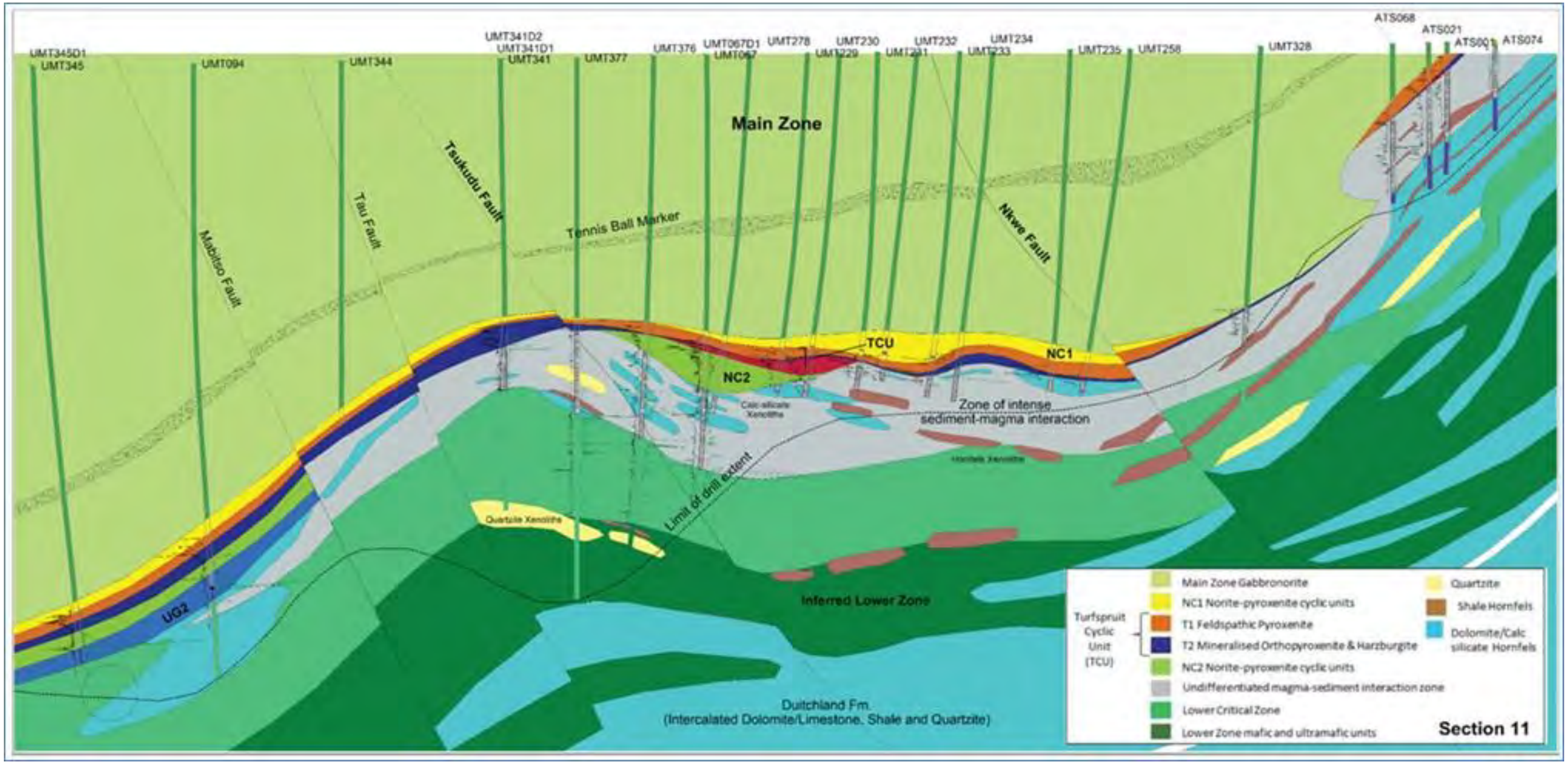
The stratigraphically lower and the up-dip magmatic units on the Turfspruit and Macalacaskop farms are characterised by interactions between Bushveld magma and the Transvaal Supergroup host sequence that is composed of diverse dolomite to alumino-silicate sedimentary rocks of the Duitschland Formation. Magma interaction on Turfspruit mainly involved dolomite/limestone, argillite/shale units, and meta-quartzite (towards the southern parts of the project).

Metamorphism of the sedimentary interlayers varies from moderately intense to locally highly metamorphosed. The contact between the sediments and the Bushveld intrusive rocks vary from sharp to transitional. Intercalated zones of sediment and magmatic units persist over a range of widths from centimetre-scale to hundreds of metres thick, in core intercepts. The degree of in-situ metasomatism and/or melting of the assimilated sedimentary clasts varies according to the sediment type and mineralogy. The metasediments are interpreted to be in-situ relicts of the original country rocks and may form continuous layers at this stratigraphic level that can range from several tens to several hundred metres in any dimension. The following have been noted:

- Partial to complete melting processes dominated the argillite/shale rich units which are normally located within magmatic units of plagioclase bearing pyroxenite and norite units.
- Skarn mineralogy can be developed along sedimentary bedding planes and along xenolith contacts where magma interacted with dolomitic limestone and/or limestone.
- Evidence exists for the inclusion of meta-quartzite assimilation within magmatic units, mainly in the southern part of the Project.

Underlying the variably differentiated CZ units are layers of LZ ultramafic cumulates, that can be a thick as 800 m in some areas (Yudovskaya et al., 2013). The top of the LZ package appears transitional into plagioclase-rich lithologies. Rafts and xenoliths of pyroxene-cordierite hornfels are common and form part of a sequence containing various metasediments metamorphosed to granulite/pyroxene-hornfels facies. These very complex rocks have been variably brecciated and consist of a variety of Mg-skarn minerals. Other alteration products such as talc and serpentine can add local complexity (Figure 7.13). Table 7.2 summarises the mineralised intercepts for the drillholes in Figure 7.13.

Figure 7.13 Diagrammatic Dip Section (see Figure 7.4 for section 11 location)



Ivanhoe, 2016

Table 7.2 Intercepts Grading > 2 g/t and > 3 g/t 2PE+Au Located on Dip Section 11 (Figure 7.13)

Drillhole	From (m)	To (m)	Drilled Length (m)	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE+Au (g/t)	Cu (%)	Ni (%)
2 g/t 2PE+Au Composites									
ATS068	199.30	206.36	7.06	2.32	2.26	0.33	4.90	0.21	0.36
ATS126	333.91	339.79	5.88	1.60	1.19	0.22	3.01	0.21	0.33
UMT328	687.59	703.00	15.41	1.14	1.26	0.27	2.66	0.17	0.29
UMT258	865.00	886.00	21.00	2.06	1.73	0.50	4.29	0.16	0.26
UMT256	836.93	841.00	4.07	2.91	2.55	0.78	6.24	0.26	0.42
UMT235	850.00	865.00	15.00	1.84	1.83	0.29	3.96	0.17	0.27
UMT234	810.00	836.06	26.06	2.12	2.32	0.32	4.75	0.20	0.38
UMT233	843.00	857.00	14.00	2.05	1.98	0.29	4.32	0.21	0.38
UMT232	835.00	853.00	18.00	2.06	1.66	0.26	3.98	0.20	0.38
UMT231	815.00	826.65	11.65	1.60	1.85	0.22	3.67	0.21	0.42
UMT230	803.00	813.00	10.00	1.67	2.10	0.22	4.00	0.17	0.38
UMT229	807.43	817.00	9.57	1.70	1.44	0.16	3.31	0.09	0.22
UMT278	780.00	795.00	15.00	2.07	2.35	0.27	4.69	0.19	0.39
UMT067	758.28	770.26	11.98	2.12	3.12	0.31	5.55	0.22	0.46
UMT376	703.18	718.00	14.82	2.06	2.24	0.34	4.65	0.16	0.32
UMT377	700.00	707.00	7.00	2.72	2.18	0.36	5.27	0.12	0.25
UMT341D1	694.00	711.00	17.00	1.61	1.49	0.35	3.44	0.16	0.32
UMT094	1256.99	1288.50	31.51	1.75	1.76	0.24	3.74	0.12	0.25
UMT345	1429.00	1440.00	11.00	1.91	1.33	0.50	3.74	0.14	0.30
3 g/t 2PE+Au Composites									
ATS068	199.30	202.34	3.04	3.95	3.36	0.51	7.82	0.30	0.53
ATS126	—	—	—	—	—	—	—	—	—
UMT328	699.00	703.00	4.00	1.16	1.56	0.35	3.07	0.23	0.39
UMT258	865.00	882.00	17.00	2.32	1.95	0.58	4.85	0.29	0.16
UMT256	836.93	841.00	4.07	2.91	2.55	0.78	6.24	0.26	0.42
UMT235	850.00	861.09	11.09	2.39	2.22	0.34	7.95	0.21	0.33
UMT234	810.00	836.06	26.06	2.12	2.32	0.32	4.75	0.20	0.38
UMT233	843.00	856.00	13.00	2.05	1.98	0.29	4.32	0.21	0.38
UMT232	835.00	851.26	16.26	2.17	1.72	0.26	4.15	0.20	0.39
UMT231	815.00	821.44	6.44	2.19	2.42	0.28	4.89	0.23	0.47

Drillhole	From (m)	To (m)	Drilled Length (m)	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE+Au (g/t)	Cu (%)	Ni (%)
UMT230	803.00	812.00	9.00	1.73	2.19	0.23	4.16	0.18	0.39
UMT229	807.43	812.18	4.75	2.44	1.51	0.22	4.17	0.08	0.17
UMT278	780.00	793.00	13.00	2.27	2.52	0.29	5.08	0.19	0.39
UMT067	758.28	770.26	11.98	2.12	3.12	0.31	5.54	0.21	0.46
UMT376	708.33	716.00	7.67	2.99	3.52	0.43	6.94	0.25	0.49
UMT377	700.00	706.00	6.00	2.92	2.33	0.39	5.65	0.12	0.25
UMT341D1	695.00	706.00	11.00	1.72	1.57	0.37	3.66	0.16	0.33
UMT094	1257.82	1275.79	17.97	2.44	2.45	0.30	5.20	0.13	0.26
UMT345	1430.00	1440.00	10.00	1.97	1.40	0.51	3.88	0.14	0.30

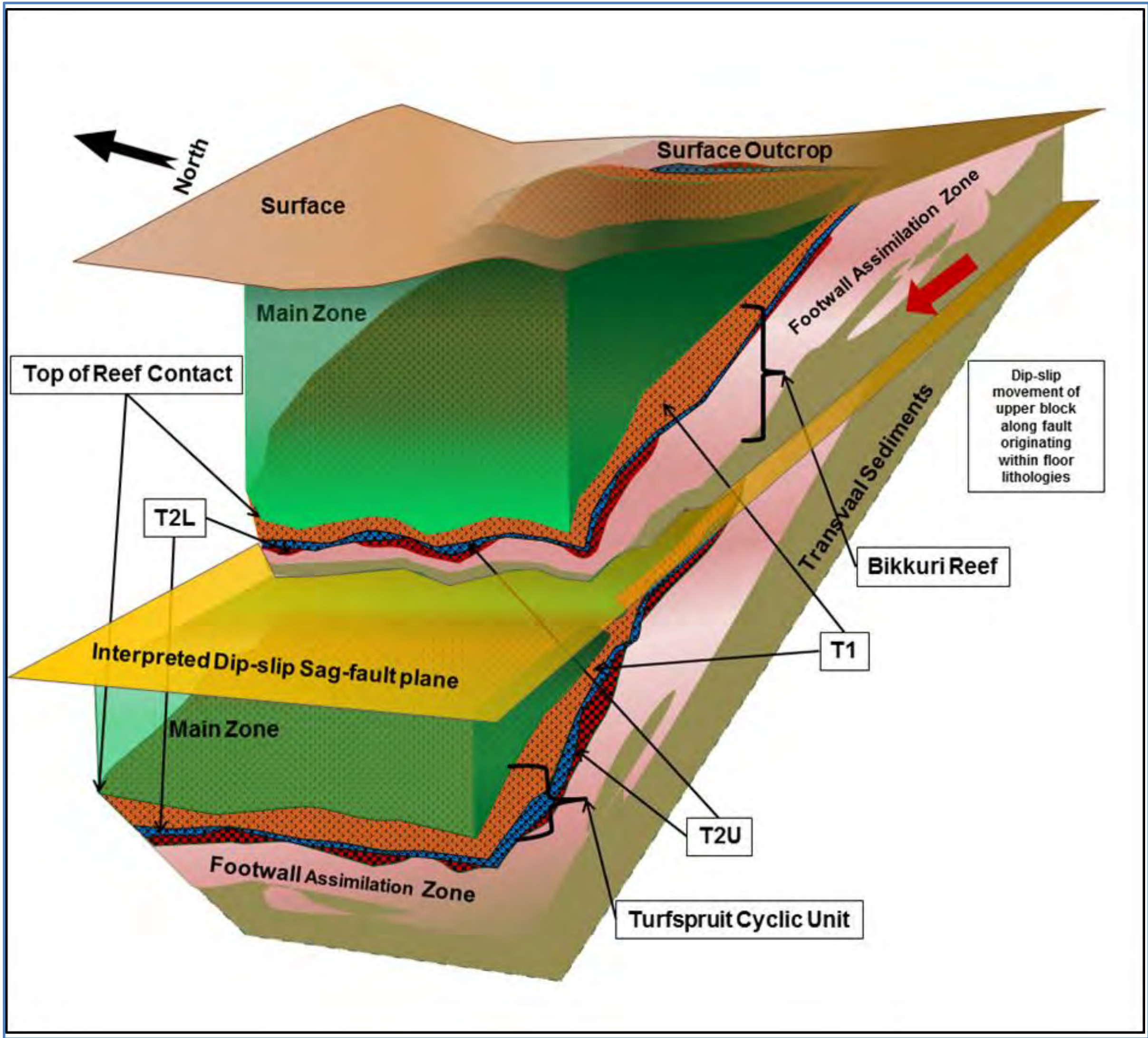
7.4 Bikkuri

Up-dip towards the north-eastern sector of Zone 1, part of the TCU occurs stratigraphically out of position. What is now called the Bikkuri Reef (Bikkuri is Japanese for "surprise") was intersected at depths around 400 m during the 2010–2011 drill programme, where normally reef intercepts were expected at 700 m depths in that area. A second Bikkuri zone has been interpreted in the southern area of Zone 2 where similar mineralisation is located stratigraphically above the TCU.

In most cases, the Bikkuri Reef is represented by thin T1 and T2 reefs (that have been denoted B1 and B2 reefs) directly in contact with highly contaminated calc-silicate footwall rock. The Bikkuri Reef is basically devoid of harzburgitic (olivine-bearing) lithologies. The B2 pegmatoidal pyroxenite is also not well developed, and the associated mineralisation is generally disrupted and of lower-grade. However, recognition of the TCU ("Merensky" analogue) containing chromitite stringers is still possible in most Bikkuri holes. If the unit had not been out of stratigraphic position and had not contained a contaminated footwall, the TCU within the Bikkuri Reef would have been regarded as part of the T2 Reef.

The Bikkuri is interpreted to be the result of semi-consolidating magma that slumped back into the crystallising magma chamber (Grobler et al, 2013). Figure 7.14 shows a diagrammatic view for the Bikkuri emplacement.

Figure 7.14 Schematic Diagram of Preferred Emplacement Mechanism for Bikkuri.



Ivanhoe, 2016, After Grobler et al (2013)

7.5 Structure

7.5.1 Regional Structures

Structurally, the Northern Limb is separated from the rest of the Bushveld Complex by the Thabazimbi-Murchison Lineament (TML). The TML is a pre-Bushveld, major, compressional tectonic boundary (suture zone) that formed as a result of the collision of the Pietersburg terrane and Kaapvaal shield around 2.97 Ga during the Murchison Orogeny (Friese, 2003, 2004). The Ysterberg-Planknek and Zebediela Faults play a significant role in the regional geology of the Northern Limb.

The tectono-thermal evolution can broadly be subdivided into pre- and syn-emplacement folding and multiple faulting events. Folding in the Northern Limb has been controlled by two principal transpressional events caused by movements along the TML in the south and the Palala Shear Zone.

According to Nex, (2005), this led to the formation of two main open-fold geometries within the Transvaal sediments. The first and most dominant folding event was caused by NE-SW sinistral transpression. This resulted in regional NNW trending low amplitude, sub horizontal open folding. These F1 folds developed within Archaean basement and Transvaal Supergroup and represent the earliest developed structures which formed contemporaneously as a result of mild ENE-WSW compression during the Limpopo-Murchison Orogeny at 2.78–2.64 Ga. Subsequent NW-SW transpressive inversion refolded the earlier F1 fold axis resulting in basin and dome fold interference patterns (Friese, 2012).

Significant brittle faults and ductile shear zones are known throughout the Northern Limb, and the major, widely-spaced, ENE-trending shear zones dominate the regional map pattern. These combine to form large strike-slip duplex systems, which host a complex array of riedel shears, normal faults, thrusts and dilational tension fractures that have been invaded in part by igneous dykes and quartz-feldspar veins. These faults are reactivated during a major E-W crustal extension event associated with major brittle fracturing.

The major fault regimes after Brits and Nielsen (2015) and Friese (2012) are summarised as:

- NW to NNW trending, moderate to steeply dipping "Pongola" extensional faults/fault zones that formed within the Transvaal Supergroup and BC by reactivation of the similar oriented Neoarchaean (~2.98-2.96 Ga) Pongola rift fault system developed in the underlying Archaean basement during the Murchison Orogeny.
- NE to NNE trending, steep to subvertical predominantly south-easterly dipping "Ventersdorp" dextral strike-slip shear zones with associated NE directed, layer/bedding-parallel thrust developed in shear zone-bounded domains. The dextral strike-slip system formed within the Transvaal Supergroup and BC by reactivation and above the Neoarchaean (~2.78-2.64 Ga) Ventersdorp sinistral strike-slip system, which developed within the underlying Archaean basement in response to sinistral transpressive tectonism during the Limpopo Orogeny (taking place at approximately the same time).
- N-S striking, moderate westerly dipping "Kibaran" extensional fault zones, with typical undulating gross geometry and an imbricate fan of combined normal dip-slip and sinistral strike-slip duplexes in their immediate hanging wall.
- WNW- to WSW-trending "Soutpansberg" extensional fracture/joint zones and associated dolerite dykes cross-cut all other structural discontinuities without significant displacement.
- Shallow NW dipping, SE-directed thrusts/thrust zones and associated ENE-trending, sub horizontal, low-amplitude regional F2 folds formed in pre- to syn-RLS time as a result of mild SE-directed in situ compressive far field stress generated within the northern Kaapvaal Craton during the early stages of the Ubendian Orogeny at ~2.1-2.058 Ga.

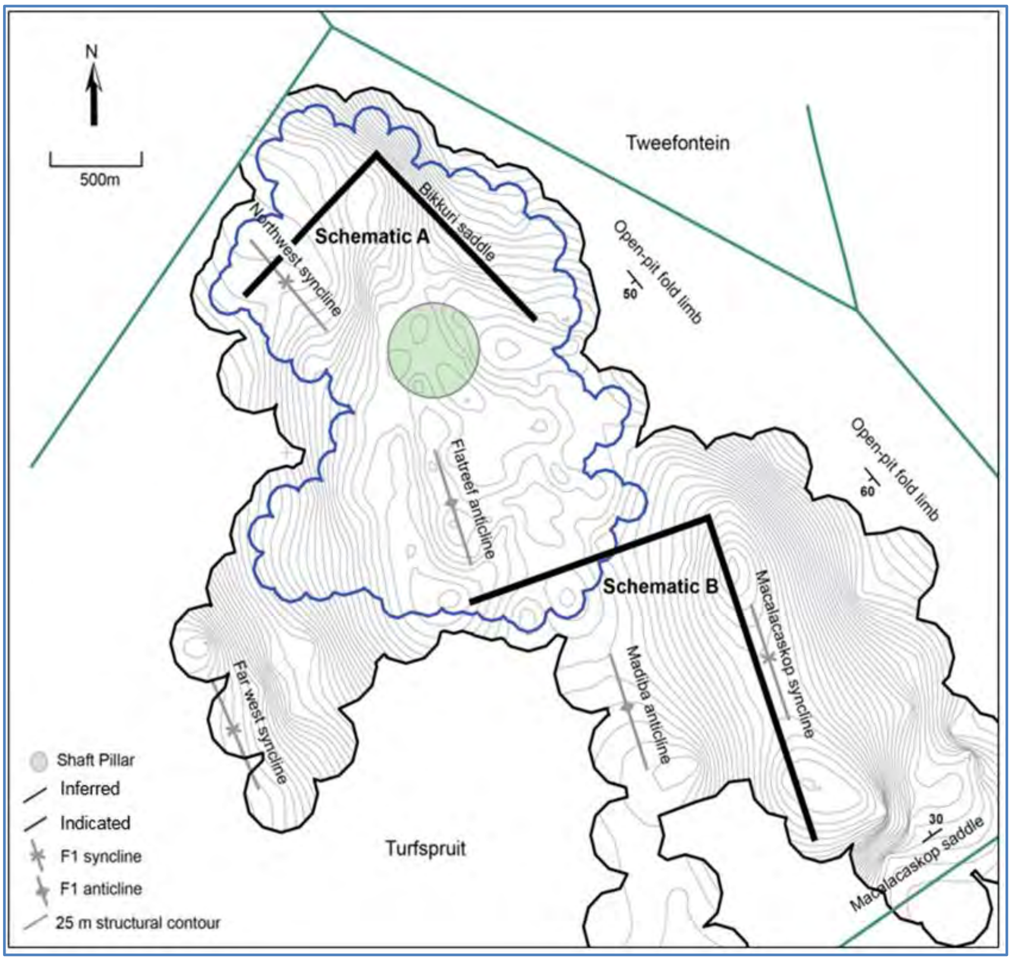
7.5.2 Project Folding

Two fold orientations have been observed, and these concur with the previous Northern Limb studies. The first and major fold axial orientation (F1) is NNW-SSE. These folds have subsequently been gently refolded with the minor fold axis (F2) trending ENE-WSW. The F1 folds are responsible for the apparent flattening of the Platreef basinward, the Macalacaskop syncline, the so called "T1-trough" and the overall 50° dip to the southwest along the open-pit fold limb. The minor folds are responsible for domes and basins within the larger folds such as the Bikkuri dome.

Broadly, Zone 1 or the 'Flatreef' could be interpreted as a monocline or parasitic fold on a major NNW-trending, SW-dipping fold limb. Syn-magmatic sagging or uplift due to crustal loading and volume increase may have locally amplified the synclines and anticlines respectively.

Figure 7.15 shows a Project-scale view of the major (F1) low amplitude open folding and Plan View of Major F1 Fold Structures on Structural Contour Map.

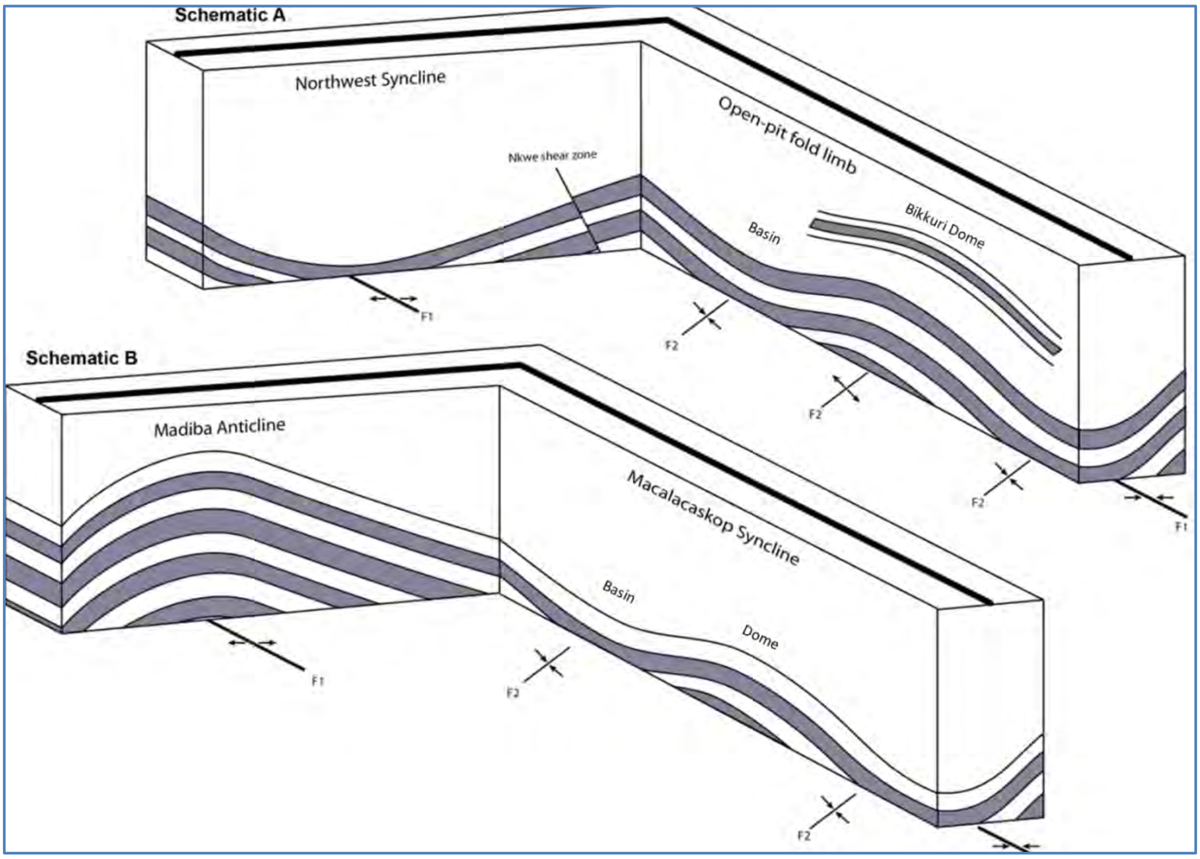
Figure 7.15 Plan View of Major F1 Fold Structures on Structural Contour Map



Ivanhoe, 2016; Structural contours define the base of the T2. Section lines refer to the sections shown in Figure 7.14.

Figure 7.16 is a schematic of the interpreted folding derived from metasedimentary interlayers or "rafts".

Figure 7.16 Schematic View of Fold Structures in Zones 1 and 2



Ivanhoe, 2016; Oblique view from south–north of schematic folding derived from metasedimentary interlayer "rafts". Figure 7.15 shows the location of the fence lines.

7.5.3 Project Fault Structures

MTS notes that a reasonable interpretation of the faulting was only possible when drill spacings were tightened to 100 m at Zone 1. The wider drill spacings of 200–400 m in Zones 2, 3 and 5 would not support a detailed interpretation. Information for Zone 4 remains to be re-interpreted.

A structural model for the Project area was constructed for Zone 1using the regional structural regime, and Project-specific information from drill core, a three-dimensional seismic survey, Falcon gravity survey and comprehensive Main Zone core photography. The structural model includes three key deformation features:

- Folding – Pre-Bushveld low amplitude, upright open folds defined by remnant metasedimentary interlayers and xenoliths that are oriented parallel to mineralised zones.
- Ductile shear zones – 30 cm to 3 m wide, NW trending, steeply dipping (60° to 70°), oblique reverse sense of movement, variable dip direction, possible antithetic riedel shear zones.

- Brittle fault zones – 5 m to 30 m wide, north trending, moderate to steeply dipping (50° to 70°), extensional (east block down) normal faults.

A total of six faults are used to define seven fault blocks for the structural model. A further three structures of lesser confidence have been interpreted and modelled but are not used as model domains. All nine structures were interpreted primarily from drill cores with use of geophysical data being limited to correlation of structural trends.

The Tshukudu Fault Zone is a brittle structure that transgresses the central portion of Zone 1 and Zone 2.

It represents a significant geotechnical hazard and comprises a wide zone of imbricate fracturing in its hanging wall and intense brecciation within the fault zone. Major fall-of-ground hazards can be expected where this brittle fault intersects ductile shear zones. Significant vertical displacement is associated with this fault zone in the order of 60 m (Brits, 2015). The fault zone is generally steeply inclined, and has an easterly dip direction and oblique normal sense of movement. The fault is defined by 129 drill core intersections and has a minimum thickness of 0.6 m and a maximum thickness of 26 m for an average thickness of 7.6 m.

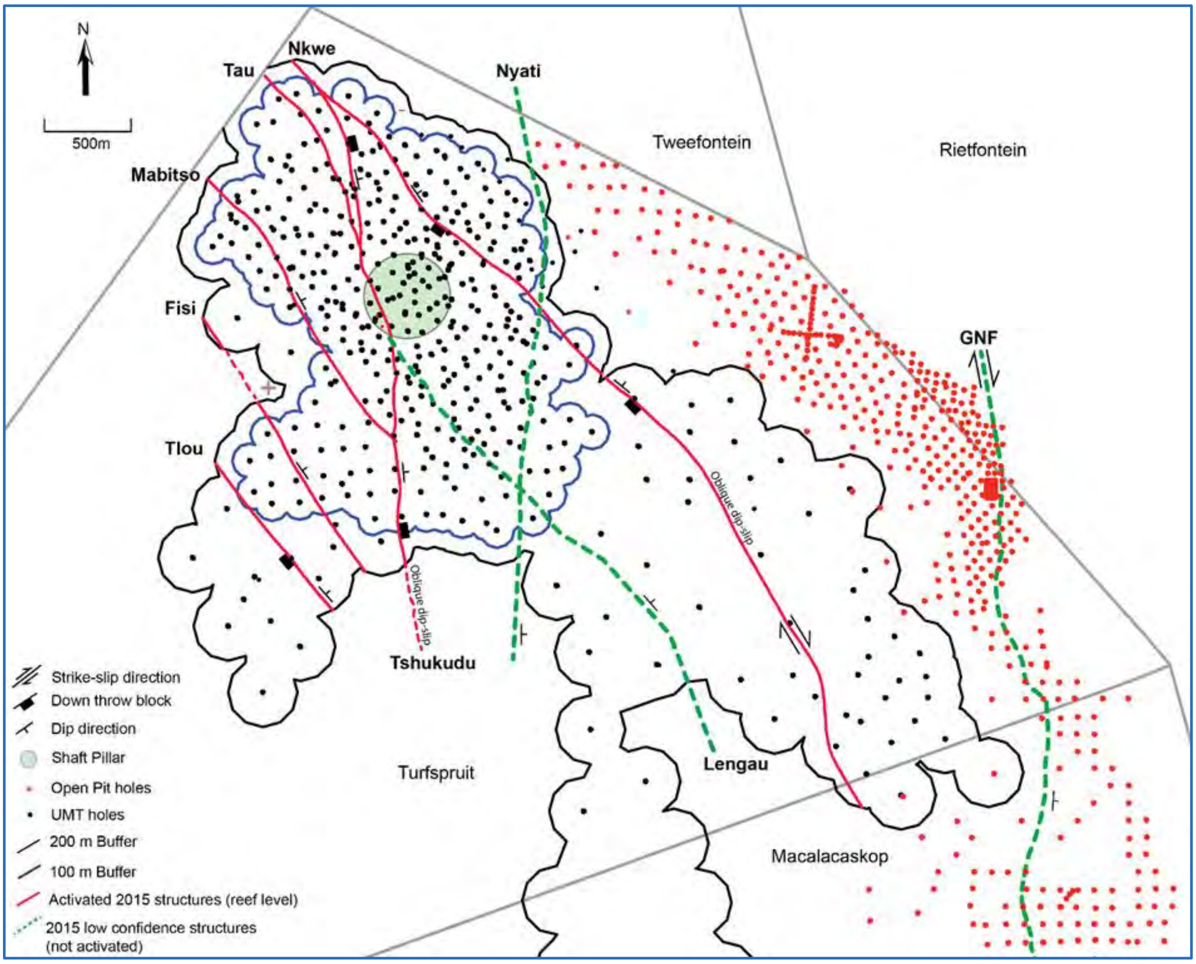
The major ductile fault structures currently recognised include:

- Nkwe: defined by 124 drill core intersections and has a minimum thickness of 0.15 m and a maximum thickness of 10.1 m for an average thickness of 1.3 m; relative movement indicators are mostly not discernible, but occasionally indicate reverse dip-slip sense of movement.
- Tau: defined by 36 drill core intersections and has a minimum thickness of 0.6 m and a maximum thickness of 10.4 m for an average thickness of 3.1 m; has a strike length of approximately 1,800 m before terminating along the major north-trending Tshukudu fault zone.
- Mabitso: defined by 25 drill core intersections and has a minimum thickness of 0.3 m and a maximum thickness of 3.5 m for an average thickness of 1.6 m; has a strike length of approximately 1,900 m before terminating along the Tshukudu fault zone.
- Fisi: defined by 11 drill core intersections and has a minimum thickness of 0.35 m and a maximum thickness of 3.3 m for an average thickness of 1.8 m; has a strike length of approximately 1,400 m before terminating along the Tshukudu fault zone.
- Tlou: defined by 6 drill core intersections and has a minimum thickness of 2.4 m and a maximum thickness of 5.4 m for an average thickness of 3.6 m; has a strike length of approximately 1,400 m beyond which no further drill data are available; displays significant vertical offset.
- Lengau: a low-confidence feature; defined by 26 drill core intersections; has a minimum thickness of 0.3 m and a maximum thickness of 5.0 m for an average thickness of 1.5 m; has a strike length of approximately 5,000 m beyond which no further drill data are available; dips north-easterly; appears to be an interlinking feature between the Tshukudu and Nkwe structures.
- The remaining fault zones are not used to delimit domain boundaries.

- Great North Fault zone (GNF): GNF is associated with interpreted offsets in surface mapping, magnetics and unreliable brittle fault development in drill cores; defined by 18 drill core intersections and has a minimum thickness of 0.5 m and a maximum thickness of 2.2 m for an average thickness of 1.7 m.
- Nyati fracture zone: defined by 58 drill core intersections and has a minimum thickness of 0.5 m and a maximum thickness of 28 m for an average thickness of 4.8 m.

Figure 7.17 shows the faults included in the Mineral Resource model. The dip direction and sense of movement are also shown.

Figure 7.17 Structure Interpretation for Model Domains



Ivanhoe, 2016

Three primary structural trends are evident for the steep structures:

- Northeast-trending 'Ventersdorp' strike slip faults, which are significant structures and are known from surface mapping along the Northern Limb to offset the Platreef contact (See Figure 7.3).

- A predominantly ductile north-west trending 'Pongola' fault and dyke system. The Nkwe, Noko, Lengau, Tau, Mabitso and Fisi faults all align on this orientation and show broadly similar characteristics. A well-developed set of granitic dykes is also evident in this orientation.
- A north- to NNE-trending set of brittle Kibaran faults. The Tshukudu Fault is the largest structure observed in the Project area and falls into this category, along with the Nyati Fracture Zone.

Additional faults, aligned either parallel to the Tshukudu Fault (i.e. Nyati Fracture Zone) or the Nkwe Fault (i.e. Tau and Mabitso Shear Zones), have also been identified and modelled. Shear zones aligned on the northwest trend (parallel to the Nkwe Fault) are occasionally associated with granitic dykes.

7.5.4 Granite Dykes

Two sets of granite dykes have been modelled based on their relative dip. Although classified separately, it is thought these dykes form part of an anastomosing swarm of syn-Bushveld intrusions contiguous with tension fractures and dilational zones in response to regional transpression.

The granite dykes form a dyke swarm cutting through the Project area. The dykes strike northwest and dip steeply (60° to 75°) towards the northeast. The dykes range in thickness from several centimetres to tens of metres. Granite dykes > 2 m thick have been modelled. The granite dykes are commonly orientated sub-parallel to the ductile shear zones.

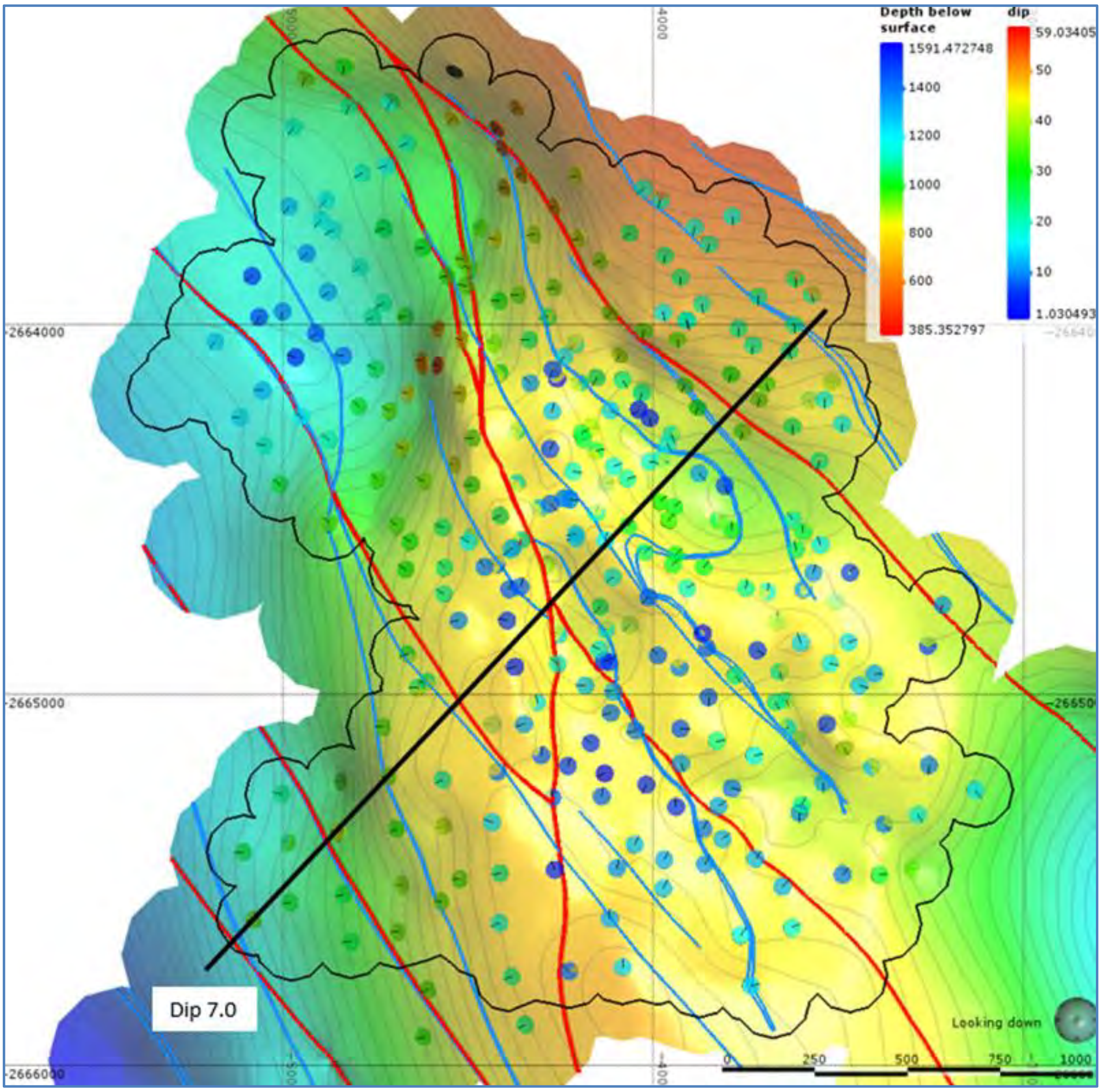
Figure 7.18 shows the locations of key dyke features in relation to the structural model at the level of the T2 horizon.

Two sets of granite dykes have been modelled, based on their relative dip:

- Low Angle Granite Veins (LGVs): strike 335° and dip at 32° towards the northeast; a total of eight sub-parallel dykes have been modelled as continuous features named LGV10, LGV20, LGV30, LGV40, LGV50, LGV60 and LGV70. Intersections of LGV40 and LGV50 in Shaft 1, and further modelling of these features from drill intersections, has shown their respective zones of influence to be highly variable in width, character and location (Brits, 2020a). Shaft mapping also revealed an additional low angle feature between LGV40 and LGV50 (LAF45) with variable granite healing (Brits, 2020b);
- Steep Granite Veins (SGVs): strike 329° and dip at 68° towards the northeast; a total of 10 sub-parallel zones with increased dyke frequency of occurrence and widths have been modelled as continuous features and named SGV10 to SGV100. Numerous additional granite intersections in drill core (< 2 m thick) indicate that a significant number of narrow stockwork-type intrusions should be anticipated during underground development. This is particularly relevant between SGV10 and SGV20.

The majority of the granite dykes are intersected within Main Zone rocks, with a relatively minor amount of intersections within the mineralised reef horizons. The granites are concentrated in the central portion of Zone 1 concordant with the gently dipping 'Flatreef', whilst the intensity of intersections decreases markedly to the west and northwest.

Figure 7.18 Structural Model Intersecting the Base of The T2 Surface



Ivanhoe, 2016; Red = Faults, Blue = granite dykes. Structural contours (25 m RL) on base of T2 show inferred fold pattern. Dip directions shown by black line on structural disc.

7.5.5 Other Considerations

The detailed structural investigations also identified other features that may impact mining and ground support, as follows:

- Low-angle flexural slip planes (micro-thrusts) sub-parallel to reef-type mineralised zones. These discontinuities have been identified elsewhere in the BIC. Displacement is expected to be centimetre-scale, and the discontinuities represent planes of weakness that will need to be carefully monitored during mining activities.

- Sedimentary xenoliths are evident throughout the stratigraphy but particularly immediately below the main mineralised zone. Geometries are expected to vary from high to low intersection angles and may represent zones of weakness.

7.5.6 Conclusions

The structural regime observed at Turfspruit and Macalacaskop appears to be a classic illustration of large-scale strike slip duplex systems compatible with the regional evolution of the BIC (Friese, 2012). At Turfspruit, it seems most likely that the orientations of the modelled ductile shear zones, the extensional Tshukudu fault zone, the observed folding and the granite dyke swarm can be explained with a certain degree of confidence as a long-lived strike-slip duplex configuration that has seen transpressive inversion.

Mine planning considers the fault orientations as well as the broader zone of faulting and fracturing associated with them. These zones are variable, but are most strongly associated with the Tshukudu Fault, particularly in the interaction zone between the Tshukudu and Pongola-related structures (i.e. the Tau, Mabitso and Fisi shear zones).

7.6 Mineralisation

7.6.1 Mineralogy of PGE-Base Metal Mineralisation in the Platreef Project Area

There are five separate PGE mineralised zones located in the UCZ on the Platreef Project (Table 7.3).

The T1 and T2 Reefs are the best developed and display good continuity across the Platreef Project area. The magmatic mineralisation on Turfspruit 241 KR exhibits similar geological characteristics as described for the Merensky Reef within the UCZ of the BIC. The T1 and T2 Reefs display much less contamination from meta-sedimentary xenoliths than the units that are stratigraphically below the TCU.

The mineralisation within the FAZ and UG2 Reef located stratigraphically below the TCU are less continuous due to meta-sedimentary xenoliths and associated contamination and/or alteration.

Two areas below the TCU have been identified where continuous mineralisation zones occur. A clinopyroxenite domain (CPX) is within the FAZ in north-western portion of the Zone 1. The CPX is a distinct lithological domain that hosts continuous low-grade Ni mineralisation with local 3PE+Au mineralisation. The CPX can form a continuous zone of mineralisation below the base of the T2MZ. No meta-sedimentary xenoliths have been identified within the CPX domain, suggesting xenoliths have been completely assimilated.

A PNZ domain includes predominantly disseminated sulfide mineralisation within homogeneous pyroxenite/norite lithologies. Locally, massive sulfides occur at contacts with hornfels rafts. Mineralisation is typically 1 g/t 3PE+Au, but locally can be 2 – 5 g/t 3PE+Au. Mineralisation also occurs at the contact between the FAZ and the PNZ.

Table 7.3 Cyclic Unit Mineralisation

Cyclic Unit	Mineralised Zone	Description
NC1CU	BAR	Fine to medium-grained magmatic sulfides hosted in feldspathic pyroxenite. BMS are predominantly chalcopyrite, pentlandite and pyrrhotite.
TCU	T1	Medium to coarse-grained magmatic sulfide grains hosted in feldspathic pyroxenite.
	T2 (Merensky reef analogue)	Very coarse-grained magmatic sulfides hosted in pegmatoidal orthopyroxenite and pegmatoidal poikilitic harzburgite. The top of the mineralised zone is commonly marked by a chromite stringer.
FAZ	UDCZ1	Medium- to coarse-grained magmatic sulfides hosted in pyroxenite, feldspathic harzburgite (FHA)/clinopyroxenite (FCPX), parapyroxenite and paraharzburgite. High percentage of base metal (Ni and Cu) is associated with this unit.
UG2CU	UG2	Fine-grained sulfides hosted in chromitite. Associated with high- grade PGEs.
PNZ Mineralisation	Platreef contact style mineralisation	Fine-grained massive sulfide bodies hosted mainly in the pyroxenite and norite of the PNZ.

UDCZ=Undifferentiated Contaminated Zone.

7.6.2 Platinum Group Minerals (PGM) and Base Metal Sulfides

Work completed by various authors has indicated there is a high variability in the character, distribution, and morphology of platinum group metals and minerals (PGM) in the BIC and on the North Limb.

Hutchison (2003), and Hutchison and Kinnaird (2005) completed work on ATS and AMK drillholes in the area of the historic open-pit resource that suggests stratigraphic interpretation influences sample selection and study conclusions. The sampling methodology employed to sample mineralised units in the Northern Limb has been found to be critical (Grobler et al, 2016). Recent knowledge and interpretations suggest the results are relevant to the sections of Platreef where assimilation of meta-sedimentary lithologies have affected PGE and base metal sulfide (BMS) assemblages.

The current understanding of the stratigraphy of the Northern Limb has guided new sampling of mineralised units on the Platreef Project. Improvements in the representivity of datasets characterising the mineralisation coupled to the latest advances in microscopy (electron microprobe (BSE) and EDS spectrometry) has led to a greater mineralogical understanding of the PGE found on the Platreef Project.

Studies have succeeded in distinguishing the magmatic, high-temperature assemblage PGMs from PGM distributions affected by assimilation, melting and alteration processes. The latter related to footwall units, the FAZ and the area that hosts the historic open pit resource.

7.6.2.1 Base Metal Sulfides

Within the Platreef Project, sulfide occurrence consists mainly of pyrrhotite, pentlandite, chalcopyrite and lesser pyrite. Sulfide distribution and concentrations vary and range from less than 1% to more than 25%. Rare sections of core may have massive sulfides over a scale of tens of centimetres (Brits, 2016).

Numerous textures are observed in drill core. The most frequent are large fractionated blebs often in association with smaller disseminated mono-mineralic grains. This textural variance suggests several phases of sulfide formation. An early phase is dominated by irregular blebs of disseminated pyrrhotite and pentlandite followed by a later phase where chalcopyrite is dominant.

Figure 7.19 and Figure 7.20 shows sulfide mineral assemblastes found in the T2.

7.6.2.2 Platinum Group Minerals

The distribution of the discrete PGMs within the Platreef is broadly controlled by stratigraphic position. The uppermost part of the Platreef commonly hosts the highest PGE grades. The PGM distribution can be erratic on a hand-specimen scale. The findings made by Yudovskaya et.al. (2011) and Hutchinson (2003) were confirmed by a study of the core samples collected by Smart (2015).

The similarity in PGE mineral assemblage between the T2U and the Merensky Reef has been confirmed by geometallurgical characterisation studies (Govender et al., 2015). The studies determined that for all the geomet units, PGM-tellurides are dominant, followed by PGE-arsenides and PGE-sulfides. The abundance of PGE-arsenides, antimonides, Bi-Te minerals and PGE-sulfide minerals that are associated with the upper part of the T2U geomet unit corresponds to the upper part of the Merensky Reef found elsewhere. The geometallurgical study has also shown that only a small fraction of PGE mineral assemblage are associated with BMS.

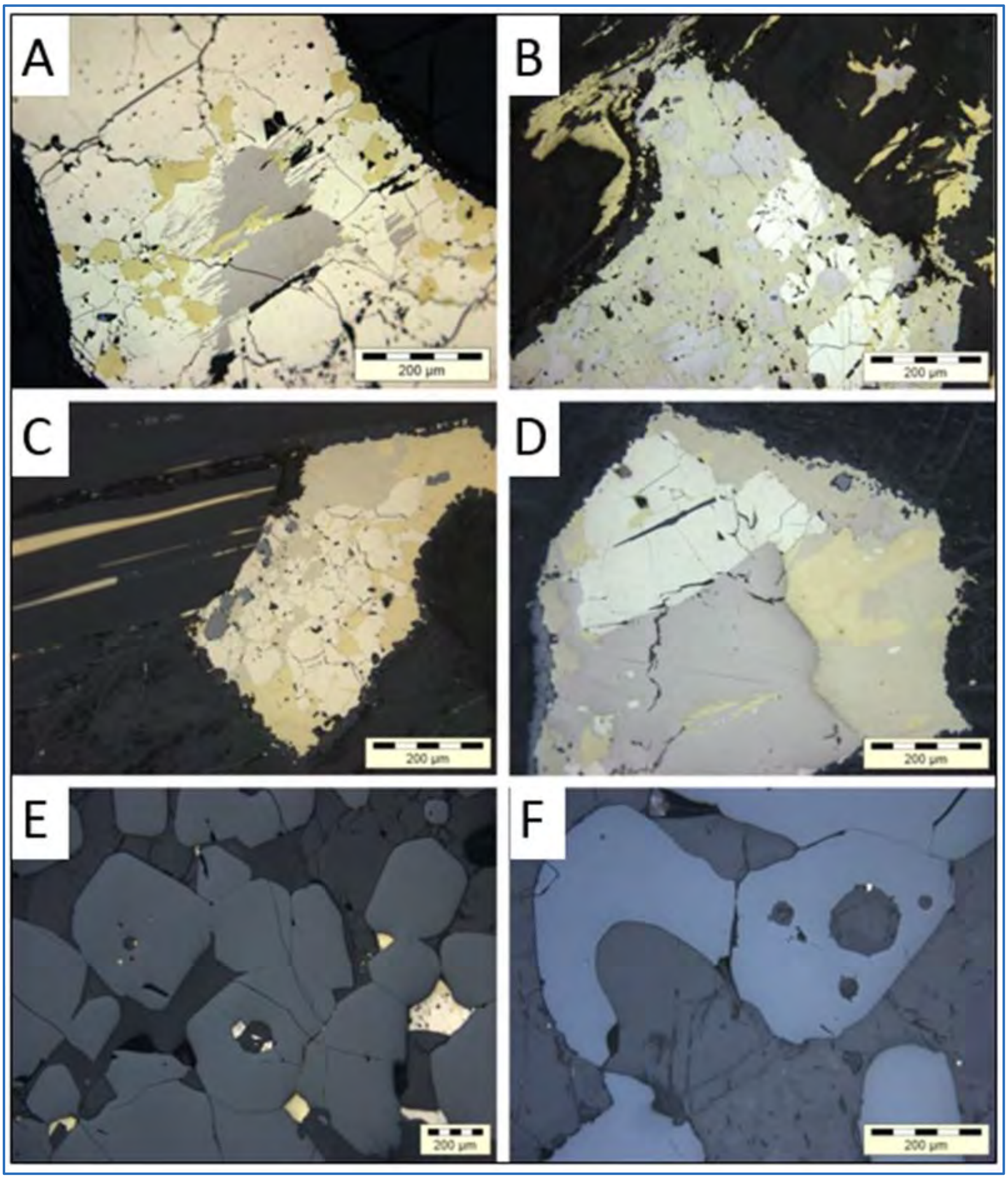
Yudovskaya (2015) determined that clear magmatic assemblages (Merensky-like trends) can be distinguished from an original assemblage influenced and overprinted by secondary effects. The zonation of PGM distribution favours in-situ crystallisation where modal PGE mineral assemblages are controlled by the thermal gradient.

The Bastard and T2U reefs contain an association of high-temperature primary magmatic Pt sulfides and Pt alloys that often form eutectoid intergrowths with base metal sulfides. This is an indication of crystallisation at around 1,000ºC. Chromitite is the only lithology which contains laurite (RuS2). Figure 7.21 shows detailed SEM images with PGE mineral assemblage and textural relationships typical of the T2 reef.

Figure 7.19 Core Photograph from UMT083 at 1,323 m Depth, Illustrating Sulfide Mineralisation

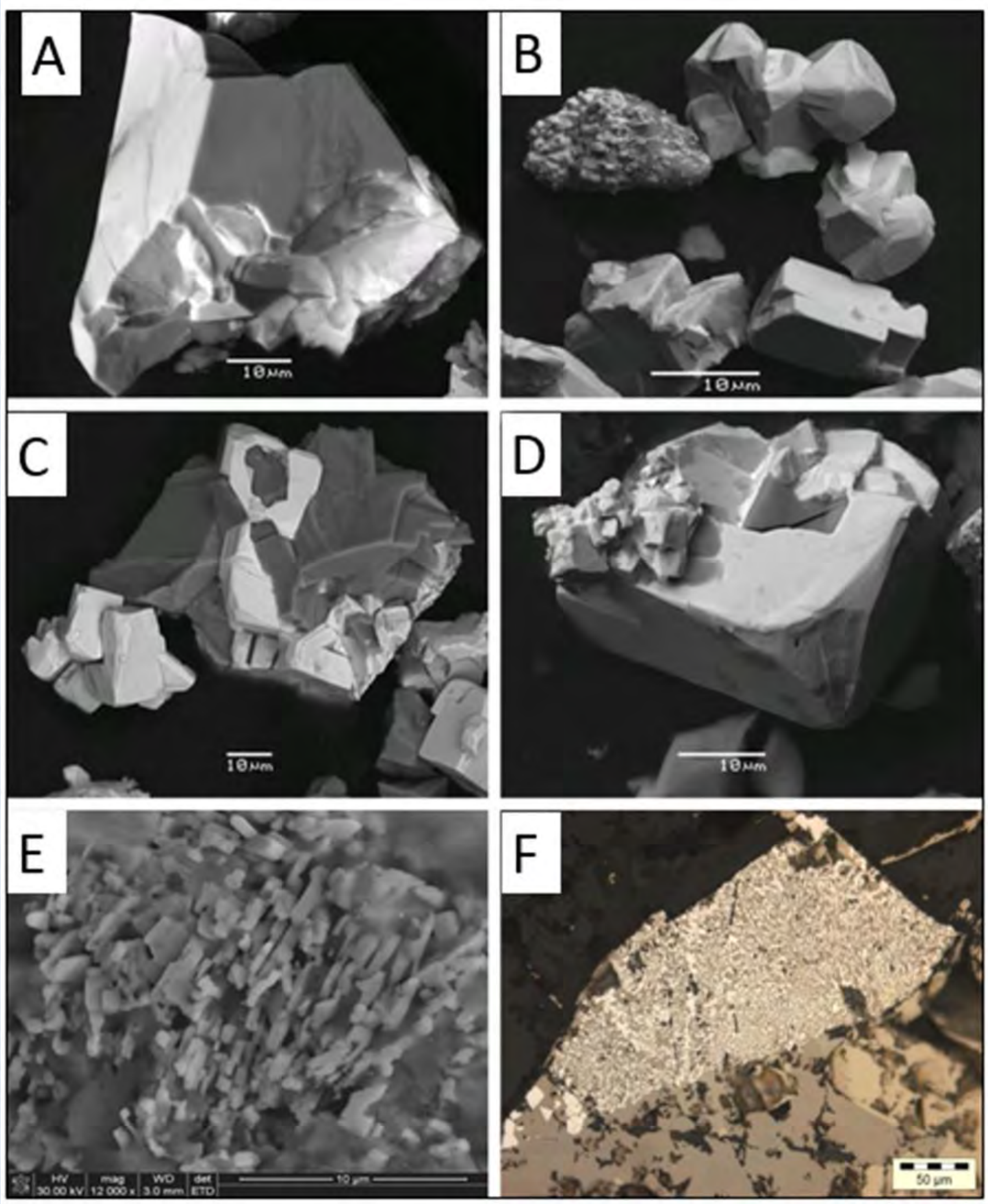
Ivanhoe, 2011; The yellowish mineral is chalcopyrite; the dull purplish mineral is pyrrhotite; the light cream mineral with higher reflectance and some cleavage is pentlandite.

Figure 7.20 Mineral Assemblages Found in the T2U



Yudovskaya, 2015; A-Flame-like pentlandite after pyrrhotite (UMT314 at 1,135.5 m). B – Relics of pyrrhotite in cubanite. Granular pentlandite is white, chalcopyrite is remobilised outside the massive sulfide intergrowth (UMT314 at 1,136.7 m); C – Pyrrhotite-cubanite-pentlandite assemblages is replaced by secondary silicates along margins (UMT314 at 1,136.7 m); D – magmatic pyrrhotite-cubanite-pentlandite assemblages is rimmed by later magnetite rim (UMT314 at 1,136.7 m); E – atoll-like and sieved chromite in the chromitite seam. Sulfides are seen as interstitial and inclusions in chromite (UMT314 at 1,135.54 m); F – embayed and atoll-like chromite of the lowermost chromitite seam (UMT314-1160) unpublished internal correspondence). See Figure 7.4 for location of UMT314.

Figure 7.21 SEM Images of Typical Platinum Minerals from The T2 Reef



Yudovskaya, 2015 (unpublished internal correspondence); A: RuS2 intergrown with IrAsS (UMT314 at 1,135.54 m); B: PtS and Pt3Fe crystals as well as fine-grained eutectoid intergrowth of Pt-Fe alloy and pyrrhotite (UMT314 at 1,135.54 m); C: euhedral skeletal crystals of isoferroplatinum intergrown with pyrrhotite (UMT314 at 1,135.71 m); D: wide range of isoferroplatinum crystal sizes (UMT314 at 1,135.71 m); E: micron-sized crystals of isoferroplatinum in sulfides (UMT314 at 1,135.71 m); F: the same type as in E eutectoid intergrowth of Pt3Fe and pyrrhotite adjacent to coarser pyrrhotite and pentlandite under reflected light (UMT314 at 1,135.54 m).) See Figure 7.4 for location of UMT314.

7.7 Comments on Section 7

In the opinion of Mr Kuhl, knowledge of the deposit settings, lithologies, mineralisation style and setting, and structural and alteration controls on mineralisation within the UMT-Bikkuri, UMT-TCU and UMT-FW deposits are sufficient to support Mineral Resource estimation.

The detailed comparison between the TCU and the Merensky Cyclic Unit and establishment of correlative subunits in uncontaminated lithologies is based on a significant accumulation of drill core, geophysical studies, geochemical and petrologic investigations.

The data support the structural model and an understanding of the magmatic stratigraphy on the Platreef Project.

8 DEPOSIT TYPES

Two main PGE deposit types occur within the Bushveld Complex:

- Relatively narrow (maximum 1 m wide) stratiform layers (reefs) that occur towards the top of the Upper Critical Zone typically 2 km above the base of the intrusion (Merensky reef-style), mainly found in the Western and Eastern Limbs. These narrow zones have been the principal targets for mining in the past; however, more recently wider zones with more irregular footwall contacts have been mined (termed potholes).
- Contact-style mineralisation at the base of the intrusion (Platreef-type) occurs mainly in the Northern Limb.

In general, within the Northern Limb, the Platreef comprises a variably layered, composite norite–pyroxenite–harzburgite intrusion that lies at the base of the Bushveld Complex, in contact with metasedimentary and granitic floor rocks. McDonald and Holwell (2011) reviewed the major literature on the Platreef and Northern Limb, and have concluded:

- The Platreef remains a complex and enigmatic deposit.
- Stratigraphic relationships with other stratiform deposits such as the Merensky and UG2 reefs have been suggested.
- The extent to which the Northern Limb was connected to the rest of the complex across the Thabazimbi–Murchison Lineament (refer to Figure 7.1 where this is shown as the TML fault) remains to be established.
- The Platreef represents a complex of sills intruded into basement granite-gneiss, Transvaal Supergroup sediments or pre-Platreef Lower Zone intrusions.
- Intrusive relationships of the Main Zone gabbronorites, into solidified and deformed Platreef, removes the Main Zone as a source of metals for the Platreef.
- Mineral chemistry, bulk geochemistry, and Sr, Nd, and Os isotope geochemistry of the Platreef are most consistent with an ultramafic (Critical or Lower zone) component.
- Platreef Nd values and 187Os/188Os initial isotope ratios overlap clearly with the Merensky Reef but not the UCZ.
- Conventional and mass-independent S isotopes suggest a primary mantle source of S that was overprinted by the addition of local crustal S where Platreef intruded pyrite-rich shales. Assimilation of S is viewed as a modifying process, not as the primary trigger for mineralisation.

Two emplacement models are considered to be the most likely to explain the mineralisation (McDonald and Holwell, 2011):

- Platreef sulfides may have been derived from the same magma(s) that formed the Merensky Reef in the central part of each Bushveld limb and which were injected up and out along intrusion walls as the chamber expanded.
- Alternatively, the sulfides may have formed in pre-Platreef staging chambers for Lower Zone intrusions where they were upgraded by repeated interactions with batches of Lower Zone magma. The sulfides were subsequently expelled as a crystal-sulfide mush by an early pulse of Main Zone magma that broke into and spread through the earlier Lower Zone magma chambers.

8.1 Comments on Section 8

The current deposit model preferred by Ivanhoe for the Platreef Project favours the stratiform Merensky-style model with the additional complexity of the UCZ coming into direct contact with footwall sedimentary units through melting and assimilation processes.

Mr Kuhl considers that the mineralisation delineated at the Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS farms is typical of Platreef-style mineralisation within the Northern Limb of the Bushveld Complex. As a result of the Ivanhoe interpretations, Mr Kuhl judges that exploration programmes using the Merensky-reef analogue are appropriate to the deposit style.

9 EXPLORATION

9.1 Grids and Surveys

Over the life of the Project to date, two different coordinate systems have been used:

- Hartebeesthoek 1994 LO29 national coordinate system.
- Local Platreef Project coordinate system.

Currently all information in the Project database has been converted to the Hartebeesthoek 1994 LO29 national coordinate system. Depending on the location within the Project area, drillholes may have negative coordinates.

9.2 Geological Mapping

Original detailed geological outcrop mapping was completed by Ivanhoe personnel in 2002 at 1:5,000 scale and was supported by trenching and percussion drilling in areas with no outcrop.

This initial exercise was expanded upon in 2013 to include near-surface information gained from close spaced drilling. The depth of weathering is controlled mainly by rock type, structure and alteration and is most pronounced along the mafic to ultramafic units and along the major fault traces at surface. The complete strike of the Platreef, on the two farms, is now mapped in detail with special attention given to hanging-wall and footwall contacts, the near surface occurrence of xenoliths and the extents of metasediment assimilation. Mapping of the Main Zone lithologies was only done in areas of excavation and making use of geophysical datasets, as the Main Zone outcrop is limited to boulders and scree.

This work has identified of at least 800 m of LZ cumulate rocks and intercalated metasedimentary rocks along the strike length of the Platreef Project (Yudovskaya et al. 2013). The intercalated metasedimentary rocks occur as interlayers (rafts) between the TCU and the Archean basement. A geological map combining the field mapping with drillhole information was included as Figure 7.3.

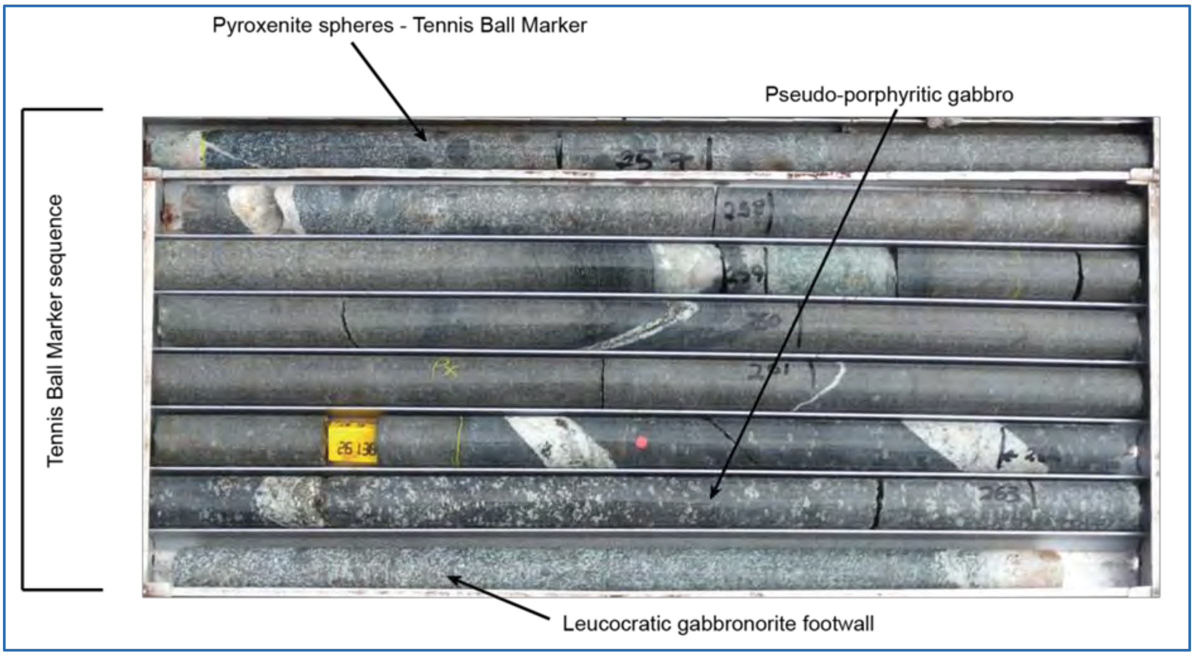
Systematic modelling of MZ and UZ lithologies or a model of the granite dykes had never been undertaken. Recent work has enabled Platreef geologists to confirm trends on the magnetics image.

Well-defined anorthosite layering in the upper portion of the Main Zone, and the lower Main Zone layering (TBM and BMGN) are distinct and define the orientation of the layered intrusion (Figure 9.1 and Figure 9.2). This is a major distinction from the orientation of Platreef-type mineralised layers. The Main Zone layers commonly strike at 334° and are generally uninterrupted for over 6 km. The apparent sigmoidal pattern which dominates the first vertical derivative image correlates strongly with the granite dyke model. These dykes change orientation from 290° in the north to 318° in the central part of Turfspruit before regaining a 290° trend in Macalacaskop. The granitic dykes are the cause of a ladder-like magnetic pattern, due to their trend at a slightly oblique angle to the magmatic layering.

Figure 9.1 Core Showing TBM

Ivanhoe, 2016

Figure 9.2 TBM Sequence in UMT070



Ivanhoe, 2016

9.3 Geochemistry

A geochemical study has been completed that focused on the correlation of the stratigraphic sequence intersected by drillholes below the MZ contact (Grobler et al., 2016).

Geochemical major, trace and rare earth element data for six core holes were investigated. These holes are mostly from the down-dip Zone 3 area where better-developed UCZ stratigraphy could be identified. One hole is from the Zone 5 Madiba area sited towards the southern extremity of the property, and one hole is located within the well mineralised north-western part of Zone 1. PGE, Ni, Cu, Cr and S data available for most exploration holes were further used in an attempt to identify geochemical signatures for the different stratigraphic units.

Drilling by Ivanplats intersected magmatic cyclical stratigraphy below the Main Zone in deeper areas towards the west of the Project area. Distinct, continuous magmatic layers could be identified in this area and included the prominent T1 and T2 mineralised layers as identified by Ivanplats. The approach was to first establish the geochemical characteristics of these relatively uncontaminated and least altered lithologies. An attempt was also made to correlate the findings with the rest of the Bushveld Complex.

The Turfspruit samples from the intersection below the Main Zone contact were found to exhibit geochemical trends similar to those reported for UCZ samples from the eastern and western Bushveld Complex.

9.4 Geophysics

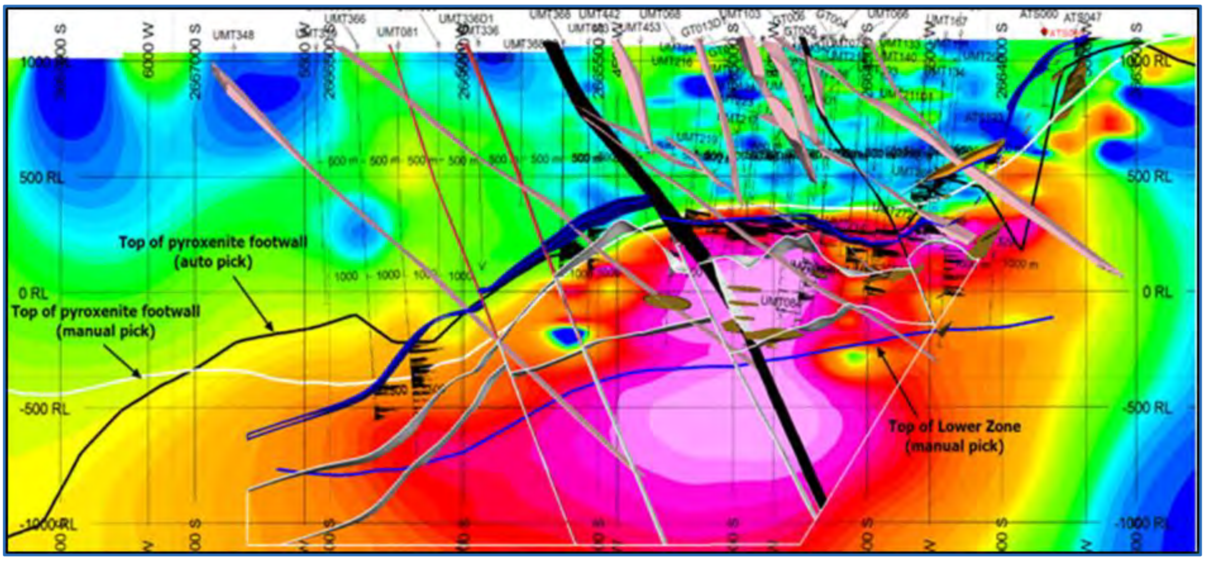
Geophysical survey methods at the Platreef Project have included aeromagnetics, gravity gradiometer and a number of downhole geophysical methods including caliper; self-potential (SP)/point resistance (PR); electrode-array-focussed resistivity (EAL); magnetic susceptibility (Msus); temperature/conductivity; fall-waveform-sonic (FWFS); acoustic televiewer (ATV); optical televiewer (OTV); induced polarisation (IP); density; neutron; induction and vertical seismic profile (VSP).

In 2012, Ivanhoe acquired 130 km^2 of Falcon gravity data that were geologically constrained and inverted by N. Williams of Ivanhoe Australia Ltd. using proprietary algorithms. The Falcon airborne gravity gradiometer system was developed by BHP Billiton, and all rights were purchased by Fugro Airborne Surveys in 2009. A 3D isosurface was generated, representing the depth to density contrast of the geological contact between the gabbronorite of the Main Zone and the T1 pyroxenite of the Turfspruit Cyclic Unit (Figure 9.3 and Figure 9.4). The Falcon data supplement previous geophysical work conducted in the Platreef Project area and indicates that the Flatreef could potentially extend to the south of Zone 1 for >3 km.

A 3D seismic survey was run by seismic specialist company CGG, headquartered in Paris, France, in Q4 2013 for the purpose of confirming and enhancing the structural interpretation in the planned initial production area. The survey included a number of vertical seismic profiles (VSPs). The findings to date have been used to support the structural model and the processing of the seismic data.

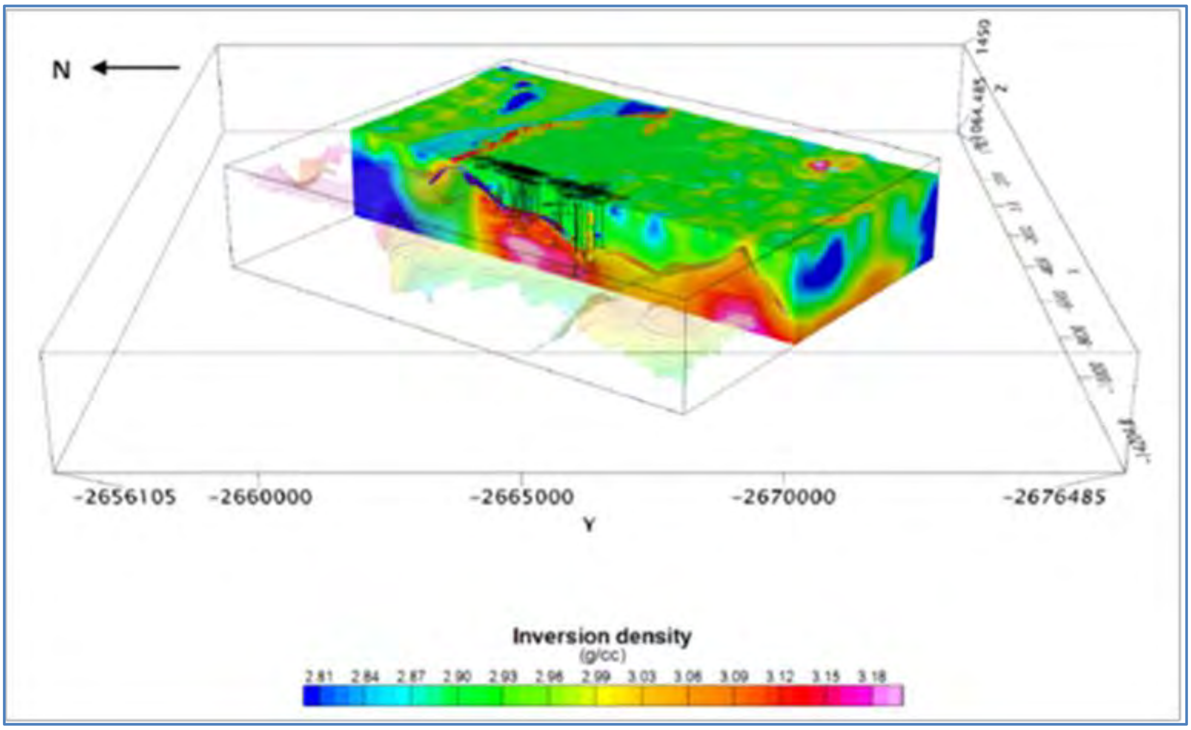
In the first quarter of 2015, Velseis (Pty) Ltd reprocessed the 3D seismic data acquired by CGG. The result of this work was a depth-converted volume constrained by the VSP data. Figure 9.5 shows a cross-section with a depth-converted seismic image showing the correlation between the xenoliths (drillholes discs) and strong reflection events. Low-angle granitic veins (grey) show correlation with seismic reflector events in some instances. The T2 unit (blue and red) and faults (red) generally do not show up as obvious features.

Figure 9.3 Geologically Constrained Falcon Gravity Inversion (Dip Section 10)



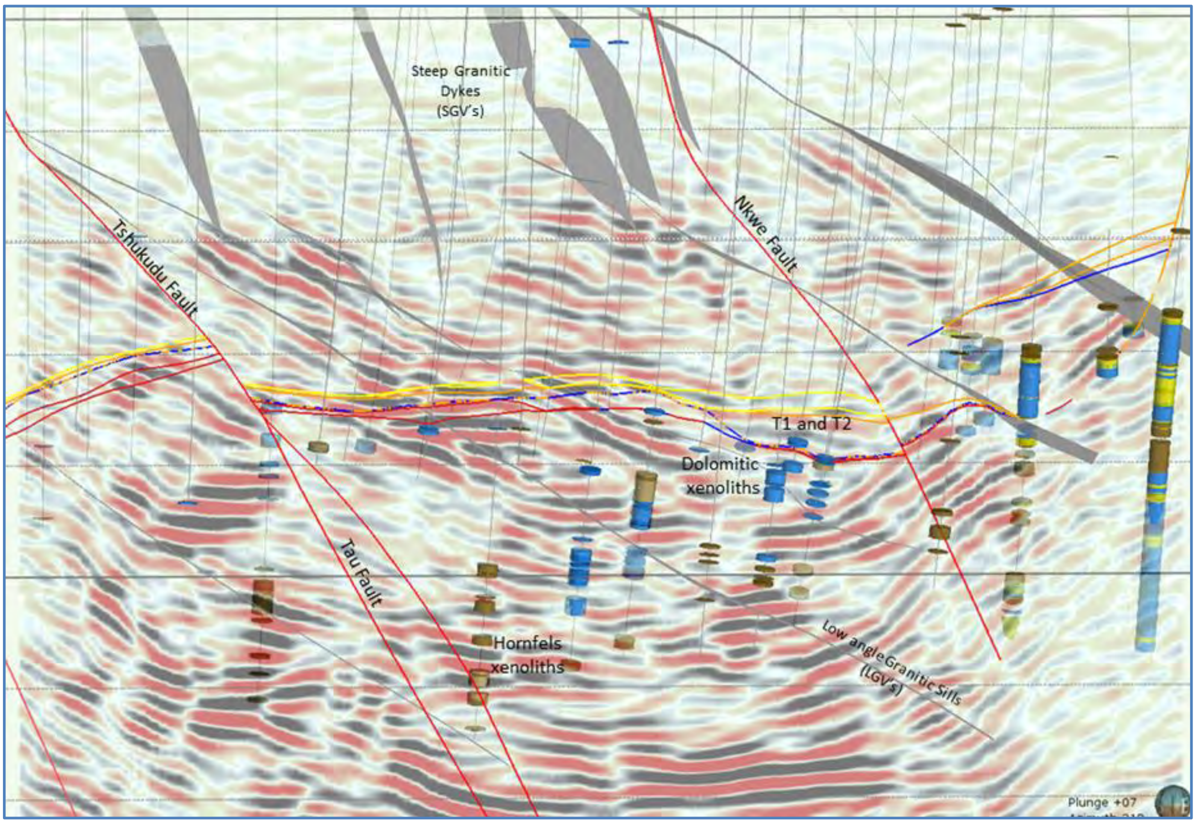
Ivanhoe, 2016; Dip Section 10 indicated on Figure 9.6.

Figure 9.4 Geologically Constrained Falcon Gravity Inversion Interpretation



Ivanhoe, sourced from Williams, 2012; Inversion sliced along a north-east oriented section. Image shows computed depth to >2.97 g/cm³ isosurface which maps the gabbronorite/pyroxenite contact and thereby depicts the approximate structure of the mineralised reef.

Figure 9.5 Dip Section 10.0 from Velseis Depth Converted Data



Ivanhoe, 2016; Location of section line is indicated in Figure 9.6.

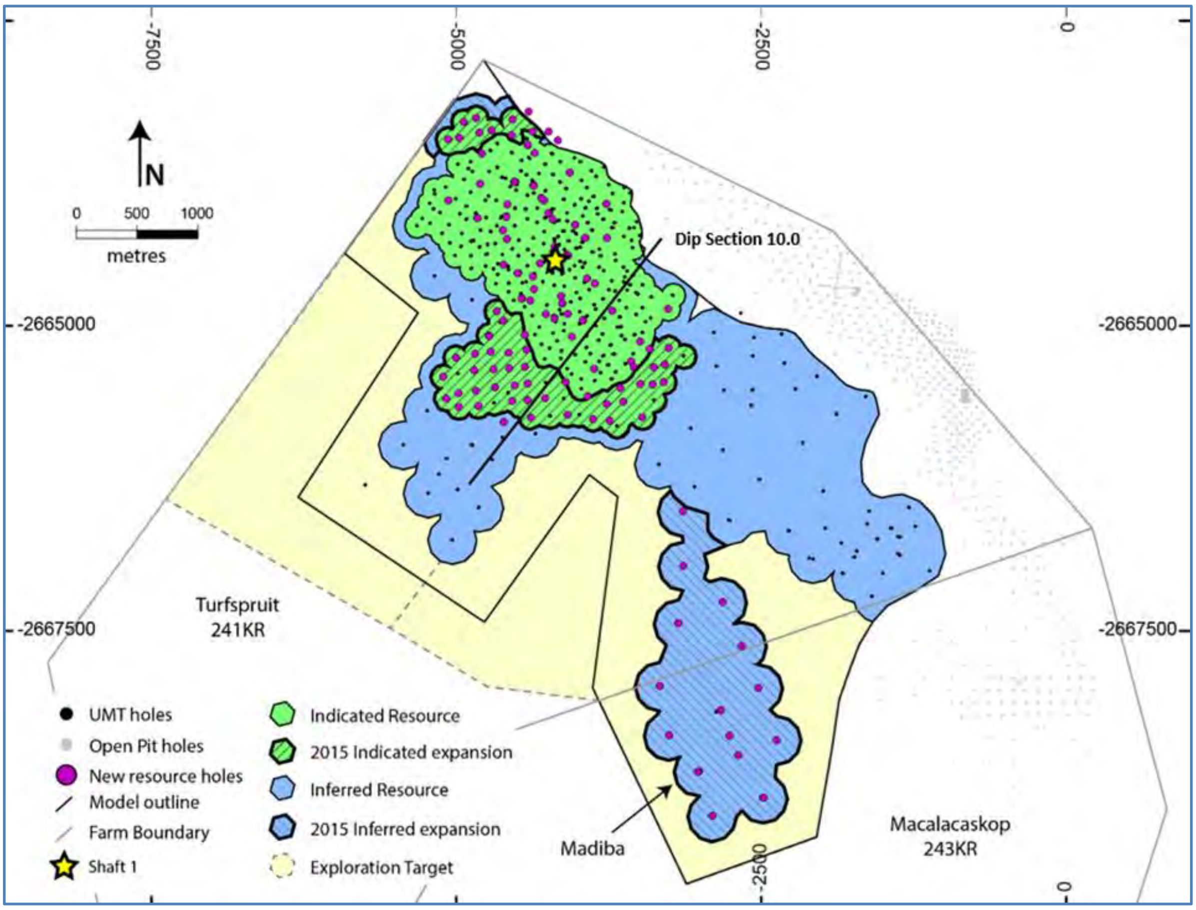
9.5 Petrology, Mineralogy, and Research Studies

Several MSc and PhD academic studies were conducted by various universities over the last three years in an attempt to test the proposal that the TCU is analogous to the Merensky Cyclic Unit (MCU) of the main Bushveld Complex (Smart, 2013; Kekana, 2014; Marquis, 2015 and Nodder 2015; Kvadsheim, 2017; Vekić, 2017; Mayer, 2018; Keir-Sage, 2019; Stephenson, 2019; and Abernethy, 2019). These major, trace and REE studies unequivocally showed significant similarities between the TCU and MCU rock units. They also highlighted signs of geochemical contamination between magmatic and metasedimentary rock units.

9.6 Exploration Potential

The Platreef mineralisation remains open along strike and down-dip. There is opportunity to expand the extent of known mineralisation with further drilling, down dip. Subsequent limited drilling within the Zone 5 area (see Figure 7.4) has shown significant grade values as part of the extension of the Flatreef towards the south and served to confirm the deductions made from the Falcon dataset (see Figure 9.6).

Figure 9.6 Plan Map Indicating Potential Exploration Areas Open Along Strike and Dip



Ivanhoe, 2016

9.7 Comments on Section 9

In the opinion of Mr Kuhl, the exploration programmes completed to date are appropriate to the style of the mineralisation within the Platreef Project area.

The exploration programmes conducted by Ivanhoe are appropriate to support Mineral Resource estimation.

10 DRILLING

10.1 Drill Summary

Drilling on the Platreef Project has been undertaken in two major phases; the first from 2001-2003 is termed the open-pit programme (designated AMK at Macalacaskop 243 KR and ATS at Turfspruit 241 KR/Rietfontein 2 KS). The open-pit programme drillholes are located in Zone 4 (see Figure 7.4).

The second phase commenced in 2007, and the most recent campaign ended February 2015. This second drill phase is termed the underground programme, is designated UMT (including Bikkuri), and nearly all drilling is on Turfspruit 241 KR. These drillholes are situated in Zones 1–3 and Zone 5. There were two drillholes (PUM001 and PUT001) drilled in 2012 which are located in Zone 4. These drillholes are grouped with the open-pit drillholes.

The database (closed on 24 July 2015) includes 578 drillholes (196,213 m) from Phase 1 (including all redrills and deflections). The Phase 1 drilling was completed in support of mineral resources amenable to open pit mining methods (See Section 6).

A total of 57 (26,790 m) drillholes from Phase 1 were relogged and included in the current resource models to aid in the geological modelling.

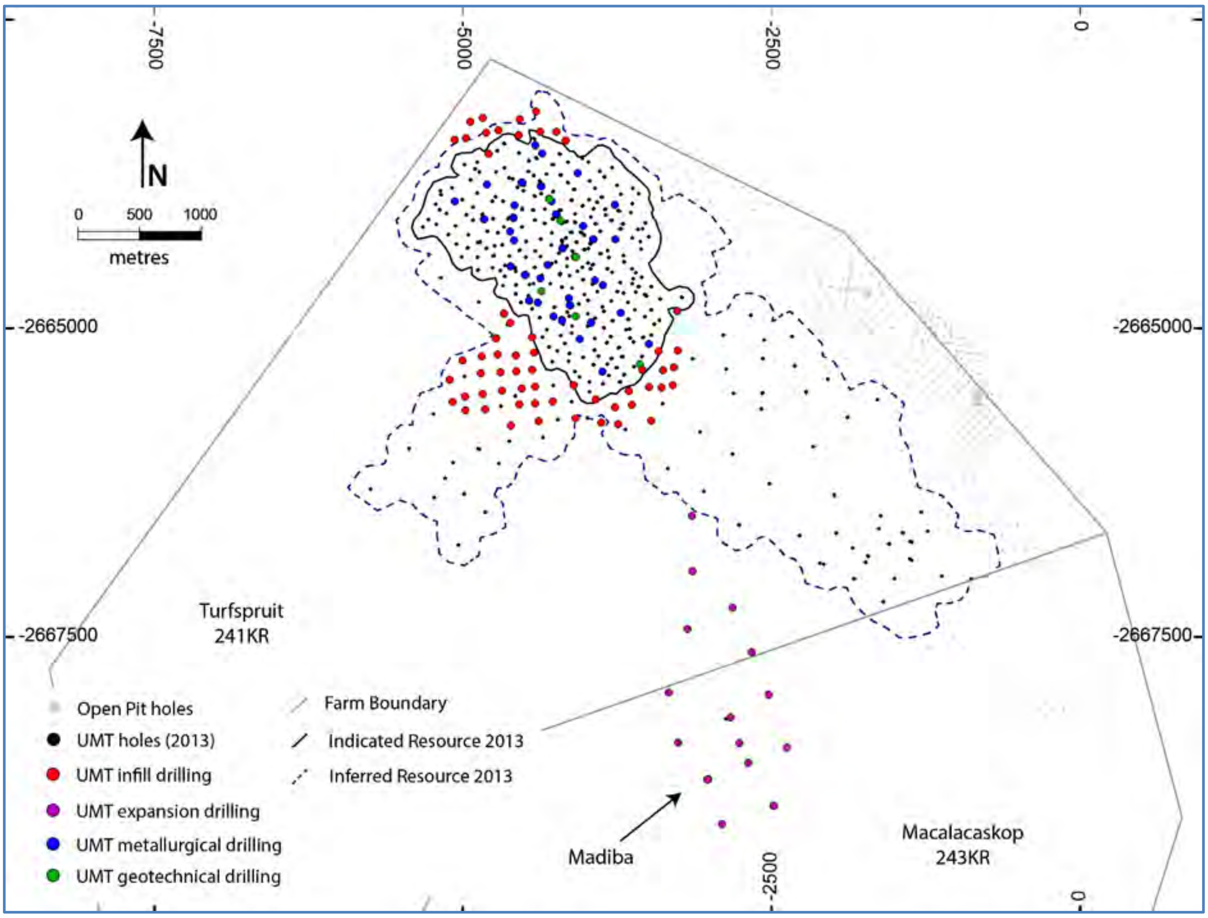
The database includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totalling 501,638 m completed by 11 February 2015. No drilling for resource estimation purposes has occurred between this date and the Report effective date; however, assay data from three drillholes has since become available; two geotechnical holes drilled down the position of Shaft 1 (GT008) and Shaft 2 (GT017) and one metallurgical drillhole (TMT015). Depths for deflections are calculated based on point of deflection and do not include the mother or pilot hole portion. This includes 33 drillholes and deflections (9,181 m) completed for geotechnical purposes and 62 drillholes and deflections (23,001 m) completed for metallurgical purposes (Figure 10.1).

The Phase 2 drilling is summarised by Zone:

- Geology, Zone 1: 321 drillholes (320,225 m) and 26 deflections (13,047 m);
- Geology, Zone 2: 47 drillholes (62,020 m) and 9 deflections (5,104 m);
- Geology, Zone 3: 46 drillholes (51,386 m) and 10 deflections (2,841 m);
- Geology, Zone 5: 15 drillholes (14,235 m) and 5 deflections (598 m);
- Geotechnical Drilling, Zone 1: 26 drillholes (7,643 m) and 7 deflections (1,538 m);
- Metallurgical samples, Zone 1: 14 drillholes (13,206 m) and 48 deflections (9,794 m).

The most recent Platreef resource drilling was completed 11 February 2015.

Figure 10.1 Drill Collar Location Plan with New Drilling Since 2013



Ivanhoe, 2016; Drilling shown on plan was current as of 11 February 2015.

10.1.1 Drilling Completed 2013–2015

Drillholes completed since 2013 – 2015 includes 97,736 m (99 drillholes and 58 deflections). The drilling was completed for the purposes of geotechnical investigations, metallurgical samples, exploration infill and exploration expansion (Table 10.1). Figure 10.1 shows the locations of drilling completed 2013 – 2015.

Table 10.1 Drilling Completed 2013 – 2015

Drill Type	Drillholes and Deflections	Metres
Geotech	7 GT holes and 3 deflections	7,599
Metallurgy	16 TMT drillholes, 26 TMT deflections, 22 UMT deflections	22,037
Exploration Expansion	14 UMT drillholes and 2 deflections	10,641
Exploration Infill	62 UMT drillholes and 5 deflections	57,459
Total	99 Drillholes and 58 deflections	97,736

10.1.2 Zone 4

Drillhole prefixes for the open-pit programme are prefixed AMK; ARF; ATM; ATS; DTS; GT (001-003); ITS; PA; PUM; PUT; STM, and STT. Most drillholes were collared as vertical drillholes with the exceptions of nine AMK drillholes which were completed at 45° to 60° inclinations and three ATS geotechnical holes completed at a 50° inclination. AMK drillholes were drilled nominally on a 100 m north–south-oriented local grid at Macalacaskop 243KR, whilst the ATS initial drill spacing is approximately 120–140 m and generally follows an east–north-east oriented drilling grid that conforms to the street plan in the Tshamahansi Township.

In addition to the exploration drilling, a cross-pattern of 21 vertical drillholes (30 m spacing) was completed for geostatistical purposes (the geostatistical grid). A mining simulation drill grid was completed at a 10 x 10 m drill spacing (DTS drillholes), and an infill programme (ITS drillholes) was completed locally to increase the drill density to approximately 100 x 75 m or 75 x 75 m.

10.1.3 Zones 1–3

Several drilling campaigns have been completed since 2007 in these zones. Ivanhoe's initial underground drill campaign at Zone 2 in 2007 was to test for mineralisation down-dip of Zone 4 and was completed in 2009. In April 2011, Ivanhoe initiated a programme to expand the geological knowledge around the Flatreef and to perform infill drilling in Zone 1 to approximately 100 x 100 m spacing.

From 2007–2015 a total of 486,806 m was drilled from 554 drillholes. Drillholes were collared as vertical up to and including UMT105; after that, holes were drilled at an 85° inclination with the exception of UMT330 which had a 60° inclination, UMT439 with an 83° inclination, UMT463 with an 81° inclination, and UMT464 had an 80° inclination. Drillhole spacing is nominally 400 x 400 m or 400 x 200 m with local 200 x 200 m coverage and 100 x 100 m coverage in much of Zone 1. There are a few areas where the spacing is somewhat wider and/or irregular (400–500 m between holes).

10.1.4 Zone 5

In October 2012, further exploration drilling for the purpose of extending the geological knowledge of the Flatreef area to the south of Zone 3 was initiated. A total of 20 drillholes (14,832 m) were completed in Zone 5. The drillholes were collared as vertical and completed on a nominal drill spacing of 400 x 400 m.

10.2 Drill Methods

All drilling has been completed by diamond drill coring methods. Drill programmes have been completed primarily by contract drill crews, supervised by Ivanhoe's geological staff.

10.2.1 Zone 4

Drilling was conducted between 2001 and 2003 by Rosond Drilling (an international contract drilling company). Drill-rig types included Longyear-44, Longyear-38, Boyles-37, Tone-TEL and Rocor/Diamech-262. Wire-line equipment extracted NQ2 (50.5 mm core diameter) and HQ (63.3 mm) core, and a limited amount of geotechnical drilling was completed with oriented NQ3 (44.9 mm) core from stabilised triple-tube core barrels. Metallurgical sample holes were completed with TNW-size (60.3 mm diameter) core. Completed holes were capped using a 1.5 m length of sealed steel pipe welded to the drillhole casing.

10.2.2 Zones 1–3 and Zone 5

Drilling of the underground deposit began in 2007, with Zone 1, 2 and 3 drilling ending in 2015. Zone 5 is the latest explored area, and drilling ended in October 2014. All drilling extracts HQ (63.3 mm), NQ (48 mm) or BQ (36 mm) sized diamond drill core. The holes were all near-vertical at their collars, but with depth the holes tend to incline less steeply. For the UMT holes (excluding deflections), the average hole length is 1,047 m; the minimum hole length is 413.5 m, and the maximum hole length is 1,973 m.

The underground-deposit drill programme has shown the Platreef extending to at least a depth of 1,525 m, and the Platreef is 300 m to 600 m thick at Turfspruit 241 KR. The average depth to the floor rocks (below the base of Platreef) is approximately 1,200 m, and the depth to the floor rocks ranges from 300 m to 1,500 m.

Completed holes were capped using a 1.5 m length of sealed steel pipe welded to the drillhole casing with drillhole labels inscribed on the drill caps.

10.3 Geological Logging

Standardised geological core logging conventions were used to capture information from the drill core. Detailed geological logging of drill core was completed daily by geologists onto log sheets. There has been an improvement in the style of logging from the historic work on the open-pit drilling programme (Zone 4) to the current underground drilling programme of Zone 5. The improvement in core logging provides more accurate and detailed information.

Platreef staff performed core handling from drill site to storage. Each core box was photographed using a digital camera. The photographs are stored on a network server and duplicate CD-ROM media. After geological logging, sample intervals were marked on the core, and drill core was sawn longitudinally for sampling.

After sampling, the remaining half core is archived in one metre-length galvanised-plate core boxes. Storage facilities consist of lockable brick and corrugated steel sheds where the core boxes are placed on 2 m high pre-fabricated core racks for ease of access.

Mr Kuhl has reviewed the local geology, including core logging and interpretations and find the data collection to have been done in a professional manner that can support Mineral Resource estimation and Project development.

10.3.1 Zone 4

Geological core logging involved the recording of lithology; grain size; type and degree of alteration (low, medium, or high); type and visible percentage of sulfide (pentlandite, pyrrhotite, chalcopyrite, and pyrite); relative sulfide ratios and structural data. Data captured include lithology by standardised abbreviation; alteration by type and relative degree; biotite alteration as a modal percentage and visible sulfide types as a total modal percentage. Structural data were noted, core axis angles taken, and RQD data were captured at maximum 10 m intervals for each drillhole.

Logs were then independently double-entered into Excel spreadsheets, and upon validation stored in an Access database.

10.3.2 Zones 1–3 and Zone 5

The detailed information recorded includes lithology; stratigraphic unit; texture; grain size; (bottom) contact type; angle to the core axis; alteration and structure which are all mandatory entries; there is an option for the geologist to record a comment(s).

The geology logs are commonly captured in a computer pad and imported into an acQuire database. Once the geology log is completed, the logging geologist reviews the core and core log with the Ivanplats geology staff.

10.4 Core Recovery

The core recovery within the first few metres of boreholes (approximately 5 m) is poor in most cases due to the associated soil horizon classified as overburden. Poor recovery occasionally extended to about 30 m depth due to the weathering of bedrock. However, in the majority of instances, core recovery improved considerably once drilling reached the Main Zone hanging wall, reef horizon (T1 and T2) and footwall rocks, and in these units, was commonly 100%. The recoveries only show a substantial decrease within faulted/sheared zones.

10.5 Collar Surveys

A contracted certified land surveyor used a differential Trimble GPS system to conduct collar surveys on all completed holes. Stations were tied in with survey stations established by the National Survey General Directorate.

Drillhole coordinates were given in the Hartebeesthoek 1994 LO29 national coordinate system (refer to Section 9.1).

10.6 Downhole Surveys

There are 34 drillholes in Zone 4 without downhole surveys. All unsurveyed drillholes are vertical and range in depth from 7–583 m. The ATS and AMK drillholes were downhole surveyed using multi-shot Reflex and Maxibor instruments. Multiple survey shots were taken at 3–6 m intervals downhole.

Downhole deviation surveys for the UMT drilling were completed by independent downhole survey technicians using gyroscopic (gyro) and/or electronic multi-shot (EMS) instruments. Surveys are recorded downhole at 3–5 m intervals. In Zones 1–3 and Zone 5, there are 21 drillholes without surveys. 15 drillholes were drilled for geotech purposes and are less than 30 m in depth. Five drillholes were deflections with depths ranging from 28 to 780 m. There are five UTM holes (deflections) without downhole survey data and one UMT drillhole without downhole surveys.

Where both an EMS and a gyro survey were completed, the gyro survey was assumed to be more accurate, and therefore in most cases was used in the geological model. There are 181 drillholes where the EMS survey has been selected, due to erroneous or uncompleted gyro surveys. A memorandum from site (Ivanplats, 2015) discussing a review of the downhole surveys states that EMS downhole surveys were selected over gyro survey results for 70 drillholes.

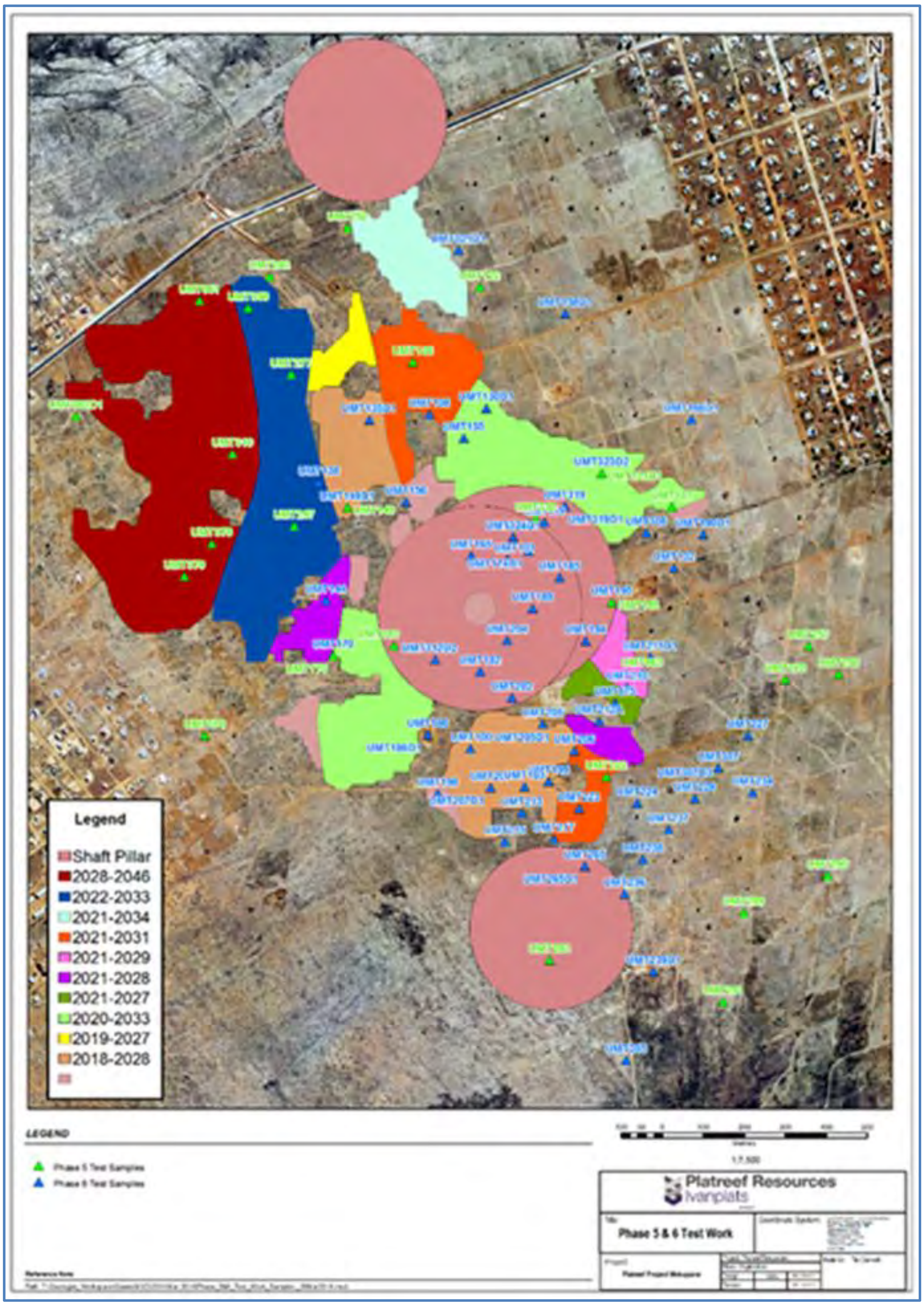
10.7 Metallurgical Drilling

The area sampled was Zone 1, and all UMT borehole data were incorporated in order to define a representative characteristic grade distribution per Geomet unit as defined by the geologists. The lithological basis used in sample selection is the main Geomet units as modelled, namely the T1, T2U, and T2L.

Initial borehole selection was done with the aim at being spatially representative. This was achieved using plots of all the UMT holes and was later confirmed with grade and thickness variation plots based on the 2 g/t 2PE+Au grade shell cut-off data. The selection criteria included 2PGE+Au grade, Ni grade, Pt/Pd ratio, and rock type.

The drill map below indicates the holes selected for the Mintek (blue markers) and SGS (green markers) laboratories on which the metallurgical testwork is based (Figure 10.2).

Figure 10.2 Metallurgical Drillhole Map



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10.8 Summary of Drill Intercepts

Selected drill intercepts showing typical grades and thicknesses of mineralisation in the various model areas are included as Table 10.2.

Table 10.2 Drill Intercept Example Summary Table

Drillhole	From (m)	To (m)	Drilled Length (m)	Azimuth (°)	Dip (°)	Top of Interval			Pt (g/t)	Pd (g/t)	Au (g/t)	2PE+Au (g/t)	Cu (%)	Ni (%)
						Elevation	Easting	Northing						
ATS – Area where Mineral Resources amenable to open pit mining methods are estimated														
ARF020	20.62	29.32	8.70	0.0	-90.0	1131.0	-950.9	-2665303.0	1.60	1.52	0.51	3.63	0.15	0.44
Includes	20.62	23.39	2.77	0.0	-90.0	1131.0	-950.9	-2665303.0	2.42	2.27	1.02	5.72	0.20	0.62
ARF020	140.45	146.67	5.92	0.0	-90.0	1010.7	-950.9	-2665303.0	1.37	0.82	0.17	2.37	0.11	0.21
Includes	140.45	142.68	1.93	0.0	-90.0	1010.7	-950.9	-2665303.0	2.64	1.28	0.23	4.15	0.09	0.15
ARF043	202.81	219.08	16.27	0.0	-90.0	947.6	-1071.3	-2665130.4	0.63	1.39	0.25	2.26	0.51	0.63
Includes	213.96	219.08	5.12	0.0	-90.0	936.6	-1071.5	-2665130.4	1.35	1.64	0.38	3.37	0.51	0.29
ATS046	424.79	467.05	42.26	0.0	-90.0	717.5	-1348.3	-2665260.0	0.99	1.49	0.28	2.75	0.49	0.42
Includes	453.48	466.09	12.61	0.0	-90.0	688.4	-1348.4	-2665260.0	1.87	2.79	0.48	5.14	0.69	0.57
AMK – Area where Mineral Resources amenable to open pit mining methods are estimated														
AMK030	134.70	172.79	38.09	0.0	-90.0	990.66	-524.7	-2668096.3	0.96	1.26	0.22	2.45	0.18	0.35
Includes	137.73	171.76	4.03	0.0	-90.0	957.63	-524.7	-2668096.3	1.15	1.68	0.28	3.11	0.14	0.32
AMK051	207.84	240.62	32.78	0.0	-90.0	915.25	-740.9	-2667993.8	0.80	0.84	0.14	1.78	0.11	0.27
Includes	226.87	230.87	4.00	0.0	-90.0	896.23	-740.9	-2667993.8	1.16	1.30	0.19	2.64	0.11	0.31
AMK081	330.59	363.93	33.34	0.0	-90.0	793.89	-825.3	-2667803.0	1.11	1.38	0.19	2.69	0.16	0.26
Includes	330.59	344.32	13.73	0.0	-90.0	793.89	-825.3	-2667803.0	1.47	1.77	0.25	3.49	0.20	0.35
UMT – Area where Mineral Resources amenable to underground mining methods are estimated														
UMT026	1232.00	1298.33	66.33	0.0	-90.0	-129.984	-2566.7	-2665533.5	1.35	1.27	0.18	2.80	0.09	0.24
Includes	1232.00	1294.33	62.33	0.0	-90.0	-129.984	-2566.7	-2665533.5	1.40	1.29	0.19	2.89	0.09	0.24
Includes	1268.50	1284.50	16.00	0.0	-90.0	-156.084	-2570.5	-2665535.0	1.24	1.90	0.22	3.36	0.11	0.35
UMT039	803.85	889.64	85.79	0.0	-90.0	246.7	-4368.2	-2663815.7	1.55	1.81	0.21	3.57	0.10	0.23
Includes	843.85	889.64	45.79	0.0	-90.0	249.5	-4371.3	-2663816.6	0.58	0.50	0.13	1.21	0.06	0.14
UMT056	772.53	858.53	86.00	0.0	-90.0	318.8	-3983.9	-2664992.8	1.32	1.33	0.21	2.86	0.17	0.34
Includes	772.53	808.15	35.62	0.0	-90.0	318.8	-3983.9	-2664992.8	2.34	2.20	0.31	4.84	0.21	0.46
Includes	772.53	785.26	12.73	0.0	-90.0	318.8	-3983.9	-2664992.8	4.74	3.81	0.51	9.06	0.17	0.43
UMT217	805.00	822.00	17.00	270.0	-85.0	312.1	-4112.9	-2665049.7	2.73	2.29	0.25	5.28	0.11	0.26
Includes	805.00	816.00	11.00	270.0	-85.0	312.1	-4112.9	-2665049.7	3.96	3.21	0.36	7.54	0.14	0.32
Includes	805.00	814.00	9.00	270.0	-85.0	293.3	-4112.9	-2665049.7	4.65	3.65	0.42	8.73	0.15	0.34
UMT281	832.00	845.00	13.00	270.0	-85.0	277.8	-14324.7	-2670596.7	1.14	1.08	0.18	2.39	0.15	0.25
Includes	835.70	843.27	7.57	270.0	-85.0	272.2	-14324.7	-2670596.7	3.68	3.43	0.55	7.66	0.47	0.77
UMT312	767.00	790.00	23.00	270.0	-85.0	334.9	-14324.7	-2670596.7	1.70	1.75	0.26	3.71	0.19	0.33
Includes	768.00	789.00	21.00	270.0	-85.0	329.4	-14324.7	-2670596.7	1.80	1.85	0.26	3.91	0.20	0.35
Includes	768.00	778.00	10.00	270.0	-85.0	329.4	-14324.7	-2670596.7	2.31	2.40	0.35	5.06	0.24	0.41
UMT-BIK – Area where Mineral Resources amenable to underground mining methods are estimated														
UMT145	412.98	415.98	3.00	280.0	-85.0	701.6	-4182.0	-2663613.6	0.93	0.58	0.18	1.69	0.10	0.21
UMT172	462.00	476.00	14.00	272.0	-85.0	654.9	-3893.7	-2663874.5	1.52	1.30	0.34	3.16	0.22	0.36
Includes	463.00	468.00	5.00	272.0	-85.0	654.1	-3893.9	-2663874.5	2.18	1.78	0.45	4.42	0.29	0.50
UMT249	416.81	421.38	4.57	267.0	-85.0	701.3	-3866.1	-2663738.5	1.09	0.99	0.25	2.33	0.16	0.31
UMT280	474.57	481.00	6.43	268.0	-85.0	673.9	-3586.8	-2664000.2	1.04	1.12	0.30	2.46	0.24	0.39

10.9 Comparisons of Intercept Positions – Twin Hole Data

A preliminary comparison was made of twin holes, which usually consist of an original hole and a deflection (see Parker 2014). The holes were hung on the base of the Main Zone and then were compared. The average differences in the contact position between the Main Zone and the top and bottom of each of the T1 and T2 between twin pairs range from 4.8 to 7.9 m, with the bottom contact of both the T1 and T2 being more variable in terms of average difference than the top contact.

The block grades used in the resource model are constrained by grade shells that have been smoothed by re-blocking (averaging) over 2 m vertical heights. MTS cautions that grade shells can result in an overestimate of recovered grade unless a suitable approach is taken in stope design and the application of modifying factors. This, and the necessity for close-spaced grade control sampling to establish stope boundaries should be evaluated in more detail in future more detailed studies.

10.9.1 Comparisons of Down-Hole Lengths, Grades – Twin Hole Data

A preliminary comparison of the down-hole lengths, Ni grade and 3PE grade between the twin holes was performed (see Parker 2014). The correlation coefficients were found to be weak for the T1MZ (1 g/t 3PE+Au shell) because the zone is thinner than the T2MZ, and fewer assay intervals are averaged into intercepts. For the T2MZ, the correlation was generally good.

10.10 Comments on Section 10

10.10.1 MTS Comments

In the opinion of Mr Kuhl the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programmes are sufficient to support Mineral Resource estimation as follows:

- Core logging meets industry standards for PGE–Au–Ni–Cu exploration.
- Collar surveys and downhole surveys have been performed using industry-standard instrumentation.
- Recovery from core drill programmes is acceptable to allow reliable sampling to support Mineral Resource estimation.
- Depending on the inclination of the drillhole, and the dip of the mineralisation, drill intercept widths are approximately equivalent to true widths for most UMT drillholes. Drill orientations are generally appropriate for the mineralisation style. In the areas potentially amenable to open-pit mining, vertical holes have been spaced closely enough (ATS) so that the geological units and trends to grade can be defined. Elsewhere, the spacing of the holes is wider, and their angle with the Platreef approaches 45°. Ivanhoe should consider drilling angled holes when infilling the more steeply dipping sections of the Platreef.

- Drill orientations are shown in the example cross-sections included in Sections 7 and 14 and can be seen to appropriately test the mineralisation. The sections display typical drillhole orientations for the deposits.
- Preliminary analysis indicates the twin data are more variable with respect to position than they are for length and grade. Following the reef will potentially be much more challenging than making local grade estimates.

10.10.2 Metallurgical

It is the opinion of the qualified person responsible for the metallurgical aspects of the Platreef Project, Mr. Val Coetzee, that, based on current understanding and information provided by the geological team, adequate sample to prepare composite domain samples was provided for metallurgical testwork and mineralogical analysis for the purposes of a feasibility study.

10.10.3 Geotechnical

The geotechnical aspects of the project are discussed in Section 16.1.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

From the time of Ivanhoe's initiation of the Platreef Project to date, Project staff members employed by Ivanhoe were responsible for the following:

- Sample collection.
- Core splitting.
- Sample despatch to the analytical laboratory.
- Sample storage.
- Sample security.

11.1 Sampling Methods

The limited geochemical sampling of trenches, performed early in the exploration programme, was superseded by core drill data; therefore, geochemical sampling is not discussed further.

Drill core is sawn in half using a wet saw. A study completed during 2011 by Long (2011c), which reviewed the differences between recovered and assayed fines lost during sawing found no significant difference in the grades of the elements of interest in the fines compared to their associated core samples.

11.1.1 Assay Sampling

AMK and ATS Sampling

AMK and AST drilling was completed to support Open Pit Mineral Resources. Ivanplats is no longer considering the open-pit option. A detailed description of the AMK and ATS samplings is available in the September 2012 Technical Report (Parker et al., 2012).

UMT Sampling

For underground drilling of the UMT deposit, assay sampling was initiated 5 m above the Platreef (in the Main Zone) and extended, for most drillholes, 20 m into the floor rocks. All drill core within the Platreef was sampled for assaying.

Sampling is completed by Ivanhoe employees based at the Platreef Project offices in Mokopane. Prior to sampling, core loss and core measurements are checked and confirmed by a geologist. The nominal sample length is 1 m, with a maximum of 1.25 m and a minimum of 0.3 m. Samples are broken at lithological contacts. The sample boundaries, lithological breaks and insertion points for blank samples are marked on the core by a geologist.

The sampling supervisor marks the 1 m sample boundaries (start and end) within lithological boundaries. Starting in 2013, a geologist was present for the sample marking and oversaw the sampling process. After mark-up, a photograph of each core box is taken. The photograph includes notations for box number, start and end depths, and the photographer's name. After photography, the core is transferred to the core sawing area.

At the cutting area, a cut line is marked on the core. The drill core is cut bottom-up (downhole to uphole direction). The cut core is placed back in the core box, and the box is placed in the sun to dry. Once dry, the core is moved to a sampling bay.

Each sample is assigned a unique identification number, and each sample batch is assigned a unique number. Sample batches consist of 200–220 samples and include ~±10 standard (certified) reference materials (SRMs or CRMs) and ~±10 blanks. Sample information is written into sample books, and sample bags are marked with sample numbers. Insertion points for standards and blanks are selected. A sample tag and two sample labels (with identical numbers) are placed in the bag of the corresponding sample number. Prior to sampling, the sample bags are inspected to ensure the sample bag, sample tag and sample labels are the same for each bag. Historically, an Excel spreadsheet was constructed that includes the drillhole ID, laboratory ID and sample number. The sampling sheet was captured into acQuire where additional checks are performed on the placement and number of CRMs.

Starting 1 May 2013, an acQuire routine automatically generates the sampling sequence including predetermined QA/QC sample locations. This sequence is reviewed by the geologist prior to collecting the samples.

Sampling is completed by at least two people. Historically, sample weights were captured in the Excel file and loaded into acQuire for the sample batch. Currently the sample weights are entered directly into acQuire. Photographs are taken of each sample displaying the bag's sample number and the sample tags and labels inside the sample bag. Sampling is conducted in sets of 10 samples, and after every 10th sample, the samples are inspected to ensure sample numbers are correct, the acQuire output corresponds, and the sample bags are not damaged.

11.2 Density Determinations

11.2.1 Zone 4

In support of Mineral Resource estimates for a proposed open- pit operation, bulk densities (SGs) were determined for wet and dry rock fragments representing the major lithologies in the AMK and ATS (Zone 4) areas. A selection of 1,088 samples from 230 different drillholes were analysed using conventional water displacement methods. These data are not used for the current Mineral Resource estimate.

11.2.2 UMT Bulk Density

Bulk density determinations from the underground-deposit drilling were completed by Ivanhoe geological staff. Sample lengths of 0.18 m were taken of sawn half-core at a nominal 5 m spacing from each drillhole. The density samples were determined by weight in air and weight in water using the formula:

$$\text{Specific Gravity} = Ma / (Ma\text{-}Mw).$$

where Ma = Mass in Air and Mw = Mass in Water

The database contains over 41,500 density determinations that were recorded from 2007 to 2014 from the underground-deposit UMT exploration drilling programme. These particular densities are representative of the stratigraphic and lithological units used within the geological model.

The different stratigraphic units are shown in Table 11.1, where the proportions of the samples for each broad stratigraphic unit are displayed. Only density determinations from valid holes used in the resource estimation are included in Table 11.1.

There are 18,406 determinations from the hanging wall to the TCU. A total of 3,662 determinations have been taken within the TCU that is the main focus for Mineral Resource estimates, and over 10,034 density determinations from the footwall of the TCU.

There are 4,047 determinations from the hanging wall to the Bikkuri. A total of 323 determinations have been taken within the Bikkuri reef and over 1,788 density determinations from the footwall of the Bikkuri.

Table 11.1 Density by Stratigraphic Unit

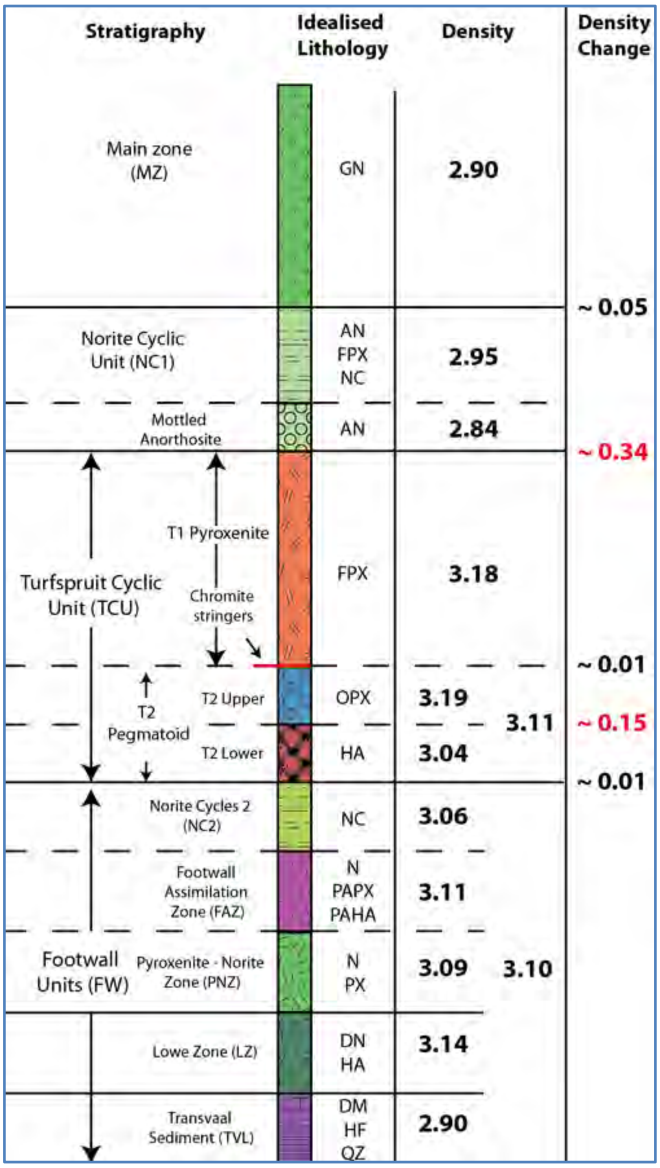
MSTRAT	Description	MSTRAT	MCODE	Number	Average	Minimum	Maximum	C.V.
Bikkuri Hanging Wall	Main Zone	BKHW	10	3,772	2.91	2.55	3.30	0.03
	Bikkuri Norite Cycles 1	BKNC1	11	244	2.98	2.62	3.26	0.04
	Bikkuri Mottled Anorthosite	BKMAN	12	31	2.83	2.63	2.95	0.02
Bikkuri	Bikkuri B1	B1	13	264	3.15	2.62	3.34	0.03
	Bikkuri B2	B2	14	59	3.13	2.84	3.37	0.04
Bikkuri Footwall	Bikkuri Norite Cycles 2	BKNC2	15	47	3.06	2.85	3.27	0.04
	Bikkuri Lower Zone	BKLZ	16	1,741	3.09	2.33	4.35	0.06
TCU Hanging Wall	Main Zone	MZ	20	17,271	2.90	2.44	3.58	0.03
	Norite Cycles 1	NC1	21	988	2.97	2.58	4.35	0.06
	Mottled Anorthosite	MAN	22	147	2.84	2.55	3.02	0.02
TCU	T1	T1	23	2,219	3.19	2.58	3.69	0.03
	T2 Upper	T2U	24	718	3.19	2.57	3.82	0.04
	T2 Lower	T2L	25	725	3.04	2.49	3.37	0.05
TCU Footwall	Norite Cycles 2	NC2	26	280	3.05	2.61	3.31	0.06
	UG2 Hanging Wall	UG2HW	27	38	3.11	2.60	3.43	0.07
	UG2	UG2	28	2	3.49	3.44	3.53	0.01
	UG2 Footwall	UG2FW	29	32	3.18	2.95	3.44	0.03
	Lower Zone 1	LZ1	30	5,993	3.11	2.48	6.82	0.05
	Lower Zone 2	LZ2	31	3,689	3.09	2.45	4.43	0.06

Figure 11.1 shows an idealised strip log with the associated densities, and two horizons of large density contrast are marked A and B. With reference to Figure 11.1, MTS notes that:

- There is a ~0.34 SG density contrast across the MZ/NC1/MAN and the T1 contact.
- Within the T2 the most significant difference is between the OPX or T2U (SG 3.19) and the HA or T2L (SG 3.04). When the T2 units are combined, the overall average SG is 3.11. The HA has a lower density than OPX because the HA is serpentinised.

The difference between the T2 (3.11) and the Footwall units (3.10) is negligible.

Figure 11.1 Idealised Density Strip Log



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11.3 Analytical and Test Laboratories

To date, laboratories utilised for the Platreef Project include the primary laboratories Set Point Laboratories (Set Point; Johannesburg, RSA), Ultra Trace Laboratory (Ultra Trace; Perth, Australia) and Genalysis Laboratory Services (Genalysis; Perth, Australia, and Johannesburg, RSA). The check laboratories were Lakefield (Lakefield Johannesburg; Johannesburg, RSA), Genalysis Laboratory Services (Genalysis; Perth, Australia, and Johannesburg, RSA), Ultra Trace Laboratory (Ultra Trace; Perth, Australia) and Acme Laboratories, (Acme, Vancouver, Canada). Bureau Veritas Minerals Pty Ltd (Bureau Veritas) assumed control of Ultra Trace in June 2007 and is responsible for assay results after that date. In 2011, a set of samples were submitted to ALS Chemex (Vancouver, Canada) to assess laboratory quality. No additional samples have been submitted to ALS Chemex.

Metallurgical laboratories include G&T Metallurgical (G&T Metallurgical; Kamloops, BC, Canada), SGS Metallurgical Services (SGS; Johannesburg, RSA), Xstrata Process Support (XPS; Falconbridge, ON, Canada), and Mintek laboratories in Johannesburg, RSA.

All of these listed laboratories were, and are, independent of Ivanhoe.

Set Point had no accreditations during the time period it performed assays of Platreef samples. Set Point was accredited to ISO17025 in 2003 and 2004. Set Point has participated in Geostats, Australia round-robin assessments since 2000.

Ultra Trace was registered with the Australian National Association of Testing Authorities (NATA number 14492) and was registered for the analysis of nickel-bearing samples by ICP methods and also by XRF. In 2007, Ultra Trace became a subsidiary of Amdel Limited (Amdel; head office: Port Melbourne, Australia). Amdel has adopted the ISO 9001 Quality Management Systems, and is a member of Bureau Veritas, an international group specialising in the inspection, analysis, audit, and certification, and management systems in relation to regulatory or voluntary standards. In June 2013 the entities Amdel, Ultra Trace, and Kal Assay Labs began trading as Bureau Veritas Minerals Pty Ltd. Bureau Veritas Minerals Pty Ltd maintains an ISO9001.2000 quality system as well as NATA ISO 17025 certifications.

Lakefield Johannesburg (now a subsidiary of SGS and renamed SGS Johannesburg) was not accredited before December 2002, but uses the same protocols and procedures as its sister laboratory, Lakefield Research, in Canada. Lakefield Johannesburg was actively working on obtaining ISO accreditation during the time period covered by its assaying of Platreef samples and became accredited to ISO 10725 in December 2002. Lakefield Johannesburg participated in proficiency testing during the time-frame covered by its check assay work on Platreef drilling samples, including the CANMET laboratory evaluation for PGEs and base metals.

Genalysis Perth is an accredited NATA laboratory (NATA number 3244). The terms of accreditation included most analyses performed for Platreef. The laboratory was accredited to AS ISO/IEC 17025–1999 and included the management requirements of ISO 9002:1994. The Perth facility is accredited in the field of Chemical Testing for the tests shown in the Scope of Accreditation issued by NATA. The South African facility holds ISO/IEC 17025:2005 accreditation for specified analytical techniques.

Genalysis also participates in a number of regular international, national and internal proficiency round-robins and client specific proficiency programmes.

G&T Metallurgical has ISO 9001:2000 registration (KPMG certificate number 1613). Their registration certifies provision of consultancy services to the mining industry including metallurgical, mineralogical, and assay testing procedures.

SGS in Johannesburg has ISO 9001 and 14001, OHASA 18001, and SA 8000 accreditation.

XPS is not accredited with ISO for metallurgical testing. They reportedly use a series of internal quality controls that assure 95% confidence in the results. This system was audited by Six Sigma and passed those criteria, although no official certificate was issued. Assaying reported by XPS is done by ALS Chemex which is registered to ISO 9001:2008. ALS Chemex also has accreditation from the Standards Council of Canada (CAN-P-4E, ISO/IEC 17025:2005), and General Requirements for Competence of Testing and Calibration Laboratories, and the Programme for Accreditation of Laboratories in Canada (PALCAN) handbook (CAN-P-1570).

In late 2010, Acme Laboratories (Acme) of Vancouver, Canada, became the check laboratory. The laboratory holds ISO/IEC 17025:2005 accreditation for specified analytical techniques. In the Q3 2011, Ultra Trace could no longer accommodate all of the Platreef Project's greatly increased sample production. Some samples were therefore submitted to Genalysis and Set Point Laboratories, both in Johannesburg, and ALS Chemex in Vancouver. Also in Q3 2011, Genalysis became the check laboratory, with some check samples submitted to Ultra Trace (for cases where Genalysis was the primary assay laboratory).

Mintek is a South African National Accreditation System accredited testing laboratory and holds ISO/IEC 17025:2005 accreditation for specified analytical techniques.

11.4 Sample Preparation and Analysis

Sample preparation for all samples was completed by Set Point. Set Point analysed samples until capacity was reached in 2002. From November 2002 to August 2011, all prepared samples were analysed by Ultra Trace. In addition to Ultra Trace, Set Point provided sample analysis from August through October 2011 as did Genalysis from October 2011 through May 2012. Ultra Trace has been the sole primary analysis laboratory since May 2012.

11.4.1 AMK and ATS Sample Preparation and Analysis

AMK and ATS drilling was completed to support open pit Mineral Resources. Ivanplats is no longer considering the open-pit option. A detailed description of the AMK and ATS sample preparation and analysis is available in the September 2012 Technical Report (Parker et al., 2012). Overall, the preparation and analytical methods used were to industry standards at the time.

11.4.2 UMT Sample Preparation

After sampling, the UMT samples are loaded on a truck and transported to the Set Point Laboratory in Mokopane for sample preparation. The samples are loaded in the presence of a supervisor and QA/QC coordinator. The transportation department records the number of samples, sample numbers and date of delivery in a chain of custody book. The receiving personnel at the laboratory sign the chain of custody.

The Set Point preparation laboratory checks the sample numbers against the sample submission form. Each sample is weighed, and the sample weight is reported to Ivanhoe. Samples are crushed to 10 mm using a Keegor crusher and milled to 1.7 mm using a Labtechnics mill (LM2); the sample mass requires that the sample be divided into two or three portions for this brief milling (approximately 15 seconds). The portions are then blended back together by passing them three times through a riffle splitter. A sample from every 20th sample is tested by screening through a 1.7 mm screen. If the specification is not met (90 % passing 1.7 mm), the sample is re-crushed, and two nearby samples (between the failing sample and the preceding and following tested samples) are randomly selected and tested. If one of these fails, the entire corresponding group of samples is re-crushed, and the crush time of the crusher adjusted.

The samples are split in half using a riffle splitter. One split is packaged and returned to the Platreef office. The second split is milled to 90% passing 106 µm. A split of the pulp sample (±200 g) is repacked for shipment to assay laboratory. All materials are returned to Ivanhoe.

After return to the Platreef Project, the pulps packed for submission are placed in numerical order, standard and certified reference material (SRM and CRM) samples are inserted into the sequence, and pulps are boxed for shipment to selected assay laboratories.

11.4.3 UMT Sample Analysis

Ultra Trace is the main laboratory used to analyse samples and used a multi-acid digestion followed by inductively coupled plasma-optical emission spectroscopy (ICP-OES) reading to determine total Ni, Cu, Cr, and sulfur. Some samples were also assayed for sulfur using a LECO furnace (controlled combustion of sample pulp with infrared reading of SO_2 gas); the LECO and ICP sulfur results show close agreement. Lead flux (collector) fire assays with an ICP-MS finish were used to determine Pt, Pd, and Au. Historically, samples within a 2 g/t 3PE+Au grade shell were selected and analysed for Rh. The current practice requires samples containing greater than 1 g/t Pd to be submitted for Rh analysis. Samples submitted for Rh analysis are assayed by fire assay using lead collection and palladium secondary collection followed by inductively coupled plasma-mass spectrometry (ICP-MS) (FA004). For comparison purposes, approximately every twentieth sample would also be assayed by fire assay with nickel sulfide collection followed by ICP-MS (FN001).

Set Point was used as an additional assay laboratory for portions of 2011. The following assay methods were used (laboratory codes included in parentheses):

- Fire assay lead collection followed by ICP-MS for Au, Pt and Pd (Code 416).
- Total acid digestion followed by ICP-OES for Cu and Ni (Code 255).
- S by Leco (Code 255).

- Fire assay Pd collector followed by ICP-MS for Au, Pt and Rh (Code 415).
- NiS collection for Au, Pt, Pd, Rh, Ir, Ru and Os (Code 419).

Ultra Trace (now Bureau Veritas Minerals) used the following analytical methods:

- Fire assay lead collection followed by ICP-MS for Au, Pt and Pd (Doc 600, now FA003).
- Total acid digestion followed by ICP-OES for Cr, Cu, Ni and S (Doc 214, current code is MA101).
- Selected samples have been analysed for Rh, Pt, Pd and Au using fire assay lead / palladium collection followed by ICP MS (code FA004).
- A subset of these samples have been analysed for Rh by fire assay with nickel sulfide collection (code NSF001).
- Small-scale aqua regia digestion followed by ICP-OES for Cr, Cu, Ni and S (AR201). This method was used for check analysis only and not for primary samples.

Genalysis was used as an additional assay laboratory for portions of 2011 and 2012 and used the following analytical methods:

- Fire assay lead collection followed by ICP MS for Au, Pt and Pd (method code FA25/MS).
- Multi acid digestion followed by ICP-OES for Cu, Cr, Ni, and S (method code 4A/OM).
- Aqua regia digestion followed by ICP/OES for Cu, Cr, Ni, S (method code AR01/OM). This method was used for check analysis only and not for primary samples.

11.4.4 Check Sample Analysis

Genalysis in Perth used the following analytical methods (laboratory codes included in parentheses):

- Fire assay lead collection followed by ICP-MS for Au, Pt and Pd (FA25/MS).
- Multi acid digestion followed by ICP-OES for Cu, Cr, Ni and S (4A/OM).
- Sieve test as indicated by individual sample breakdown (SV02).
- Aqua regia digestion followed by ICP-OES for Cu, Cr, Ni, and S (AR01).

In contrast, the Johannesburg branch of Genalysis used the following methods on selected samples:

- NiS fire assay for Au, Pt, Pd, Rh, Ru, Os and Ir (NS25/MS).
- Pd Collector fire assay for Rh (FA25P/OE).

ACME used the following protocols:

- 3B03 - Lead fire assay followed by ICP MS for Au, Pt and Pd.
- Group 1E – Four-acid digestion followed by ICP OES (for Al, Ca, Cr, Cu, Fe, Mg, Ni and S).
- Group 1D01 - Aqua regia digestion followed by ICP OES (for Al, Ca, Cr, Cu, Fe, Mg, Ni and S).

No check samples have been submitted since October 2012.

11.5 Quality Assurance and Quality Control

11.5.1 AMK and ATS QA/QC

AMK and AST drilling was completed to support Mineral Resources amenable to open pit mining methods. Ivanplats is no longer considering the open-pit option. A detailed description of the AMK and ATS QA/QC is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted.

11.5.2 UMT QA/QC

Control Samples

As is prevalent throughout the industry, all laboratories employed by the Platreef Project use their own quality-control materials (blanks, pulp duplicates, standards) within each laboratory process batch. Laboratories routinely re-ran batches that failed their quality control requirements. Batches, which vary in size, typically include two duplicates, one or two blanks and a laboratory reference material. Results of laboratory quality controls are included in the laboratory reports. These results are informative because they show what the laboratory considers to be acceptable performance; batches showing inadequate performance are re-run, and the original assays are not part of laboratory final reports.

The Platreef Project inserted coarse reject duplicates, field blanks, and packets of certified reference materials (CRMs) in order to independently monitor laboratory performance.

Blanks

Blanks utilised locally sourced natural rock materials that have <10 ppb concentrations of Au, Pt, and Pd, but have copper concentrations of < 35 ppm and Ni concentrations of < 65 ppm. Blanks underwent preparation steps and therefore provide an upper limit on levels of contamination caused by preparation. One blank sample is inserted every 20th sample.

Coarse Reject Duplicates (CRDs)

Coarse reject duplicates were created by the preparation laboratory by routinely making a sample from the coarse reject of every 20th sample and assigning it the same sample number as its duplicate pair, with the addition of a suffix CRD.

Certified Reference Materials (CRMs)

All sample submissions included packets of CRMs inserted every 20th sample. These CRMs were purchased from commercial African Mineral Standards (AMIS, Johannesburg), and/or in-house SRMs were used; the in-house SRMs were made from composites of drill sample coarse rejects that were prepared by SGS (Johannesburg), with best values assigned based upon round-robin results. Details are provided in Acuity (2015), Reid (2011, 2014) and Long (2013a). In-house SRMs were phased out as appropriate materials became available from AMIS.

As many as 15 CRMs and SRMs have been used extensively enough to compare Ultra Trace's mean results of each for comparison to best values. Currently, nine CRMs are in use. Excluding outliers that triggered follow-up investigation (for control insertion mix-ups) and in very rare cases remedial re-assaying of some laboratory batches, the average of the Ultra Trace results is within 10% of the certified value for the major elements of interest (Ni, Pt, Pd, Au, Cu) and in most cases for the added element, sulfur. Ultra Trace results for Cr are much lower than the AMIS certified values (based upon fusion or XRF pellet analysis), indicating that the multi-acid digestion method is not adequate for this element. This is a known problem with acid digestion for Cr.

Check Assays

Approximately 5% of drill sample pulps previously assayed by Ultra Trace were forwarded, along with blind CRMs and blanks, to Genalysis, Perth. Genalysis performed the same assay suite, plus aqua regia digestions for Ni and Cu. Agreement was usually adequate and, in all cases where it was not, samples were re-assayed by both laboratories to resolve the problems. The assay database was routinely updated where remedial assaying was performed.

In 2010, Genalysis began to exhibit some systematic errors in its acid digestion assays, likely attributable to introduction of new heating blocks. The problem was eventually resolved, but the decision was taken to suspend sending check assays to Genalysis. Sample pulps were instead submitted to Acme Laboratories, Vancouver.

Prior to suspending submissions to Genalysis, the Platreef Project used Genalysis aqua regia results to estimate, for each rock type, the fraction of total Ni likely to be in sulfide minerals that could potentially be recovered by the flotation process. However, inserted controls showed increased batch-to-batch variations in aqua regia results, and Genalysis stated that their results should be considered semi-quantitative for this method.

Ivanhoe selected some mineralised samples to undergo an additional nickel sulfide collector fire assay to validate the conventional lead collector fire assay results for Pt and Pd, and to determine the grade of other PGEs, particularly Rh. NiS fire assays return lower Au results and are not regarded as reliable for Au. Pt and Pd results were on average slightly higher (about 5%) compared to the lead collector fire assays.

No check assays have been completed since October 2012.

11.6 Databases

The drillhole data were maintained in a Fusion database, created by Century Systems Technologies Inc. The Fusion database is maintained at the Platreef Project site. All available drillhole data including data from the AMT and ATS drill campaigns have been captured in the database.

The drillhole database was migrated from the Fusion database to an acQuire database on 1 May 2013.

11.6.1 AMT and ATS Data Entry

A description of the AMK and ATS data entry is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted.

11.7 Sample Security

Pulp rejects and coarse rejects were returned to the Ivanhoe offices in Mokopane, where they were stored in warehouses. Access to the warehouses is restricted to Ivanhoe employees with the appropriate security clearance. The compound containing the offices and warehouses is guarded on a 24-hour basis. Pulps sent to Ultra Trace are stored at Ultra Trace, with the exception of those pulps selected for check assays, which were in most cases exhausted after conducting checks.

11.8 Comments on Section 11

The sample preparation, sample analyses, data entry and security have been done to industry-standards for large exploration and development projects. Ivanhoe personnel involved in these activities have been well-trained to maintain the integrity of samples and their analyses. Mr Kuhl is of the opinion that the quality of the Pt, Pd, Au, Rh, Cu, and Ni analytical data are sufficiently reliable (also see discussion in Section 0) to support Mineral Resource estimation as follows:

- Data are collected following industry-standard sampling protocols.
- Sample collection and handling of core were undertaken in accordance with industry-standard practices, with procedures to limit potential sample losses and sampling biases.
- Sample intervals in core are 1 m intervals within lithological boundaries in the UMT area; the sample intervals are considered to be adequately representative of the mineralisation.
- Bulk density determination procedures are consistent with industry-standard procedures, and there are sufficient bulk density determinations to support tonnage estimates.
- Sample preparation for samples that support Mineral Resource estimation has followed similar procedures since 2001. The preparation procedure is in line with industry-standard methods for Pt-Pd-Au-Rh–Cu–Ni deposits.

- Core drill programmes were analysed by independent laboratories using industry-standard methods.
- Typically, Platreef drill programmes included insertion of blank, duplicate and SRM or CRM samples.
- Data that were collected were subject to validation, using in-built programme triggers that automatically checked data on upload to the database.
- Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar coordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards.
- Sample security has relied upon the fact that the samples were always attended or locked in the onsite sample preparation facility.
- Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.
- Current sample storage procedures and storage areas are consistent with industry standards.

12 DATA VERIFICATION

Several reviews of the database have been made since 2002. These include Wood reviews conducted when the Qualified Person was still employed at Wood and those performed by independent consultants. Database audits were performed by Wood in 2007, 2010, 2012, 2014 and 2015 to ensure its suitability for resource estimation.

12.1 Wood Site Visits

12.1.1 Site Visits by QPs During UMT Drilling

In the April 2010 site visit (Kuhl, 2010), Mr Kuhl completed a database audit and performed field checks of drill collars. No significant errors were noted that could affect Mineral Resource estimation.

Mr Kuhl also visited site in July–August 2011, and observed drilling operations and reviewed geology logging.

Mr. Kuhl completed a site visit between 25 January and 2 February 2012 and reviewed the TCU geological interpretation in cross-sections and drill core. Mr. Kuhl also visited drilling locations.

Mr. Kuhl visited the Platreef Project between 25 November and 12 December 2012. Mr. Kuhl reviewed the geological interpretation for the TCU in cross-sections and drill core. Mr. Kuhl completed preliminary exploratory data analysis and initiated work constructing the geological model. Mr. Kuhl supervised the packaging and shipment of 20 witness samples from Set Point (Mokopane) to the Ultra Trace Laboratory.

Mr. Kuhl visited the Platreef Project between 13 May and 23 June 2015 and between 9 July and 3 August 2015. During these site visits, the structural and geological interpretations were reviewed in both cross-section and drill core. Mr Kuhl also initiated exploratory data analysis and collected 20 witness samples from recent drillholes, observed the sample preparation at Set Point (Mokopane) and supervised the packaging and shipment to the Ultra Trace Laboratory.

12.1.2 Other Site Visits

Mr Scott Long visited the site under the supervision of Dr Harry Parker (both from Wood) on a number of occasions between 2001 and 2013, most recently between 26 February and 2 March 2013. During these visits, Mr Long created and maintained the QA/QC programme for sampling and assaying, trained Ivanhoe QA/QC specialists and periodically reviewed their work, upgraded and expanded the QA/QC programme where warranted, including addition of new assay laboratories, and assisted with resolution of problems identified by the QA/QC programmes.

Dr Harry Parker visited the site on several occasions between 2011 and 2015. During these visits, Dr. Parker reviewed geology logging, inspected core, verified collar coordinates, collect witness samples, and observed data collection programs.

12.2 ATS and AMK Database Reviews

A description of the AMK and ATS database reviews is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted during the Fusion database reviews.

Given the problems identified while migrating the database from Fusion to acQuire for the UMT data (see Section 12.4), MTS recommends a complete review of the assay database for the ATS and AMK drilling be completed against assay certificates prior to using ATS and AMK assay data for Mineral Resource estimations.

12.2.1 UMT Database

The data acquisition procedure includes filing of hard copies of drillhole data after the data have been captured in the SQL Fusion database (coordinate surveys, total depth, down hole surveys, updated drillhole logs and assay certificates). An additional database administrator and additional database entry clerks were employed and trained to assist with the increased amount of data from the drill programmes. The Fusion 6.6 SQL logs authorised changes to data, thereby creating an audit trail. The changes are date and time-stamped and include the name of the person who made the changes. From 1 May 2013, all data are captured into the acQuire database, with the same hard copy system in place.

12.3 UMT Database Reviews

Multiple reviews of the database were conducted by Mr Kuhl while still employed at Wood.

12.3.1 March 2010 Review

Wood completed a database review in April 2010. The review included collar and survey checks for 53 UMT drillholes completed after the 2007 database review of open pit data. All collars and surveys were checked against supporting documents. Lithology and density data were compared to supporting documents for five of the additional 53 drillholes. Assay data were checked for 5% of the assays from the additional 53 drillholes. No issues that could affect Mineral Resource estimation were noted.

12.3.2 August 2012 Review

Wood completed a database review in August 2012 for drillholes completed after April 2010 (Yennamani, 2012). The review compared the collar survey, down-hole survey, geology logs and assay data (Au, Pt, Pd, Ni, and Cu) against supporting documents. Wood concluded the database was acceptable to support Mineral Resource estimation.

12.3.3 December 2012 Review

A database review was completed in December 2012 for drillholes completed after August 2012 (Yennamani, 2013). The review compared the collar and downhole surveys, geology logs and assay data (Au, Pt, Pd, Ni, and Cu) against supporting documents. Minor errors were identified and corrected by Ivanhoe staff. The assay database was considered acceptable to support future Mineral Resource estimation.

12.3.4 February 2014 Data Review

A database review was completed in February 2014 for drillholes completed after December 2012. The review compared collar surveys, downhole survey and geology logs against supporting documentation. Wood verified >95% of the assay results against original laboratory reports. The Qualified Person concluded that the drillhole database was acceptable to support Mineral Resource estimation.

12.3.5 October 2014 Data Review

The Qualified Person completed a database review in October 2014 for drillholes completed after February 2014. The review compared collar surveys, downhole surveys and geology logs against supporting documents. Checks included 100% of assays for Pt, Pd, Au, Ni, Cu, Cr, and S. Available Rh assays were also checked. Wood concluded the database is acceptable for Mineral Resource estimation. A programme of assaying selected samples for Rh was proposed and initiated.

12.3.6 May 2015 Data Review

The Qualified Person completed a database review in May 2015 for drillholes completed after October 2014. The review compared collar surveys, downhole surveys and geology logs against supporting documents. Checks included 99% of assays for Pt, Pd, Au, RH, Ni, Cu, Cr, and S. Wood concluded the database was acceptable for Mineral Resource estimation.

12.4 Database Migration

Although data capture into the acQuire database was initiated on 1 May 2013, the final database migration from Fusion to acQuire was not completed until Q1 2014. The Qualified Person compared the previously audited data from Fusion database to the acQuire database. Data checks included collar survey table, deviation survey table, geology tables and specific gravity table. Errors were identified and corrected (King, 2015 and Reid, 2016b). Errors identified in the assay table review resulted in the Qualified Person checking approximately 100% of the assay data for the UMT drillholes against laboratory certificates. A review of the assays from the AMK and AMT drillholes is pending.

12.5 Quality Assurance and Quality Control Results

Ivanhoe monitors QA/QC data (blanks, duplicates and CRMs) when results are received. If results are not within established limits, re-analysis of samples in the vicinity of the failing controls are requested. The data are not accepted unless re-assays produce acceptable results. Overall, a small number of reports have been rejected, and these have been remediated. The Qualified Person periodically reviewed QA/QC data.

Multiple QAQC data reviews were conducted when the Qualified Person was still employed at Wood.

12.5.1 UMT QA/QC (To March 2011)

Wood obtained and reviewed the available QA/QC data for the UMT drilling in March 2011. It was noted:

- All Ultra Trace means on SRMs are within 5% of recommended values for the five major elements of economic interest (Pt, Pd, Au, Cu, Ni). Results are sufficiently accurate for Mineral Resource estimation for all five elements of economic interest.
- Generally, the results for Au, Pt, and Pd blanks were satisfactory. Significantly poorer performance was noted for Cu and even more so for Ni results. The apparent poor performance for Cu may be a consequence of a low bias in Set Point Cu assays (used to certify the blank material). Nickel values were of concern because approximately 80% of samples exceeded the 8 ppm value stated by Set Point.
- Genalysis results for Cu, Pt, Pd, and Au were in line with the SRMs, but Genalysis showed a low bias for Ni. Wood recommended that all samples with Ni results greater than 10,000 ppm undergo an additional check assay by XRF fusion, which is likely to be more reliable in this grade range.

In mid-2010, approximately 5% of pulps were selected from pulps stored at Ultra Trace. The submission included certified reference materials. Data review indicated that:

- Acme results were approximately 10% higher for PGE fire assays compared to Ultra Trace results. Inserted CRMs in both Ultra Trace and Acme submissions indicated this can be accounted for by a slight low bias in the Ultra Trace results and a slight high bias in the Acme PGE results. The Ultra Trace results likely slightly underestimate PGEs by approximately 5% and therefore have very low risk of being biased high.
- Acme produced mean sulfur grades that are 20% higher than Ivanhoe's average by one method it used, and 20% lower than Ivanhoe's average by the other. Taken together, these two methods average to agree with Ivanhoe's average result.

Wood evaluated the duplicate samples were evaluated by calculating the absolute value relative difference (AVRD), equal to the absolute value of the pair difference divided by the pair mean. Evaluating the AVRD of the coarse-reject duplicates indicated that AVRD for Au, Pd, Cu, and Ni met the 90th percentile goal of 20%. Pt exceeded the threshold, with AVRD values of 28% at the 90th percentile.

Wood noted that Ivanhoe were not submitting pulp duplicates as part of their QA/QC programme, and recommended that Ivanhoe use Ultra Trace's reported pulp-duplicate results to assess the precision of pulp duplicates.

12.5.2 UMT QA/QC March 2011–June 2012

Wood obtained and reviewed the available QA/QC for the period between March 2011 to June 2012. Results were as follows:

- Approximately 3,100 blanks were passed through preparation and assay during the period. Three clusters of low-grade contamination were found in three different drillholes (UMT 146, 155, and 181), all assayed by Genalysis. Indications were that the contamination likely occurred during sample preparation. The level of contamination is too low to have any impact on the future use of the samples in Mineral Resource estimation.
- The Platreef Project's increased drilling rate necessitated using Genalysis and Set Point laboratories, in addition to Ultra Trace. Wood separated the results by laboratory and calculated each laboratory's median result for each element of interest for each AMIS CRM. Results showed acceptable agreement between the laboratories.
- Multi-acid digestion results show good accuracy by all laboratories for copper and nickel but pronounced low biases by Genalysis and Ultra Trace for Cr. Set Point does not report Cr results. The Cr assays are not accurate by multi-acid digestion. Reliable Cr results most likely would require a fusion followed by reading by XRF. The low bias seen here is consistent with that seen previously in Ultra Trace results.

Except for Cr, which is not used in the resource estimations, accuracy of these elements is sufficient by all laboratories for use in estimation of Mineral Resources.

In 2016, Wood (Reid, 2016a) reviewed the results of check samples submitted in 2011 and 2012.

Between 28 June 2011 and 25 October 2012, Ivanhoe submitted 20 batches of check samples (comprised of a 5% selection of available pulp material) to ACME Laboratories (Acme) in Vancouver, Canada. Although a review of the included CRMs showed poor performance with respect to the CRMs by ACME, the check assay results were generally within 5% of the primary assay laboratory results. Only Pd showed a slight bias outside of ± 5%; Pd results indicated the primary laboratory was 5.5% lower than ACME check results.

12.5.3 UMT QA/QC June 2012–July 2014

Wood obtained and reviewed the available QA/QC for the period between June 2012 to July 2014. Results were as follows:

- Approximately 899 blanks were passed through preparation and assay during the period. The average results for Pt, Pd and Au were less than 5 ppb, and the average results for Ni and Cu were less than 20 ppm. This is comparable to previous results. The level of contamination observed is too low to have any impact on the use of the samples in Mineral Resource estimation.

- Eight CRMs were submitted for analysis. The overall relative bias for the CRMs is within 5%, and the assay accuracy is sufficient for resource estimation.
- Duplicate results from coarse reject material indicate acceptable precision is obtained by Ultra Trace.

Ivanhoe did not submit any samples from this period for check assays.

Based on the above results, the Qualified Person is of the opinion the Pt, Pd, Au, Ni, and Cu assay results for this period have sufficient accuracy and precision to support resource estimation.

In October 2012 Ivanhoe submitted three batches to Genalysis Laboratory Services Pty Ltd (Genalysis) in Perth, Australia and one batch to Ultra Trace Assay Labs (Ultra Trace) in Perth, Australia. A review of the included CRMs showed poor performance with respect to the CRMs by Genalysis while there were too few results from Ultra Trace to express an opinion on assay accuracy.

The check assay results from Genalysis were within 5% of the primary assay laboratory results with the exception of Au, which showed a 10.9% positive bias. This indicates the primary assay laboratory results are higher than the check assay (Genalysis) results.

The check results from Ultra Trace were within 5% of the primary assay laboratory results with the exception of Au, which showed a 10.9% negative bias. This indicates the primary assay laboratory results are lower than the check assay (Ultra Trace) results.

Due to the low-grade nature of the Au check samples, these biases are not considered to be material.

Ivanhoe has not submitted additional samples for check assay since October 2012.

12.5.4 UMT QA/QC July 2012–July 2015 (Acuity Geoscience)

Mr. Dale Sketchley of Acuity Geoscience Ltd. (Acuity) completed a review on the QA/QC data available for drilling completed between July 2012 and July 2015 (Acuity, 2015). Results include:

- Results for Au, Pt and Pd from 920 blank samples indicated only two samples above the 40 ppb threshold for Pt and Pd. Copper and Ni results from 846 samples indicated two samples with Cu results and three samples with Ni above the 100 ppm threshold. Approximately 899 blanks were passed through preparation and assay during the period. The average results for Pt, Pd and Au were less than 5 ppb, and the average results for Ni and Cu were less than 20 ppm. This is comparable to previous results. The level of contamination observed is too low to have any impact on the use of the samples in Mineral Resource estimation.
- Acuity's review of results from the nine CRMs submitted for analysis determined that the overall relative bias for the CRMs is within 5% and concluded the assay (with exception of Au results below 75 ppb) accuracy is sufficient for resource estimation.

- Duplicate results from coarse reject material indicated acceptable precision with the exception of gold as obtained by Ultra Trace. Acceptable precision was defined as having an absolute relative difference at the 90th percentile of 20%. Gold was observed to have a difference of 31-37%. Based on limited tests, Acuity has recommended finer grinding (85-90% passing 75 µm).

Check assay data were not reviewed in Acuity's memorandum.

Based on the above results, Acuity was of the opinion the Pt, Pd, Au (with the exception of values less than 75 ppb), Ni, and Cu assay results for this period have sufficient accuracy and precision to support resource estimation. Initially Acuity stated Au results below 50 ppb should be excluded from resource estimation; however, subsequent discussions between Wood and Acuity indicated that Au results below 75 ppb should be reduced due to high bias. The recommended adjustment would be to reduce Au results below 50 ppb by 2% and to reduce Au results between 50 and 75 ppb by 1%. Wood reviewed the impact of the Au grade reduction (Wood, 2016) and found the impact to be negligible. The recommended correction was not made.

12.6 Witness Samples

Four groups of witness samples have been collected at Platreef by Wood, in April 2010, February 2011, November 2012 and May 2015. The purpose of collecting these samples was to confirm the presence of mineralisation.

12.6.1 2010 Witness Samples

Wood collected 20 witness samples in 2010 by selecting individual sample intervals of varying Ni grade. The selected sample intervals were re-sawn, and quarter core samples were prepared and submitted to SGS Lakefield. There were some large differences, particularly for Pt, but differences in mean grade were not statistically significant. Follow-up evaluation involving re-assaying of original and new quarter core coarse rejects and pulps by both SGS and Ultra Trace laboratories revealed that the differences stemmed from differences in the grades of the original (half core) and witness (quarter core) samples. Long and Parker (2011) concluded a larger number of samples were required in order to achieve a reliable verification of the original assays or if large differences were found, showing them to be statistically significant.

12.6.2 2011 Witness Samples

Long (2011a) collected a second group of 260 witness samples.

Quarter-core samples were prepared in the same way as routine samples. All samples were submitted to Ultra Trace for the current standard suite of analysis: Au, Pt, and Pd by lead fire assay (sample weights approximately 40 g) with ICP/MS finish (2 ppb detection limit); Cu, Ni, and Cr by multi-acid digestion followed by ICP/OES (1 ppm detection limit); and S by Leco furnace (50 ppm detection limit).

Very close agreement was obtained between original and quarter-core samples for Cu, Ni, and S, and adequate agreement was obtained for Au. There was no preferential sampling of sulfides in the original (half core) samples.

Pt and Pd returned lower average results in the quarter-core sampling compared to the original sampling. However, the results of the inserted CRMs indicated that the Pt and Pd results had a low (but within the acceptable range) bias for Pt and Pd, or around 5%, and the CRMs associated with the original results for these samples did not.

After applying a correction to the Pt results for the low bias shown by CRM results, the difference between the original and new results was no longer statistically significant. However, the correction applied for a low bias shown by CRMs for Pd is smaller, and the data have less variance; consequently, the difference between the original and re-assay results remains statistically significant after applying a correction.

12.6.3 2012 Witness Samples

A third set of witness samples were taken in November 2012, and assay results were received in January 2013. Original and witness assay values were compared for Pt, Pd, Au, Ni, Cu, Cr, and S. Comparison of means of witness samples to means of original results show agreement within 5% for base metals, sulfur, and Pd. The original Au mean is 19% lower than the witness sample mean, and the original Pt mean is 14% higher than the witness sample mean.

Further investigation of Au and Pt results showed the percentage of occurrences where the original result of a pair was less than the witness sample result was not statistically significantly different from the expected 50–50 distribution expected. In the case of Pt, nine out of 20 pairs had a lower Pt result for the original assay.

12.6.4 2015 Witness Samples

Wood obtained a fourth set of 20 witness samples from recent drillholes in May 2015, and assay results were received in June 2015. The samples were collected from quarter-split core contained in intervals within the 1+2+3g/t 3PE+Au grade shells in the TCU stratigraphy. Original and witness assay values were compared for Pt, Pd, Au, Rh, Ni, Cu, Cr, and S and graphed. The resulting charts do not suggest any obvious sample mix-ups or outliers that are not a consequence of variation in grade. Wood concluded no bias is present between the assay values in the database and the values obtained from the witness samples.

12.7 Verification of Grind–Assay Function

Wood selected 92 pulp samples of pyroxenite and harzburgite for screening at 75 µm, because metallurgical test data available in 2011 indicated that there may be enhanced 2PE+Au grades related to the grinding of pulps, particularly for harzburgite. XPS recommended a grind of 80% passing -75 µm. Long (2011b) concluded that over 90% of harzburgite sample pulps are likely to achieve the recommended grind quality. Hence no modification of the grind protocol was recommended, nor was remedial work or further investigation considered warranted.

12.8 Gold Variability Testwork

Acuity (Acuity, 2015a) conducted a number of tests to investigate the high variability of gold lead collector fire assays and to recommend improvements for sample preparation and assaying work. QA/QC monitoring work has shown that gold CRM and duplicate assays typically show high variability for all grade ranges, whereas companion platinum and palladium assays demonstrate high variability above approximately 1,000 ppb. The testwork comprised varying the sample preparation grind size and litharge content of lead collector fire assay flux. Several important observations were noted, which have a bearing on the quality of data available for resource estimation.

- The high variability of gold appears to be at least partly related to pulverising grain size and flux composition. Testwork showed that the 106 µ pulverising size is not optimal, resulting in lower gold values with higher variability, whereas the 75 µ pulverising size is closer to optimal, resulting in higher values with lower variability. The grade increase is noticeable at higher gold values, and the variability increase is noticeable at lower gold values. Moreover, the variability of gold generally decreased with increasing flux litharge content.
- Platinum and palladium grade patterns are not as well developed as for gold, but where there are grade differences of more than several percent, the finer grind size samples returned higher grades. Variability is lower for all of the finer grind size samples compared to the coarser grind size samples.
- Previous metallurgical testwork at Platreef reported similar results, referred to as grind-assay functions, and referenced research with the same findings on other projects in the Bushveld. The cause of the high variability may be that host silicate minerals have much higher melting temperatures and higher viscosity slags, which hinders the collection of precious metals into lead buttons.
- Additional testwork is required to investigate how well-developed are the observed grade and variability relationships at Platreef as there could be an impact on estimated resources. This work should include checking of the pulverising grain size to grade relationships for different ore types and styles of mineralisation, and flux tests aimed at increasing the fluidity of slag and reducing shotting of lead to improve recovery of precious metals. It would be beneficial to construct a 3D spatial variability model of duplicate data together with geology to assist in understanding trends and selecting additional samples for testwork. Laser ablation testwork would provide additional information on mineralogy and grain size relationships. Additionally, sample preparation protocols need to be revised to reduce the routine grinding size to 75 µm from 106 µm.

Although note (Section 12.7) that Long (2011b) thought that harzburgite samples were already being ground to 75 µm.

12.9 Rhodium Analysis Using a Palladium Spike

A separate evaluation of the Pd-spike method for Rh analysis was performed on a subset of 22 samples (plus three duplicate samples). This comparison showed that the addition of the Pd to the conventional fire assay did not affect the Au and Pt results, with means agreeing within 3%. A comparison of a much smaller subset where there were original fire assays by NiS fusion covered five samples plus two duplicate samples. The mean of the Pd spike method was about 4% lower than the NiS fusion result.

The number of pairs is too few for a meaningful statistical test, but the agreement in means suggests this method is likely working sufficiently well for estimating Rh content in Platreef samples. Additional data from sample pulps assayed by both methods are needed to further substantiate this interpretation.

Platreef routinely checks 5% of samples with elevated PGEs using NiS fusion fire assay. These data are consistent with the initial finding that Rh by Pd spike produces a slightly lower (3 to 5%) value than that obtained by NiS fire assay. Pt and Pd are also slightly higher by a similar amount by NiS fusion compared with Pb fusion fire assay. Gold, however, is slightly lower by the NiS fusion method.

As a result of the reviews of the check data, Mr Kuhl concluded that the check data validate the original Rh assays.

12.10 Comparison of Ultra Trace and Mintek Assays

In 2013, Wood conducted a number of comparisons of Ultra Trace (Perth) assays to Mintek (Johannesburg) assays on pulp samples. This was designed to produce assurance that the Mintek head assays, on which metallurgical recovery equations depend, are consistent with the Ultra Trace assays which are the basis for the Mineral Resource estimates. The evaluation commenced with a January 2013 submission of stored pulp splits of exploration drill samples corresponding to drill sample intervals that were used to make up a 2012 bulk sample for metallurgical testwork at Mintek.

Mintek's assays included fire assay results for Au, Pt, Pd, and some Rh (only on samples with elevated PGEs); Leco total sulfur; and two sets of ICP (optical emission) determinations for base metals using an aqua regia digestion and a more robust fusion (followed by acid dissolution of the fused pellet) method. These Mintek assay methods were the same as those used for Mintek's metallurgical testwork.

The principal finding from this initial submission was a high bias of 10 to 15% in Mintek's Ni results; this was shown both by the blind insertions of AMIS standards and by comparison with the original Ultra Trace results on a split of the same pulp. In March 2013, Wood informed Mintek of their poor Ni accuracy and requested re-assay using an atomic absorption instrument. Wood also requested assay proficiency information from Mintek which included a Geostats (Perth) October 2012 proficiency report showing a high bias on Mintek Ni results of similar magnitude. In March 2013, Wood also made a new submission of blind Platreef SRMs (former in-house standards) together with AMIS CRMs with much greater variety and number than what was included in Wood's prior submission to Mintek.

At the same time, Mintek also elected to re-assay the first submission of samples by ICP. All the subsequent results showed acceptable Ni accuracy; Mintek explained that their ICP calibration for Ni had been incorrect. Wood investigation of Mintek internal quality controls revealed that they were relying upon two SRMs that had not been assayed by any other laboratory.

Later in March 2013, Platreef QC Manager Annelien Parsons obtained all available pulp rejects from Mintek testwork, together with Mintek's assay results for those materials. These samples included various kinds of tails and concentrate samples. A few samples of Mintek's SRMs were obtained as well. These samples were submitted to Ultra Trace for base metal analysis in April 2013. The Ultra Trace assay report in May 2013 confirmed acceptable accuracy on all elements except Ni, which showed Mintek metallurgical assay results have high bias exceeding 10%. A regression equation for adjustment of Mintek Ni assays was recommended by Long (2013b):

$$\text{Adjusted Ni} = 0.87*\text{Mintek ICP Ni} + 207\text{ ppm.}$$

This equation shows essentially no adjustment is required for low nickel values, such as around 1,000 ppm, which is the tails assay for nickel; however, the head assays for metallurgical samples (>2,000 ppm Ni) will be affected.

Mintek's stated best values on its two in-house SRMs were also found to overestimate Ni by a similar amount, indicating a long-standing high bias in Mintek Ni results that covers all the Mintek metallurgical testwork performed on Platreef samples in Q4'12 (Long, 2013b).

12.11 Comments on Section 12

Wood has been involved in the Platreef Project from 2001 to 2020, when continuous monitoring of data collection and data entry were conducted. Through his previous employment at Wood, and now at MTS, Mr Kuhl has been involved in the Platreef Project since 2010 Minor problems have been identified and resolved by improving procedures at the site. In the opinion of Mr Kuhl and the QA/QC and database specialist Mr. Reid, sufficient verification has been conducted to provide support that the data collected are suitable for use as a basis for Mineral Resource estimation.

The Qualified Person completed an audit of the UMT drillholes migrated to the Fusion database to the acQuire database. Data checks included collar survey table, deviation survey table, geology tables, assay tables and specific gravity table. Errors were identified and corrected (King, 2015 and Reid, 2016b). Checks of the migrated assay table for the ATS and AMK drilling are still required.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

Various metallurgical testwork campaigns have been conducted since October 2001 on a number of drill core samples originating from the Platreef deposit. Metallurgical testwork has been focused on providing data for flow sheet development whilst aiming to maximise the recovery of platinum group elements (PGEs) and base metals, mainly nickel, and producing an acceptably high-grade concentrate suitable for further processing and/or sale or toll treatment by a third party.

Prior to 2006, testing was predominantly conducted on lower-grade PGE material from the potentially large open-pit area. In 2008, a deep drilling exploratory programme was launched, and the resource was updated to include deeper higher-grade PGE material.

Between 2010 and November 2014, a series of metallurgical testwork campaigns were carried out on the Platreef mineralised material. This, named Phase 1 to Phase 6, testing included comminution characterisation, bench scale flotation testing and laboratory scale dewatering testwork. The findings from this, Phase 1–6 testing, were presented in the Platreef 2015 PFS.

As part of the Platreef 2017 FS, comminution variability testwork was conducted on approximately 1,286 kg of HQ drill core representing the geometallurgical units T1, T2U, T2L and the Contaminated Zone (footwall). Flotation testing was conducted on approximately 1,140 kg of quarter PQ drill core samples representing the geometallurgical units T1, T2U, T2L as well as the Contaminated Zone (footwall). Testing was conducted at the Mintek laboratories in Johannesburg, South Africa. The Platreef 2017 FS testwork programme included, comminution variability testwork, mineralogical characterisation, open circuit flotation development and optimisation testwork, open circuit flotation variability testwork, bench scale locked cycle testwork (LCT) and tailings dewatering testwork.

Further test work was conducted in 2020 and 2021 as part of the current Platreef 2022 FS. The aim of the Platreef 2022 FS test work campaign was to evaluate the potential for inclusion of an HPGR circuit, to further characterise the flotation response of Platreef composite samples, evaluate the potential for Jameson cell technology in the cleaner circuit, determine the effect on flotation response when using site water and conduct preliminary pilot scale test work to produce bulk concentrate samples for settling, filtration and Kell hydrometallurgical refining test work. The Platreef 2022 FS test work was conducted on drill core sample intervals that reflect un-crushed drill core remainders from the 2017 FS variability test work drilling campaign as well as two bulk samples comprised of crushed material from the current surface stockpiles.

The selection of samples, done in conjunction with the mining and geological teams, submitted for the metallurgical testwork for the purposes of the Platreef 2017 FS, and used in Platreef 2022 FS, are deemed to be sufficient.

The Platreef 2015 PFS flow sheet was based on a single-stage milling circuit followed by flotation (MF1) using an oxalic acid and thiourea reagent suite with the inclusion of a post mill conditioning stage. At the time of publishing the Platreef 2015 PFS, MF1 testing of an alternate reagent suite containing a targeted copper collector indicated potential for a simpler flow sheet, using a more conventional reagent suite. Further development of this MF1 copper collector flow sheet was conducted as part of the Platreef 2017 FS. Testing confirmed that this simpler flow sheet was able to achieve a similar metallurgical response compared to the MF1 circuit using oxalic acid and thiourea. As per the Platreef 2015 PFS outcomes, flotation testwork confirmed an optimal target grind, for this flow sheet, to be 80% passing 75 µm. It was decided to base the remainder of the Platreef 2017 FS testing on the MF1 flow sheet, using the targeted copper collector reagent suite.

Comminution variability testwork conducted during the 2017 FS confirmed the previous testwork findings, indicating that the plant feed can be characterised as being hard to very hard and thus not suitable for Semi-Autogenous Grinding (SAG) milling. The comminution variability testwork indicated no significant difference in competency between the ore from the early production years (Year 1 to Year 5) and the later production years. Minor variation in hardness was noted for the T1 and T2U domain samples as compared to the T2L and CZ domain samples.

For the 90% confidence interval, Bond ball work indices were in the range 19.0–24.2 kWh/t. The Bond ball work index data has indicated an approximate 0.5 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1 to Year 5 samples. The abrasion index indicated that the ore can be classified as having a medium abrasion tendency.

The 85th percentile comminution test data was used as the design basis for the crusher and grinding circuit. The crusher circuit was sized based on the 85th percentile crusher work index results. The grinding circuit was sizing was based using the 85th percentile Bond work index data, which were used in combination with particle breakage rates derived from grindmill testwork. The average ore hardness and abrasion index data has been used to derive the operating cost estimate.

The Platreef 2022 FS testwork programme included multiple single pass crushing tests by Thyssenkrupp using a semi-pilot scale HPGR (SMALLWAL). The test work was conducted on a bulk shaft intercept composite sample sourced from the current surface stockpiles.

The testwork confirmed that Platreef samples are amenable to HPGR technology, achieving a specified throughput of 310 to 338 ts/m^{3}h with a specific energy ranging 0.94 to 1.47 kWh/t.

A three stage crushing (primary crushing within mining scope) and ball mill circuit was identified as the preferred option, with lowest associated technical risk during the 2017 FS. A high level HPGR assessment which considered differential capital and operating costs, has indicated that an HPGR circuit offered no/limited benefit at the lower throughput rate for the 0.77 Mtpa Phase 1 concentrator plant. However, for the larger 4.4 Mtpa Phase 2 concentrator, this option offers the potential for an approximate 7% operating cost reduction for the crushing and milling circuits. This option will be considered in more detail during the phased implementation programme.

Bench scale, batch open circuit flotation testwork was performed during the 2015 PFS and 2017 FS to derive the optimal flow sheet. This development work, indicated that a MF1 flow sheet, using the targeted copper collector reagent suite and a split cleaner flotation circuit configuration was optimal. The split cleaner circuit allows for the fast-floating fraction to be treated in a separate cleaner to the medium and slow floating fractions resulting in optimal 3PE+Au, Cu and Ni recoveries for the targeted concentrate grades.

Once the optimum flow sheet had been derived, bench scale, batch open circuit flotation variability testwork was performed.

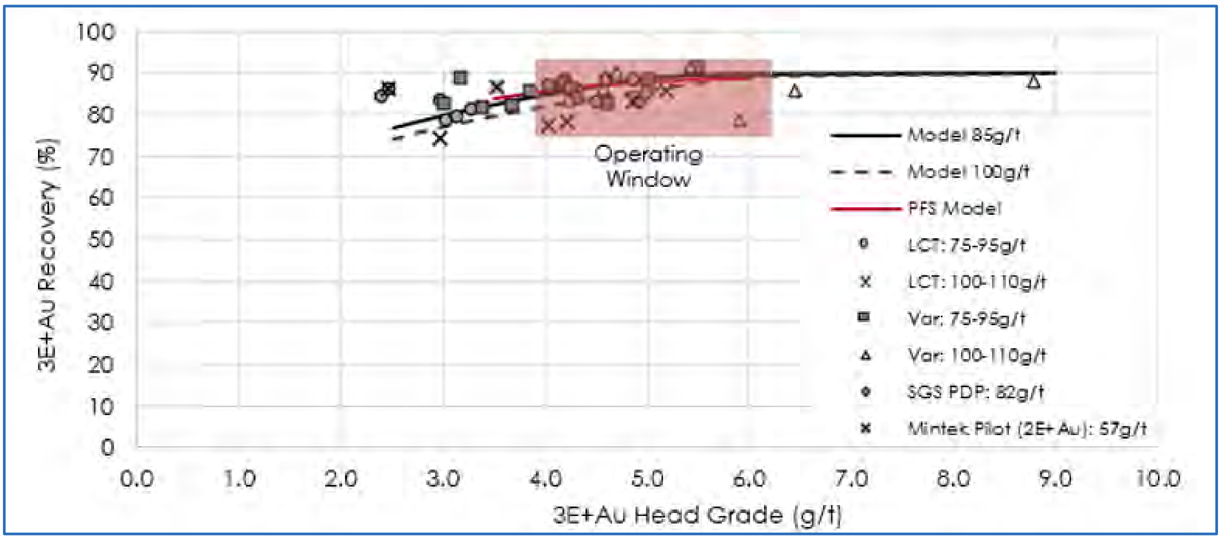
In addition to the open circuit variability testwork, locked cycle testwork was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore schedule, with the focus placed on testing of an ore blend representing the first five years of mining. This locked cycle flotation testing achieved recoveries (3PE+Au) in the range 83.1–88.7% at a saleable final concentrate of approximately 60–95g/t (3PE+Au). Locked cycle testing on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries (3PE+Au) in the range 77.3–85.5% with a final concentrate ranging approximately 80–120 g/t (3PE+Au).

A further locked cycle test using the Platreef 2017 FS flow sheet, was conducted at SGS Lakefield in 2020. This test was conducted on a blend composite containing geometallurgical units from various drill core intervals in the ratio 23% T1, 45% T2U, 24% T2L and 8.8% CZ. This test achieved a recovery (3PE+Au) of 85% at 82 g/t grade (3PE+Au).

Locked cycle testing indicated that targeting higher concentrate grades, in excess of 100 g/t (3PE+Au) resulted in reduced metal recoveries. 3PE+Au recovery was found to be dependent on the 3PE+Au head grade and target concentrate grade, which is typically referred to as an upgrade ratio in South African platinum processing terms. For Platreef, as higher 3PE+Au concentrate grades are targeted (High upgrade ratio) the overall concentrate mass pull will decrease and consequently the 3PE+Au recovery would be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio).

This relationship between 3PE+Au head grade, mass pull and final concentrate grade was used to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as summarised in Figure 13.1.

Figure 13.1 3PE+Au Recovery as a Function of Concentrate Grade



DRA, 2021

Further flotation testwork was conducted in 2020 and 2021 as part of the current Platreef 2022 FS. This testwork included open circuit, bench scale, batch flotation testwork using filtered site water representative of the expected water quality from the Masodi grey water system. These tests indicated that the filtered site water achieved a similar 3PE+Au upgrade profile as the baseline tests using Mintek tap water. A similar trend was observed for Cu and Ni.

An initial mini-pilot plant commissioning run was conducted at Mintek in June 2021 with a further commissioning run in November 2021. The aim of the mini pilot plant programme was to produce bulk concentrate samples for concentrate Kell hydrometallurgical refining testwork and concentrate de-watering testwork. Additionally, the intention was to derive process design information to supplement the design data as derived from bench scale flotation test work. The latter objectives were only partially achieved as the MINTEK mini-plant was not adequately commissioned, stabilized and optimized due to operational challenges which included stoppages due to power interruptions, inability to consistently dose copper collector reagent on a continuous basis, mechanical breakdowns and lack of assay data for operational control.

Concentrates generated during the June 2021 pilot run were re-floated in an 80 L flotation cell in batch mode in order toto produce timed kinetic samples for concentrate Kell testwork and concentrate de-watering test work at Metso Outotec South Africa (MO).

The November 2021 commissioning run of the Platreef circuit was conducted on a low grade bulk shaft intercept sample with a measured 2PE+Au head grade of 3.8 g/t. The run achieved stable mass flows however large variances in final concentrate mass pull resulted in combined final concentrate grades of approximately 50 g/t to 78 g/t 2PE+Au at a mass pull of 5% to 8%. The averaged metallurgical projection data indicates that the mini pilot run achieved an average 2PE+Au concentrate grade of 57 g/t at a recovery of 87% and 5.3% mass pull. A copper recovery of 87% was achieved and a nickel recovery of 81% was achieved.

The mini pilot plant data from the June 2021 and November 2021 runs are considered to reflect preliminary commissioning results on low grade samples and do not reflect representative metallurgical performance data.

Bench-scale filtration test work was conducted on bulk concentrates generated from the min pilot plant runs. The testwork achieved a high flux of approximately 600kg/m^{2}h for all samples tested and confirmed the potential to achieve a final concentrate moisture of <14% (w/w).

13.2 Previous Metallurgical Testwork

Various metallurgical testwork campaigns have been conducted using core samples from the Platreef deposit since October 2001. Prior to 2006, testing was conducted on predominantly lower-grade material from the potentially large open-pit area. In 2008, a deep drilling exploratory programme was launched, and the resource was updated to include deeper higher-grade material. Between 2010 and November 2014, a series of metallurgical testwork campaigns were carried out on the Platreef mineralised material as summarised in Table 13.1.

Table 13.1 Summary of Previous Metallurgical Testwork

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 1 (2010)	SGS Booysens	1 x TLZ-PX composite 1 x BLZ-PX composite 1 x TLZ-SP composite	Bench scale flotation testing included, grind optimisation testing, reagent scouting tests, locked cycle flotation testing and mineralogy.	A circuit that allowed for the recovery of base metal sulfides at a coarse grind followed by a regrind step to allow for improved metal recovery at the finer grind was believed to be the optimal processing route. This circuit is known as an MF2 configuration and is common in the South African PGE industry.
Phase 2 (2011–2012)	Xstrata Process Support	A master composite comprised of five geometallurgical units from the TCU, then called the Upper Unit TLZ 1 x T2U composite	Bench scale flotation testing, focused on mineralogy, obtaining baseline flotation test conditions and producing grade recovery relationships.	An optimised two stage milling and two stage flotation flow sheet, commonly referred to as MF2 in South African processing terms was developed based on results from grind optimisation testwork and reagent dosage testing. The development testwork at XPS was unable to produce a concentrate grade of 80–100 g/t and PGE recovery to final concentrate was approximately 60%.
Phase 3 (2012) Managed by Ivanplats and Wood	SGS Lakefield	1 x Master composite comprised of the newly classified geometallurgical units, namely, T1, T2 upper (T2U) and T2 Lower (T2L).	Bench scale flotation testing included flow sheet development and reagent scouting tests.	A single stage milling and flotation circuit, commonly referred to as MF1 in South African processing terms was developed. A reagent suite that included oxalic acid and thiourea addition with extensive conditioning time in the mill, prior to flotation, indicated that a PGE recovery of 83% could be achieved at a concentrate PGE (3PE) grade of 123 g/t, based on samples tested. High chrome media was used as grinding media in the laboratory mill.
Phase 4 (2012–2013) Managed by Ivanplats and Wood	Mintek	Drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and a composite containing 15%T1, 42.5% T2U and 42.5% T2L	Bench scale flotation testing included grind optimisation and flotation circuit development testing aimed at reproducing the SGS Phase 3 results.	The findings of the Mintek Phase 4 testing were largely in agreement with the SGS Phase 3 findings. A reagent suite that included oxalic acid and thiourea addition with conditioning in the mill prior to flotation indicated that a PGE recovery of 85% could be achieved at a concentrate PGE (4E) grade of 120 g/t. The type of grinding media used was also determined to be critical, with stainless steel and high chrome media consistently returning superior results compared to tests using carbon steel media. Locked cycle tests were conducted with an oxalic acid and thiourea in-mill conditioning time of 109 minutes which would not be possible to during commercial operation.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 5 (2013) Managed by Ivanplats and DRA	SGS Lakefield	Drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and footwall contaminated zone (CZ). Testing was conducted on domain composites as well as ore blend composites representing the expected mine blend ratios.	Determination of comminution parameters (JK data, BRWi, BBWi, Ai) Bench scale flotation testing included grind optimisation and flotation circuit development testing aimed at obtaining optimised flotation conditions and expected performance for a reduced oxalic acid and thiourea in-mill conditioning time of five minutes. Dewatering and rheology characterisation testwork.	Testing indicated that due to the high competency of both the mineralised material and the footwall composites with regard to SAG milling, a crusher and ball milling circuit would be better suited to the Platreef ore. The ball mill work indices indicated that the Platreef material can be classified as hard to very hard. Rougher kinetic testing indicated that the optimum mill grind was 80% passing 75 µm. Cleaner circuit optimisation testing highlighted that a split cleaner configuration, treating the fast, medium and slow floating PGE fractions separately, resulted in improved metallurgical performance. Locked cycle testing of a single stage milling and flotation circuit, with an oxalic acid and thiourea in-mill conditioning time of five minutes, indicated that a PGE recovery of 85% could be achieved at a concentrate 3PE+Au grade of 85 g/t. High chrome 440C grinding media was used in the laboratory mill. Phase 5 and Phase 6A test results formed the basis of the Platreef 2014 PEA flow sheet.
Phase 6A (2013) Managed by Ivanplats and DRA	Mintek	Drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and footwall contaminated zone (CZ). Testing was conducted on domain composites as well as ore blend composites representing the expected mine blend ratios.	Bench scale flotation testing included grind optimisation and flotation circuit development testing aimed at obtaining optimised flotation conditions and expected performance for a reduced oxalic acid and thiourea in-mill conditioning time of five minutes. Modal mineralogy and PGE investigations on flotation concentrates and tailings. Dewatering and rheology characterisation testwork.	Bench scale flotation testing confirmed that the type of grinding media impacted on flotation response, with stainless steel and high chrome media consistently returning superior results. locked cycle testwork indicated that a PGE recovery of 87.8% could be achieved at a concentrate PGE (3PE+Au) grade of 93 g/t. Mineralogical studies on tailings indicated that PGE losses to rougher tailings were comprised predominantly of PGE tellurides (43%) with lesser PGE arsenides (26%), gold (17%) and alloys (14%). Liberated PGE's in the rougher tailings were all in the <10 µm size fraction, which are regarded as non-recoverable by flotation. Phase 5 and Phase 6A test results formed the basis of the Platreef 2014 PEA flow sheet.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 6B (2014) Managed by Ivanplats and DRA	Mintek	Comminution variability testwork was conducted on approximately 1346kg of HQ drill core sample representing T1, T2 upper (T2U) and T2 Lower (T2L) and footwall contaminated zone (CZ). Flotation testing was conducted on drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and footwall contaminated zone (CZ). Testing was conducted on domain composites as well as ore blend composites representing the expected mine blend ratios.	Comminution variability testwork (Grindmill, UCS, CWi, Ai, BRWi, BBWi) Bench scale flotation testing aimed at optimising the Platreef 2014 PEA flow sheet and reagent suite. The testwork evaluated the effect of conditioning, depressant addition, alternate reagent suites and circuit configurations on metallurgical performance as compared to the results of locked cycle testing published as part of the Platreef 2014 PEA in Phase 5 and Phase 6A.	Bond ball work index test data and grindmill results confirmed previous competency testwork highlighting that the ore can be classified as hard. Evaluation of alternate flow sheets and reagent suites indicated that a standard flotation reagent suite, similar to that employed by platinum flotation operations in the vicinity of the Platreef deposit, provided comparable results to the oxalic acid and thiourea reagent suite when using an MF2 (mill-float followed by mill-float) circuit configuration. Preliminary MF1 (single stage mill and float) testing of an alternate reagent suite containing a targeted copper collector was able to achieve similar PGE, copper and nickel recovery and grades to those achieved for tests using an oxalic acid and thiourea reagent suite. In Phase 6B locked cycle testwork using the MF1 flow sheet and the oxalic acid and thiourea reagent suite achieved PGE (3PE+Au) recoveries of 82.8–87.5% and PGE (3PE+Au) grades of 78–96 g/t. A locked cycle test conducted on a footwall (contaminated zone) composite achieved 77% PGE recovery at a concentrate PGE (3PE+Au) grade of 57 g/t.

The findings from the Phase 1–Phase 6B testing were presented in the Platreef 2015 PFS.

The Platreef 2015 PFS flow sheet was based on a single stage milling circuit followed by flotation (MF1) using an oxalic acid and thiourea reagent suite with the inclusion of a post mill reagent conditioning stage.

At the time of the Platreef 2015 PFS, MF1 testing of an alternate reagent suite using a targeted copper collector was able to achieve similar PGE, copper and nickel recovery and final concentrate grades to those achieved during testwork using an oxalic acid and thiourea reagent suite. However, at the time, testing of this alternative copper collector reagent suite was ongoing, and further development of this flow sheet was incorporated into the Platreef 2017 FS metallurgical testwork scope and was used in the Platreef 2022 FS.

13.3 Platreef 2017 FS Testwork Samples and Scope

This section summarises metallurgical testwork carried out between January 2015 and November 2021 as follows:

- 2017 FS TestWork: January 2015 and October 2016 at Mintek, called Phase 7, under the management of technical teams from Ivanplats and DRA.
- Platreef 2022 FS Test Work: March 2020 and July 2020 at SGS Lakefield, called Phase 8, under the management of technical teams from Ivanplats and DRA.
- Platreef 2022 FS Test Work: March 2021 and November 2021 at Mintek, Thyssen Krupp and MO Group called Phase 9, under the management of technical teams from Ivanplats and DRA.

These testwork results were used in conjunction with the Platreef 2015 PFS testwork results, from Phase 5 and Phase 6, to further quantify the flotation response, support flow sheet development, evaluate the degree of variability and derive metal recovery estimates.

The Phase 7 testwork at Mintek included initial flow sheet development and optimisation testwork as well as ore variability assessments. Comminution variability testing was conducted on approximately 1,286 kg of HQ drill core representing samples from the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ). Flotation testing was conducted on approximately 1,140 kg of quarter PQ drill core samples representing samples from the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ). Subsequent to the selection of Platreef 2017 FS samples, the CZ has been re-named in the latest geological 3 model, more appropriately, to the Footwall Assimilation Zone (FAZ). It is however still referred to as CZ in this report and within the Mintek test reports.

The Phase 7 flotation testwork was aimed to further evaluate the alternative copper collector reagent suite identified in the Platreef 2015 PFS with the aim of deriving an optimal flow sheet for flotation variability testing. In addition to the bench scale flotation testwork, comminution variability testwork was conducted on the remaining variability drill core samples that were not tested in Phase 6B.

The Phase 8 flotation test work was aimed further characterising the flotation response of the Platreef 2022 FS composite sample, using conditions to replicate potential toll processing options and to compare this to the metallurgical performance using the optimized Platreef flowsheet as derived for the 2017 FS.

The Phase 9 flotation testwork was conducted on two core composite samples containing geometallurgical units from various drill core remainders from the FS variability drilling campaign and two bulk composites of shaft intercept material obtained from the surface stockpiles. The aim of the test work was to derive HPGR design parameters, evaluate the potential for Jameson cell technology in the cleaner circuit, determine the effect on flotation response when using site water and conduct preliminary pilot scale test work to produce bulk concentrate samples for settling, filtration and Kell hydrometallurgical refining test work.

13.3.1 Platreef 2017 FS Sample Selection

13.3.1.1 Comminution Variability Test Samples

Comminution testwork was conducted using samples representing the following Table 13.2:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example, T1 from drill core hole TMT009.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example, a blend of T1, T2U, T2L, CZ and HW from drill core hole TMT009.
- Domain composites representing blends of individual geometallurgical units from various drill core samples. For example, a blend of T1 from drill core holes TMT009 – TMT016.

Table 13.2 Comminution Variability Samples

Core ID	Total Mass (kg)	T1	T2U	T2L	CZ	HW
TMT009	256.4	X	X	–	X	X
TMT010	149.2	X	–	X	X	X
TMT011	257.7	X	–	X	X	X
TMT012	180.0	X	X	X	–	X
TMT013	85.5	X	X	–	–	–
TMT014	102.0	X	–	X	–	X
TMT016	255.3	X	X	X	–	X
Total	1,286.1	7	4	5	3	3*

*Three hanging wall composites representing T1, T2U and T2L hanging wall were tested.

13.3.1.2 Flotation Testwork Samples

Ivanplats delivered quarter PQ drill core samples representing geometallurgical units T1, T2 upper (T2U), T2 Lower (T2L), hanging wall (HW) and footwall (CZ) to Mintek in Johannesburg. The Platreef 2017 FS flotation testwork samples are listed in Table 13.3 below, and the locality of the drill core samples is shown in Figure 13.2.

Flotation development testing was conducted on samples representing the following:

- Blend composites representing blends of geometallurgical units from various drill core samples. For example, a blend of 14% T1, 51% T2U, 30% T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

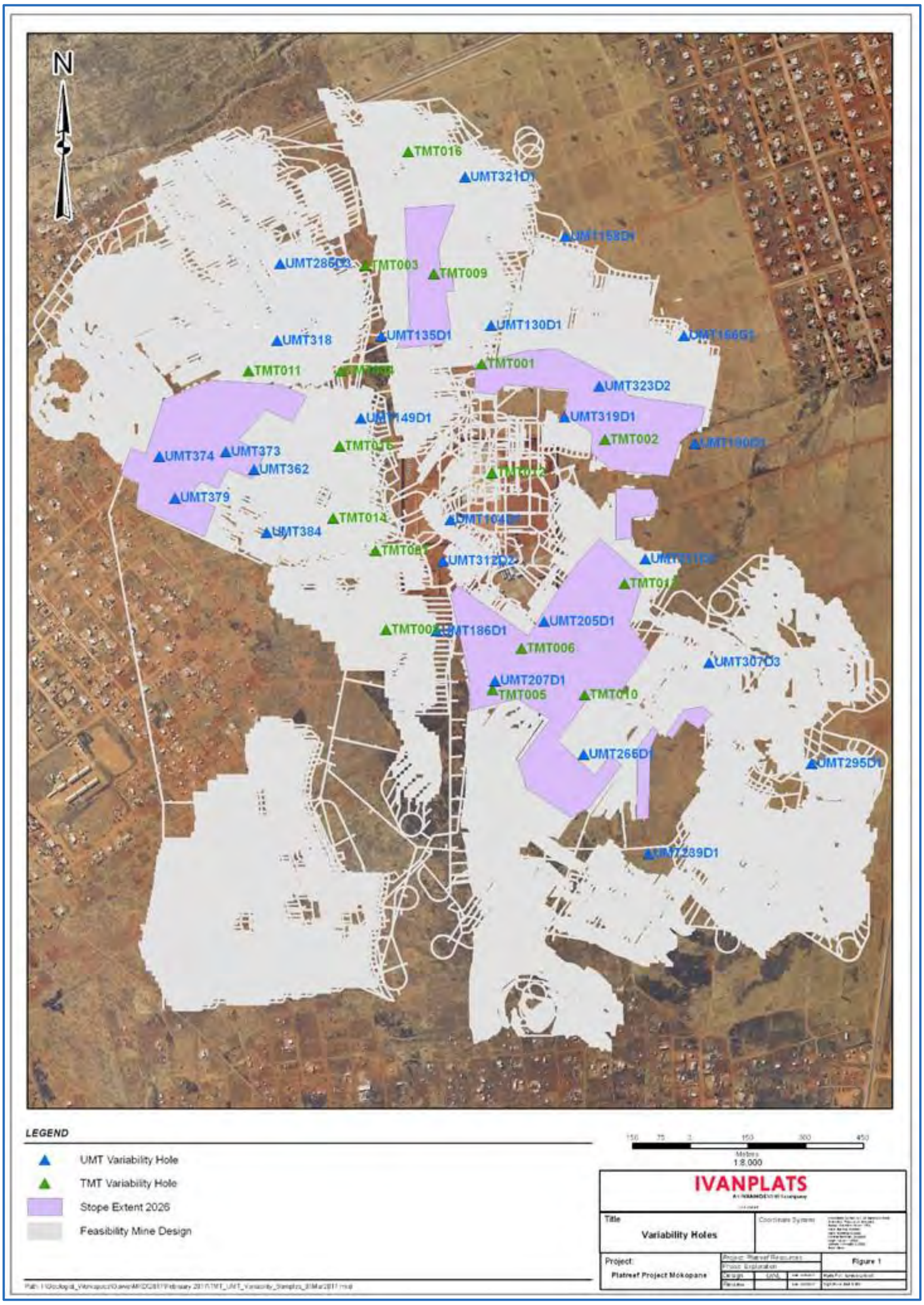
Flotation variability testing was conducted on samples representing the following:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example, T1 from drill core hole UMT104D1.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example, a blend of T1, T2U, T2L and CZ from drill core hole UMT166D1.
- Blend composites representing blends of geometallurgical units from various drill core samples. For example, a blend of 14% T1, 51% T2U, 30% T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

Table 13.3 Flotation Variability Samples

Core ID	Total Mass (kg)	T1	T2U	T2L	CZ	Point Sample Blend Composite
UMT104D1	12.0	X	–	–	X	X
UMT130D1	83.7	X	X	–	X	X
UMT135D1	104.0	X	X	–	X	X
UMT149D1	56.7	X	–	X	X	X
UMT158D1	54.0	X	–	X	X	X
UMT166D1	76.8	X	X	X	X	X
UMT186D1	57.6	X	–	X	X	X
UMT190D1	59.5	X	X	–	X	X
UMT205D1	42.0	–	X	–	X	X
UMT207D1	43.2	–	X	X	–	X
UMT211D1	35.9	X	X	–	–	X
UMT239D1	36.0	X	–	X	–	X
UMT265D1	66.3	X	X	X	X	X
UMT286D3	42.4	X	X	–	X	X
UMT295D1	17.3	–	–	–	X	X
UMT307D3	56.0	X	X	–	X	X
UMT312D2	41.9	–	X	–	X	X
UMT319D1	63.0	X	–	X	X	X
UMT321D1	82.2	X	X	X	–	X
UMT323D2	62.0	X	X	X	–	X
UMT318	5.2	–	X	X	X	X
UMT362	11.5	X	–	X	X	X
UMT373	6.7	–	X	X	X	–
UMT374	12.6	X	X	–	–	X
UMT379	4.6	X	X	–	–	X
UMT384	7.4	–	X	X	X	–
Total	1,140.0	19	18	14	19	24

Figure 13.2 Variability Sample Drill Core Location



Ivanplats, 2017

13.3.2 FS Testwork Scope

The Platreef 2017 FS testwork scope is summarised in Table 13.4.

Table 13.4 2017 FS Testwork Scope

Testwork Description	Laboratory	Samples	Scope
Comminution Variability Testing	Mintek	7 x T1 4 x T2U 5 x T2L 3 x HW 1 x CZ	Bond crushability work index (CWi) Bond abrasion index (Ai) Bond rod work index (BRWi) Bond ball work index (BBWi)
Chemical Composition and Mineralogy	Mintek	6 x T1 7 X T2U 3 x T2L 6 X CZ	Bulk mineralogy (x-ray diffraction (XRD)) Base metal sulfide (BMS) characterisation Platinum group metals (PGM) search
Open Circuit Flotation Development and Optimisation Testwork	Mintek	Platreef 2015 PFS Composite Development Composite 1 Development Composite 2 Development Composite 3 Development Composite 4 Development Composite 5	MF1 copper collector flow sheet development Sulfur optimisation Grade optimisation Flow sheet validation testing
Open Circuit Flotation Variability Testwork	Mintek	16 x T1 13 x T2U 10 x T2L 15 x CZ 26 x Blend Composites	Open circuit variability tests using optimised Platreef 2017 FS flow sheet
Locked Cycle Testing	Mintek	Development Composites 1,3,4 and 5 4 x Year 1–5 Blend 1 x Life of Mine (LOM) Blend	Locked cycle testing of interim flow sheets; Locked cycle variability testing of optimised Platreef 2017 FS flow sheet
Dewatering Testwork	Vietti Slurry Technology	Blend of FS Composites 1–4	Thickening testwork

13.3.3 Platreef 2022 FS Sample Selection

Ivanplats delivered approximately 70kg of quarter PQ drill core samples representing geometallurgical units T1, T2 upper (T2U), T2 Lower (T2L), HW and footwall (CZ) to SGS Lakefield in 2020. The drill core sample intervals reflect un-crushed drill core remainders from the 2017 FS variability testwork drilling campaign. SGS prepared the Phase 8 Platreef 2022 FS blend composite.

Ivanplats delivered approximately 2.9 t of quarter PQ drill core samples representing geometallurgical units T1, T2 upper (T2U), T2 Lower (T2L), HW and footwall (CZ) and two bulk samples sample weighing approximately 23 t and 24 t to Mintek in 2021. Mintek prepared the Phase 9 Platreef 2022 FS blend composites for bench scale and pilot scale test work. The drill core sample intervals reflect un-crushed drill core remainders from the 2017 FS variability testwork drilling campaign while the bulk samples reflect of crushed rock samples from the current surface stockpiles.

13.3.4 Platreef 2022 FS Testwork Scope

The Platreef 2022 FS testwork scope, is summarised in Table 13.5.

Table 13.5 Platreef 2022 FS Testwork Scope

Testwork Description	Laboratory	Samples	Scope
Comminution Test Work	Thyssen Krupp	Shaft intercept bulk composite #1	HPGR Test Work
Open Circuit and Locked Cycle Flotation Test Work (2020)	SGS Lakefield	Phase 8 Core Blend Composite	Bench Scale Flotation Test Work
Open Circuit Flotation Test Work (2021)	Mintek	Phase 9 Core Blend Composites Shaft intercept bulk composite #1	Bench Scale Flotation Test Work
Mini Pilot Plant Commissioning and Bulk Concentrate Production	Mintek	Phase 9 Core Blend Composites Shaft intercept bulk composite #1 and #2	Mini Pilot Plant Commissioning and Bulk Concentrate Production
Concentrate Dewatering Test Work	MO Group	Bulk Concentrates generated from shaft intercept comp # 1	Concentrate Dewatering Test Work
Concentrate Bulk Handling Test Work	Green Technical	Bulk Concentrates generated from shaft intercept comp # 1	Concentrate Bulk Handling Test Work

13.3.4.1 Testwork Sample QA/QC

During the PFS, 2017 FS and Platreef 2022 FS metallurgical test campaigns, geological core log interval assay data from Set Point was compared to Mintek and SGS assay data to assess if any major bias or analytical deviation existed. No bias was noted between the PGE, Cu, Ni and S assay values in comparison to the core log data. In addition, during the 2017 FS test work campaign, 90% of all PGE, Cu, Ni and S analysis data (80 tests) was found to fall within a 10% variance between the assay data as compared to the core log data. Mintek's QA/QC reports were also reviewed in 2017 and found to be satisfactory.

During the Platreef 2022 FS mini-pilot campaign, Mintek process control sample assay data from select mini-pilot runs was compared to the assay results from SGS and found to be in good agreement.

13.4 Ore Mineralogy

During the 2017 FS test work programme, 22 run-of-mine (ROM) samples obtained from individual drill cores, representing the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ), were sent for mineralogical analyses and characterisation. The objective of the mineralogical investigation was to identify the minerals present, the platinum group elements (PGEs), the base metal sulfides (BMS), grain sizes, liberation characteristics, mineral associations and to describe their mode of occurrence.

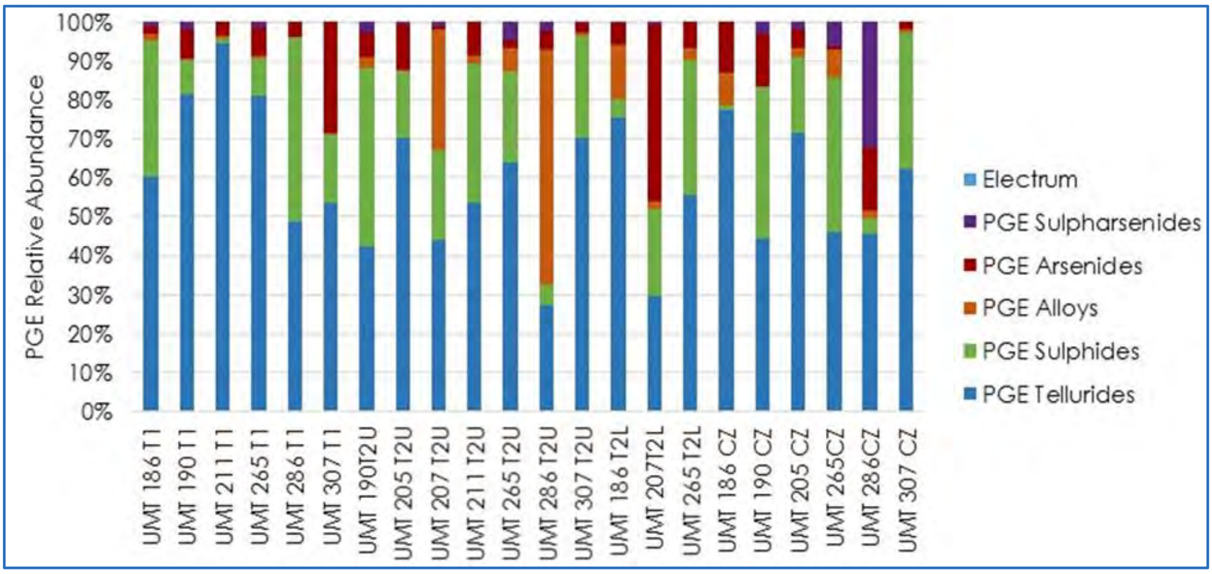
A maximum of 20 unsized polished sections of one portion were prepared for PGE and BMS search analyses by automated scanning electron microscope (AutoSEM). A second sub-sample was pulverised and submitted for X-Ray diffraction analysis (XRD). The XRD results were used to verify the minerals identified during by AutoSEM modal analysis.

The mineralogical study indicated that pyroxene is the predominant mineral Phase present throughout all 22 samples, followed by minor to trace amounts of olivine, dolomite, magnetite, serpentine, clay minerals, quartz, amphibole, chlorite, mica, plagioclase, talc, pentlandite, chalcopyrite and pyrrhotite.

13.4.1 PGE Relative Abundance and Liberation

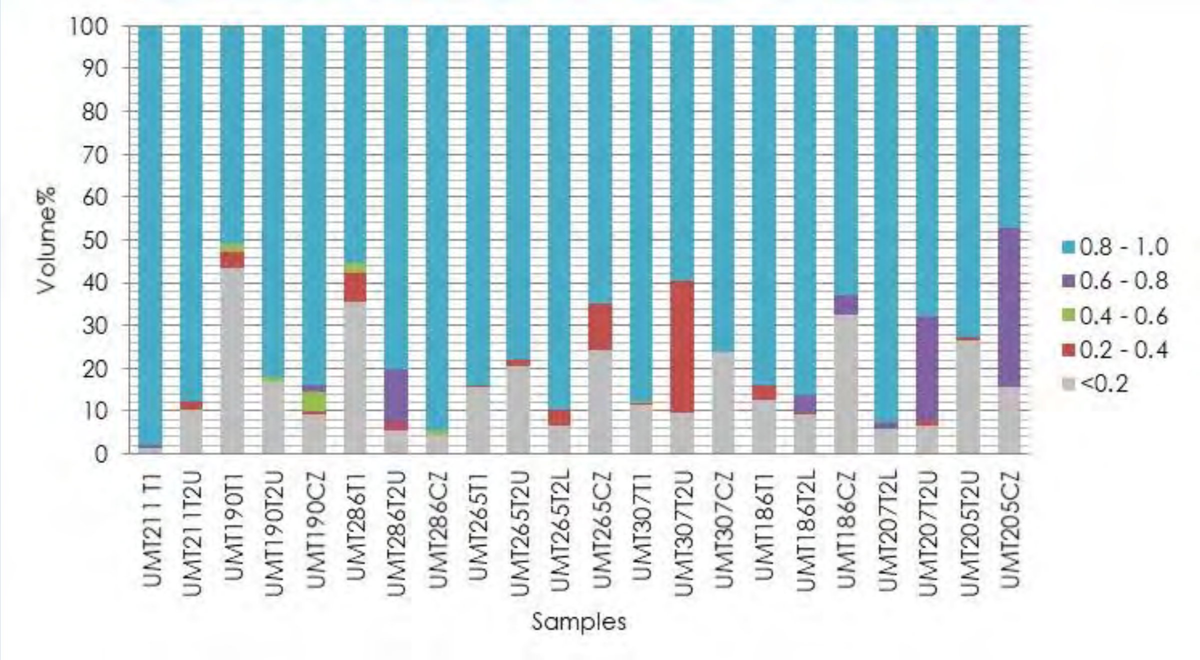
A summary of the PGE relative abundance, liberation index and floatability is presented in Figure 13.3 to Figure 13.5.

Figure 13.3 Variability Sample PGE Relative Abundance



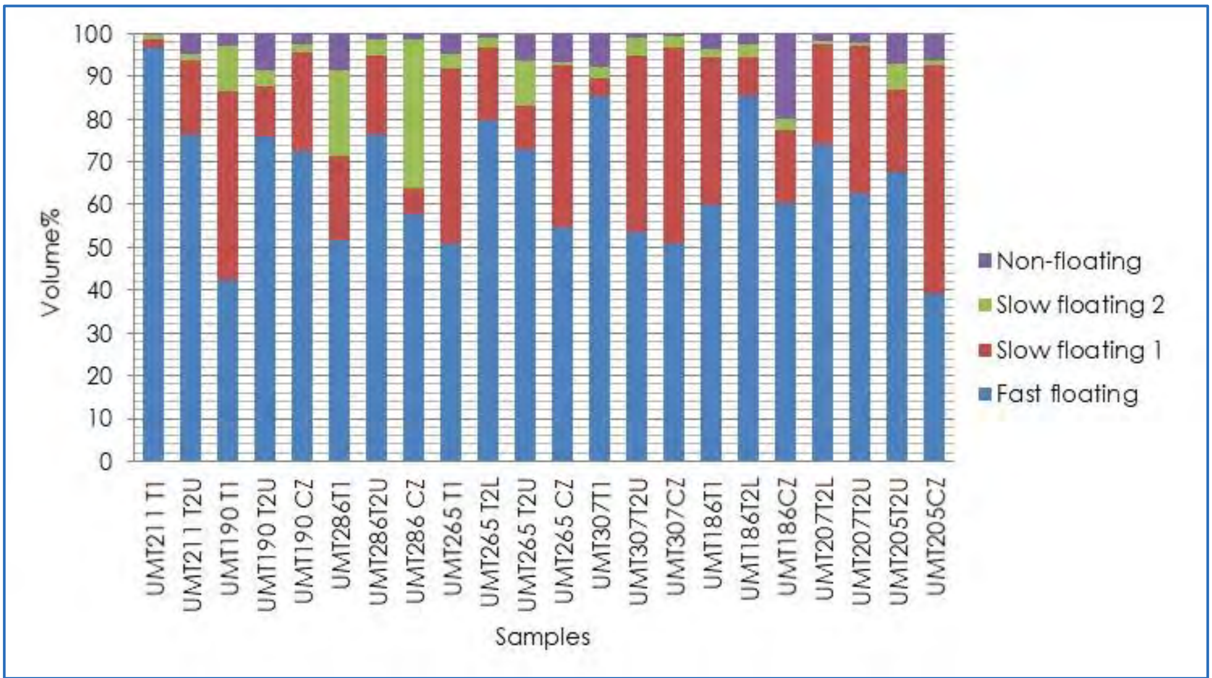
DRA, 2017

Figure 13.4 Variability Sample PGM Liberation Index



DRA, 2017

Figure 13.5 Variability Sample PGE Floatability Index



DRA, 2017

Mineralogy indicated the following with regard to PGE speciation and liberation:

- PGE-tellurides are the dominant PGM species, averaging 59% by volume in the samples tested, followed by lower amounts of PGE-arsenides, PGE-sulfides and PGE-others. Laurite is present in minor to trace amounts in only a few samples.
- The majority of PGMs identified in the samples, reported to the 0–15 µm size class and were found to be well liberated, however, more than 20% (by volume) of PGM grains observed in the UMT190 T1, UMT265T2U, UMT265 CZ, UMT307 CZ, UMT186 CZ and UMT205 T2U samples were associated with relatively large gangue particles (<0.2 liberation index) compared to the other samples where low liberation indices occurred in minor to trace amounts.
- The PGM fast floating fraction varied in the range 40–97% (by volume) and averaged 66%. A significant portion of slow floating PGMs were identified (3–55% (by volume)).

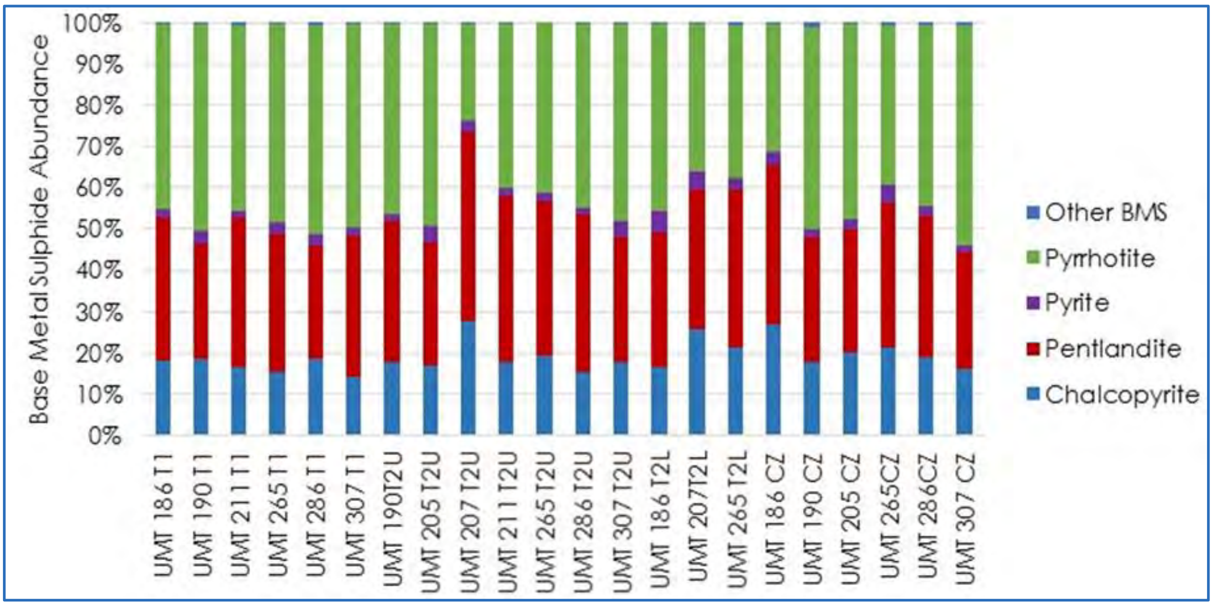
13.4.2 Base Metal Sulfide Mineralisation and Liberation

A summary of the base metal sulfide mineralisation is presented in Figure 13.6.

Mineralogy indicated the following with regard to BMS mineralogy and Liberation:

- Pentlandite and pyrrhotite are the dominant sulfide mineral phases present, with minor to trace amounts of chalcopyrite and trace amounts of pyrite. The majority of pentlandite, pyrrhotite and chalcopyrite were found to be associated with free surfaces, and minor to trace amounts are associated with gangue and other BMS mineral phases.
- Pentlandite, pyrrhotite and chalcopyrite grain size was found to be variable. The majority of pyrite grains, however, occur in the <10 µm size class.
- Pentlandite displays a similar high cumulative liberation in all samples except UMT265 CZ, where it is poorly liberated.
- Chalcopyrite displays a similar cumulative liberation in all samples except UMT211 T1, UMT190 T1, UMT265 T1 and UMT307 T1, where it is poorly liberated.
- Pyrite was found to be poorly liberated, while pyrrhotite was found to be well liberated in all samples except samples UMT186 T2L and UMT207 T2U.

Figure 13.6 Variability Sample Base Metal Mineralisation



DRA, 2017

13.5 Comminution Variability Testwork

In addition to the Phase 6B comminution variability testwork that was conducted as part of the Platreef 2015 PFS, additional comminution variability testing, was conducted as part of the Platreef 2017 FS testwork programme. In total 22 variability samples were tested as part of the Phase 7 comminution testwork campaign.

13.5.1 Comminution Variability Test Results

A summary of all the comminution variability test results is presented in Table 13.6 to Table 13.8.

Table 13.6 Summary of Comminution Results for Year 1 to Year 5 Samples

Description		BRWi (kWh/t)	BBWi (kWh/t)	CWi (Avg) (kWh/t)	Ai (Avg) (g)
Domain Point Samples					
TMT004	T1	–	–	12.5	0.40
TMT006	T1	–	–	13.4	0.27
TMT007	T1	–	–	14.1	0.34
TMT009	T1	13.7	21.3	16.8	0.47
TMT011	T1	14.2	21.7	21.1	0.34
TMT014	T1	15.7	20.7	20.9	0.37
TMT002	T2U	14.6	21.6	16.2	0.45
TMT004	T2U	–	–	12.4	0.38
TMT005	T2U	–	–	14.0	0.42
TMT006	T2U	–	–	12.9	0.33
TMT007	T2U	–	–	13.6	0.31
TMT009	T2U	15.6	22.2	23.4	0.41
TMT005	T2L	–	–	8.4	0.06
TMT011	T2L	19.1	23.6	15.3	0.12
TMT014	T2L	19.8	22.2	13.8	0.11
TMT004	CZ	16.7	19.0	12.4	0.41
TMT006	CZ	20.9	23.7	17.5	0.10
TMT007	CZ	16.0	20.7	10.6	0.27
Point Sample Blend Composites					
TMT 004	45%T1:55%T2U	16.2	21.6	–	–
TMT 005	20.%T2U:80%T2L	18.7	22.9	–	–
TMT 006	47%T1:53%T2U	16.8	20.0	–	–
TMT 007	70%T1:30%T2U	19.2	23.6	–	–

Table 13.7 Summary of Comminution Results for Year 6+ Samples

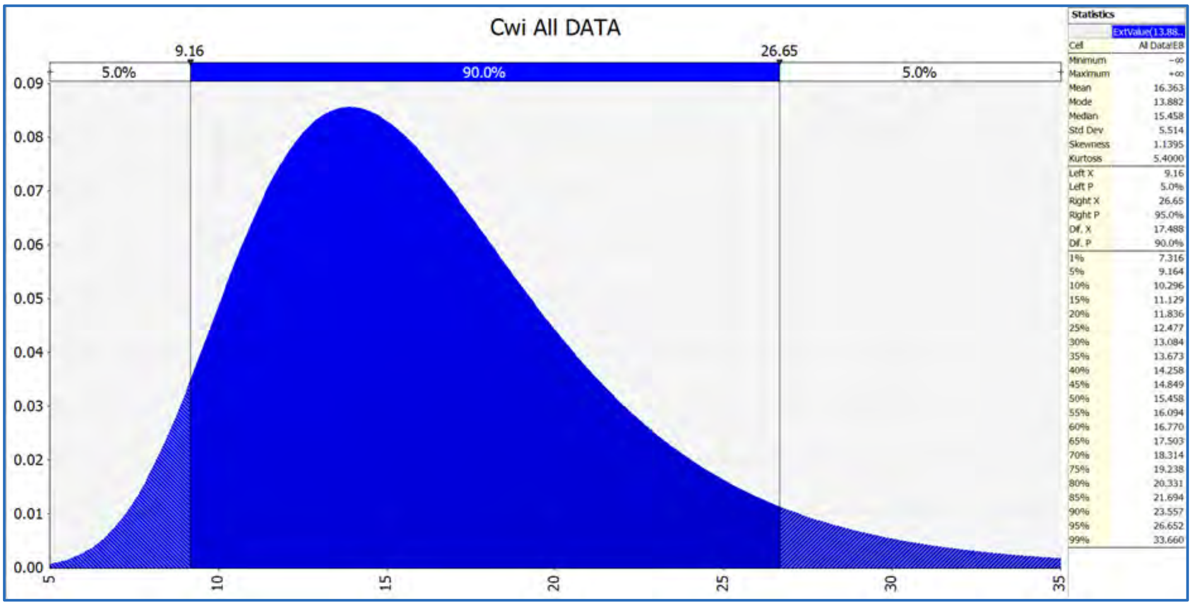
Description		BRWi (kWh/t)	BBWi (kWh/t)	CWi (Avg) (kWh/t)	Ai (Avg) (g)
Domain Point Samples					
TMT001	T1	–	–	12.4	0.39
TMT003	T1	–	–	14.9	0.39
TMT008	T1	–	–	13.4	0.32
TMT010	T1	17.2	18.9	19.2	0.22
TMT012	T1	17.0	20.3	21.0	0.34
TMT013	T1	15.0	22.4	15.3	0.39
TMT016	T1	16.6	20.9	18.7	0.26
TMT001	T2U	–	–	16.2	0.45
TMT003	T2U	–	–	12.4	0.38
TMT008	T2U	–	–	14.0	0.42
TMT012	T2U	12.0	18.3	12.9	0.33
TMT013	T2U	19.6	23.6	13.6	0.31
TMT016	T2U	17.1	23.7	23.4	0.41
TMT003	T2L	–	–	15.5	0.20
TMT008	T2L	–	–	14.3	0.11
TMT010	T2L	19.2	23.4	14.2	0.12
TMT012	T2L	20.0	23.2	17.1	0.24
TMT016	T2L	23.0	25.4	21.1	0.25
TMT001	CZ	19.7	22.9	10.4	0.26
TMT003	CZ	17.1	21.3	12.5	0.44
TMT008	CZ	15.2	21.9	17.6	0.40
Point Sample Blend Composites					
TMT 001	41.% T1:59% T2U	16.6	20.7	–	–
TMT 003	25% T1:11%T2U:64% T2L	18.6	19.9	–	–
TMT 008	24%T1:3%T2U:73%T2L	18.4	21.9	–	–

Table 13.8 Summary of Comminution Results for the Domain Composite Sample

Description	BRWi (kWh/t)	BBWi (kWh/t)	CWi (Avg) (kWh/t)	Ai (Avg) (g)
Domain Composites				
T1 (TMT 001–008)	–	20.5	17.4	0.35
T2U (TMT 001–008)	–	22.2	26.0	0.38
T2L (TMT 001–008)	–	23.4	14.4	0.14
CZ (TMT 001–008)	–	22.4	16.7	0.35
CZ (TMT 009–016)	17.2	24.4	18.0	0.33
T1-HW (TMT 001–008)	–	20.3	22.1	0.27
T1-HW (TMT 001–008)	16.6	20.8	13.0	0.36
T1-HW (TMT 009–016)	16.6	20.2	21.7	0.36
T2-HW (TMT 001–008)	–	21.2	11.8	0.40
T2-HW (TMT 001–008)	16.2	20.4	15.9	0.36
T2U-HW (TMT 009–016)	16.0	21.3	17.4	0.36
T2L-HW (TMT 009–016)	16.2	21.4	20.1	0.40

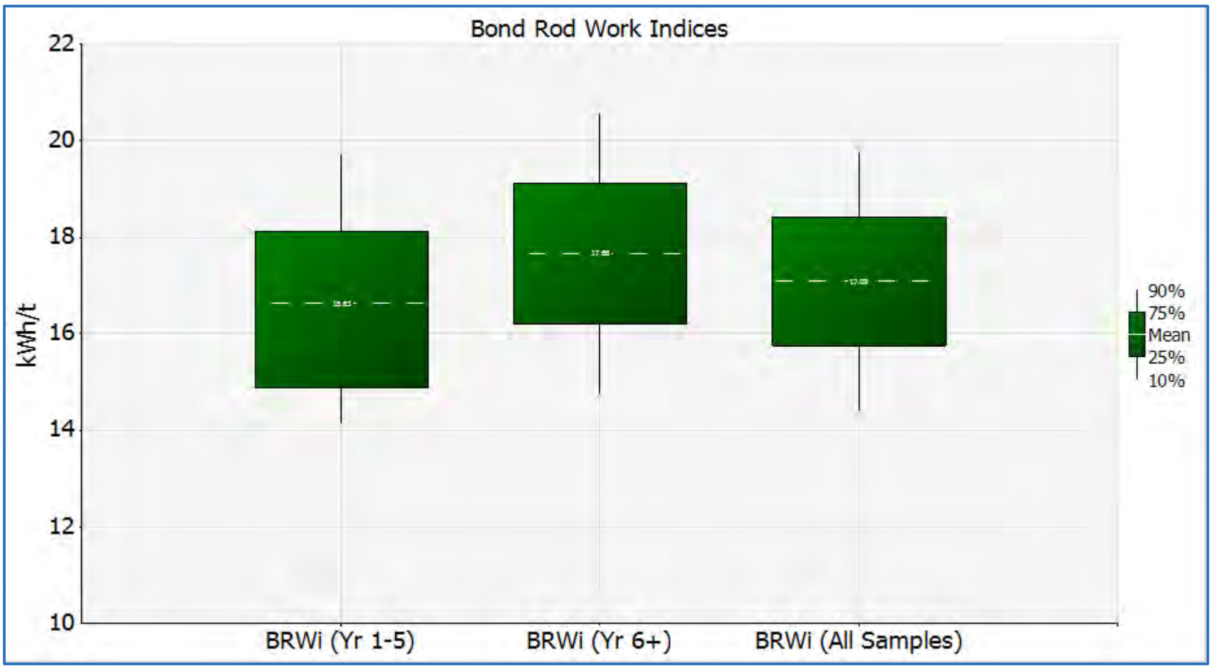
The frequency distributions obtained using all comminution results conducted in Phase 6B and Phase 7 were analysed for each ore type and are presented in Figure 13.7 to Figure 13.10.

Figure 13.7 Probability Distribution of CWi Data



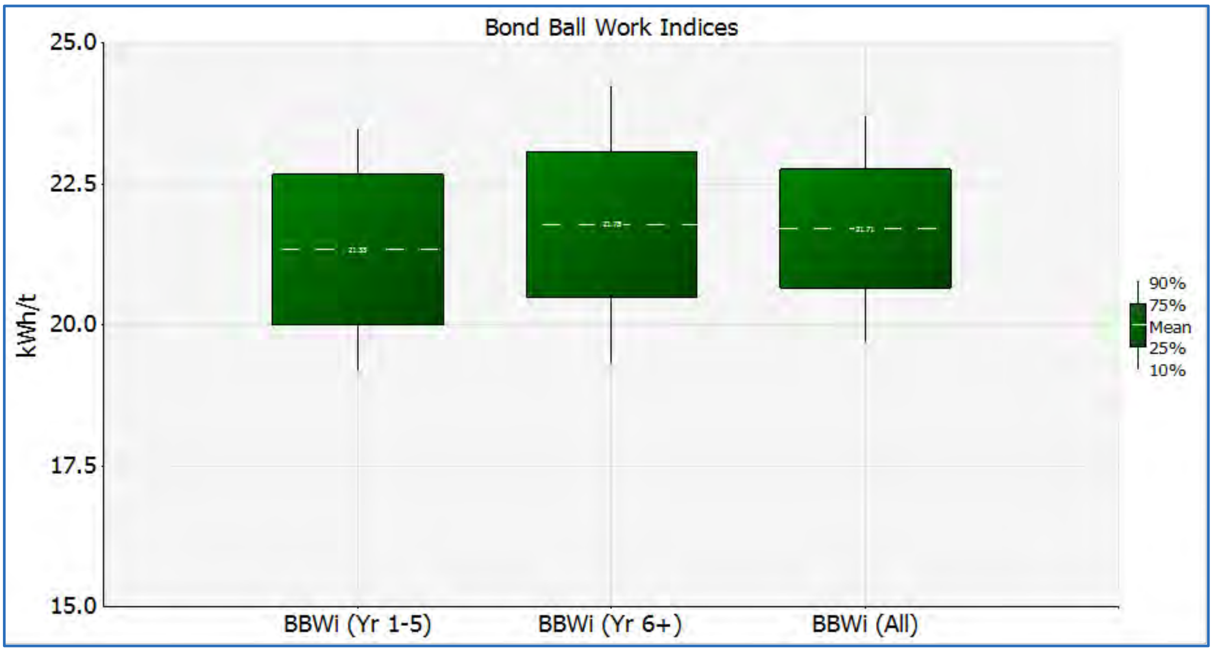
DRA, 2017

Figure 13.8 Distribution of BRWi Variability Data



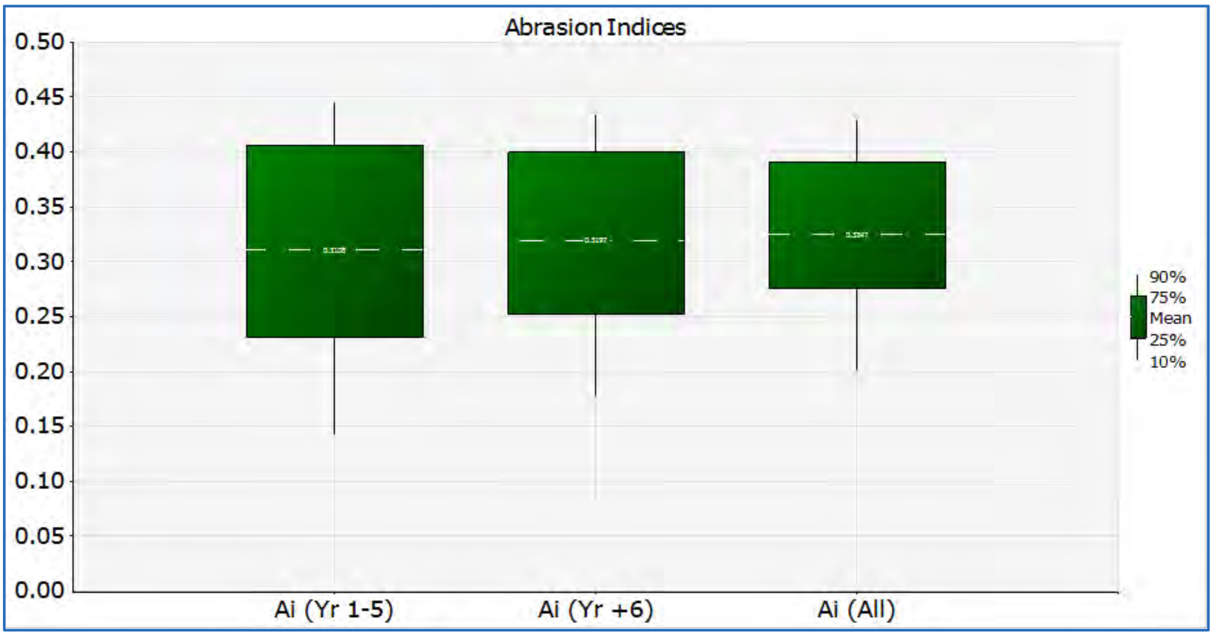
DRA, 2017

Figure 13.9 Distribution of BBWi Variability Data



DRA, 2017

Figure 13.10 Distribution of Ai Variability Data



DRA, 2017

Based on the crusher work index data the ore tested can be classified as being variable with medium-hard to very hard classification. The T2L and CZ ore types were noted as having a lower crusher work index as compared to the other ores and were classified as medium-hard to hard.

For the 90% confidence interval, Bond rod work indices were in the range 14.2–19.5 kWh/t in the first five years of mining and 15.0–20.5 kWh/t in the later production years. The ore tested can be classified as medium-hard to very hard. The T2L domain samples indicated a higher resistance with respect to rod milling, with hardness classification of hard to very hard. The Bond rod work index data has indicated an approximate 1.0 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1 to Year 5 samples.

For the 90% confidence interval, Bond ball work indices were in the range 19.0–23.0 kWh/t in the first five years of mining and 19.4–24.2 kWh/t in the later production years. The samples were competent with respect to ball milling with hardness classification of hard to very hard. The T2L domain samples indicated a higher resistance with respect to ball milling. The Bond ball work index data has indicated an approximate 0.5 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1 to Year 5 samples.

The abrasion index was found to fall within a similar range of approximately 0.25–0.50 g, for all ore types, with the exception of T2L, with an abrasion index of 0.06–0.25 g. The ore can be classified as having a medium abrasion tendency.

The comminution variability testwork indicated no significant difference in the mean competency between the ore from the early production years (Year 1 to Year 5) and the later production years. Average Bond ball work indices indicate that the ore from the later production years was approximately 5% harder than ore from the first five years of mining.

There was a minor variation in hardness for the T1 and T2U domain samples as compared to the T2L and CZ domain samples as summarised in Table 13.9.

Table 13.9 Summary of Comminution Classification per Domain

Domain	CWi (kWh/t)			BRWi (kWh/t)			BBWi (kWh/t)		
	P_{50}	P_{85}	Class	Min.	Max.	Class	Min.	Max.	Class
T1	16.0	21.6	Medium–Hard	13.7	18.9	Medium–Hard	17.2	22.4	Hard–Very Hard
T2U	15.9	24.2	Medium–Very Hard	12.0	18.3	Medium–Hard	19.6	23.7	Hard–Very Hard
T2L	13.5	18.8	Medium–Hard	19.1	22.2	Hard–Very Hard	23.0	25.4	Very Hard
CZ	14.3	18.8	Medium–Hard	15.2	19.0	Hard	20.9	23.7	Very Hard
T1-HW	17.5	26.1	Hard–Very Hard	–	16.6	Hard	20.3	20.8	Very Hard
T2-HW	14.7	22.3	Medium–Very Hard	16.0	16.2	Hard	20.4	21.4	Very Hard

The 85th percentile test data was used as the design basis for the comminution circuit. The crusher circuit was sized based on the 85th percentile crusher work index results. The grinding circuit was sizing was based on 85th percentile, Bond work index results, which were used in combination with particle breakage rates derived from grindmill testing. The average ore hardness and abrasion index data has been used to derive the operating cost estimates, whilst effect of variability is also quantified.

13.6 HPGR Testwork

The Phase 9 Platreef 2022 FS test work programme included multiple single-pass crushing tests which were conducted by Thyssenkrupp using a semi-pilot scale HPGR (SMALLWAL). The test work was conducted on bulk shaft intercept composite sample #1 which was sourced from the current surface stockpiles.

The aim of these tests was to provide data to evaluate expected HPGR throughput, size reduction and power consumption performance. In addition to this, ATWAL wear tests were conducted, to provide data for operating wear rate predictions when treating the Platreef material. The HPGR test work results can be summarized as follows:

- SMALLWAL test work achieved a specified throughput of 310 to 338 ts/m³h at a specific energy of 0.94 to 1.47 kWh/t with a product PSD of 77 to 83% passing 8 mm and 55 to 61% passing 3 mm. Maximum product fineness was achieved at a grinding force of 42 N/mm², but at significantly higher energy input. The optimum operating pressure for this material was determined to be approximately 3.0 N/mm².
- The ATWAL specific wear rate was 12.9 to 13.2 g/t at a specific grinding force of 4 N/mm². The sample was classified as low/medium abrasive material.

13.7 Flotation Circuit Development Testwork

As part of the Platreef 2017 FS, open circuit flow sheet development and optimisation flotation testwork was conducted. All testwork was conducted at bench scale using industry standard testing techniques. The aim of this testing was as follows:

- Further development and optimisation of the MF1 flow sheet using the copper collector reagent suite. This testing included evaluation of optimal grind, circuit configuration, depressant addition and evaluating flotation circuit solution pH.
- Compare the performance of the optimised MF1 flow sheet using the copper collector reagent suite to the MF1 flow sheet using the oxalic and thiourea reagent suite presented in the Platreef 2015 PFS.
- Evaluate the effect of rougher solids concentration.
- Reduce pyrrhotite (sulfur) recovery to final concentrate.
- Optimise final concentrate PGE (3PE+Au) grade.
- Evaluate the effect of sample composition and head grade.
- Finalise the flotation flow sheet for open circuit variability and locked cycle testing.

13.7.1 Development Sample Assays

The measured head assay data for the development testwork composites is presented in Table 13.10. The development Composites 1–4 represent an ore blend of 12%T1, 32%T2U, 50%T2L and 6% CZ, and the Platreef 2015 PFS Composite (PFS Composite) represents an ore blend of 15%T1, 42.5%T2U and 42.5%T2L.

Table 13.10 Development Sample Measured Head Assays

Test ID	Grade									
	3PE+Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
PFS Composite	3.99	1.79	1.87	0.10	0.23	0.21	0.39	1.00	24.36	10.13
Development Composite 1	3.25	1.34	1.62	0.06	0.23	0.20	0.39	1.28	23.02	9.15
Development Composite 2	4.11	1.83	1.94	0.10	0.24	0.21	0.40	1.04	26.36	10.13
Development Composite 3	3.48	1.53	1.61	0.10	0.24	0.18	0.36	0.81	27.56	10.15
Development Composite 4	5.71	2.49	2.70	0.19	0.33	0.21	0.46	0.97	27.06	10.40
Development Composite 5	5.39	2.24	2.70	0.15	0.30	0.21	0.46	0.97	25.07	9.77

Composites 1 and 2 were selected using the final Platreef 2015 PFS mine blend and represent a low 3PE+Au grade of 3.25 g/t (Composite 1) and the average expected PFS LOM 3PE+Au grade of approximately 4.1 g/t (Composite 2), while Composite 3 was selected to specifically target a lower sulfur grade in feed of 0.8% as per the expected sulfur grade in the early years of mining as compared to the >1.0% sulfur for Composites 1 and 2.

Composites 4 and 5 were selected using the final Platreef 2015 PFS mine blend and represent the higher grade 3PE+Au feed grade of approximately 5.50 g/t that will be targeted in the early years of mining.

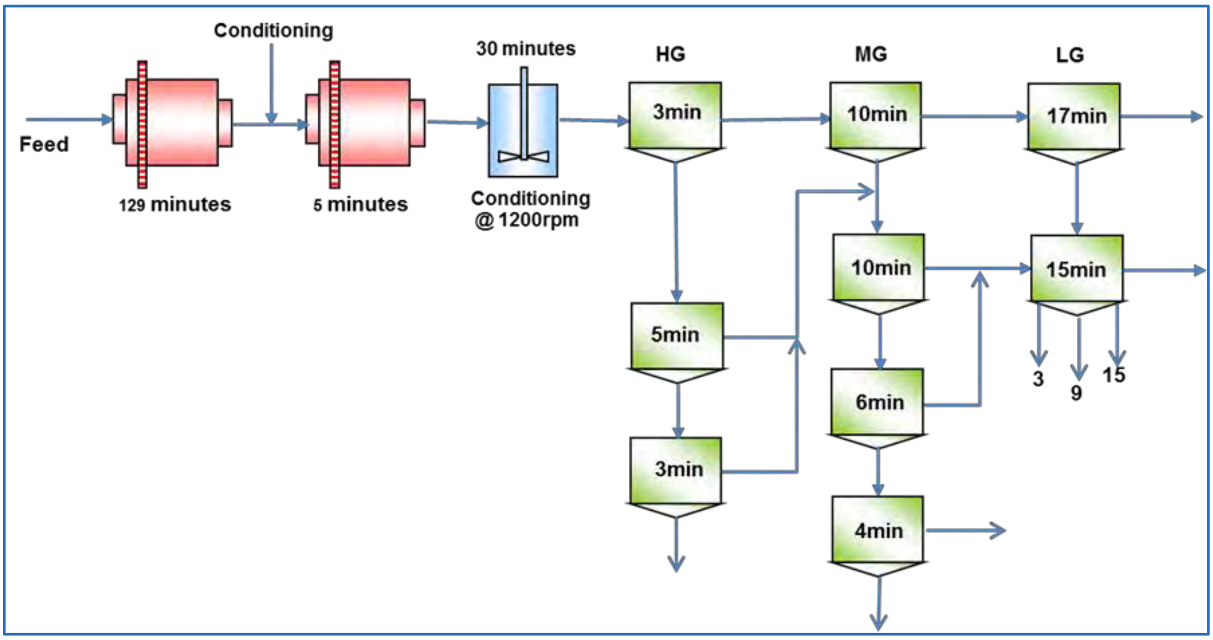
13.7.2 Summary of Results

13.7.2.1 Development Testing of the MF1 Copper Collector Reagent Suite

Development testing of the MF1 copper collector flow sheet was conducted on the remaining PFS Composite sample, to evaluate the effect of grind, pH and depressant addition for the MF1 copper collector reagent suite. The remaining Platreef 2015 PFS sample was used in order to allow for a direct comparison to the Platreef 2015 PFS test results for the oxalic acid and thiourea reagent suite (PFS Test 69) and MF1 copper collector reagent suite (PFS Test 54).

A summary of the flow sheets used in this testing is presented in Figure 13.11and Figure 13.12. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.11.

Figure 13.11 Oxalic Acid and Thiourea Reagent Flow Sheet (Platreef 2015 PFS)



DRA, 2017

Figure 13.12 Copper Collector Reagent Flow Sheet (Platreef 2017 FS)

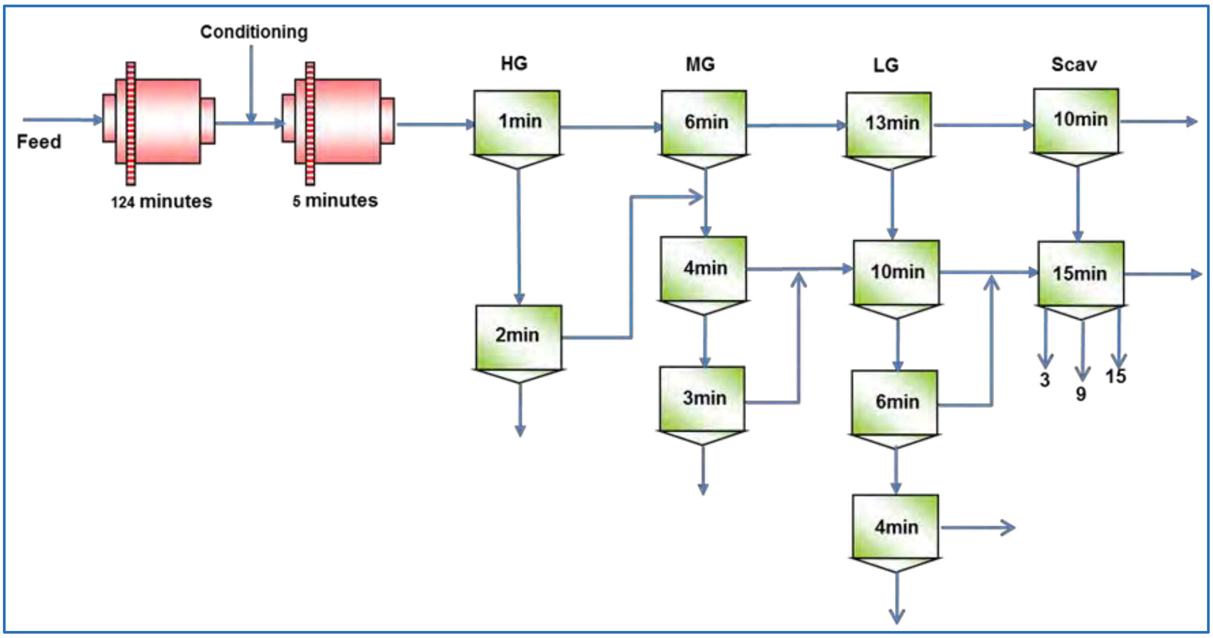


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Table 13.11 Results of MF1 Copper Collector Flow Sheet Development and Optimisation Testing

Test No.	Sample	Reagent Suite	Grind P_{80}	Depressant		pH	Mass (%)	Grade					Recovery				
				Rougher (g/t)	Cleaner (g/t)			3PE+Au (g/t)	Cu (%)	Ni (%)	Fe (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	Fe (%)	S (%)
69 PFS	PFS Comp	Oxalic & Thiourea	75	–	93	Natural	4.10	75.88	4.22	7.06	23.90	17.66	78.04	84.30	68.06	9.78	65.81
54 PFS	PFS Comp	Copper Collector	75	–	85	Natural	3.23	89.68	5.04	8.46	20.86	16.99	74.20	83.00	64.51	6.87	51.25
79	PFS Comp	Copper Collector	75	150	90	Natural	3.27	88.52	5.43	7.72	26.02	19.32	75.85	86.43	64.41	8.48	61.92
80	PFS Comp	Copper Collector	106	150	90	Natural	3.40	81.04	4.67	6.82	23.84	12.90	68.86	80.81	58.42	7.98	48.58
81	PFS Comp	Copper Collector	150	150	90	Natural	3.56	78.51	4.58	6.23	21.23	15.36	70.26	81.60	62.19	7.55	58.33
88	PFS Comp	Copper Collector	75	–	85	Natural	2.63	89.46	6.03	7.20	23.23	17.68	63.92	77.16	48.53	6.13	43.62
89	PFS Comp	Copper Collector	75	10	185	pH ~11 in Rougher	1.85	113.4	9.11	4.61	20.87	16.95	54.08	82.83	22.23	3.85	30.53
90	PFS Comp	Copper Collector	75	10	185	pH ~11 in Cleaner	3.29	81.50	5.36	7.04	24.90	19.79	69.86	85.91	57.95	8.08	60.68
91	PFS Comp	Copper Collector	75	–	190	Natural	3.05	87.21	5.96	8.31	24.57	18.77	71.64	79.14	61.85	7.58	53.58

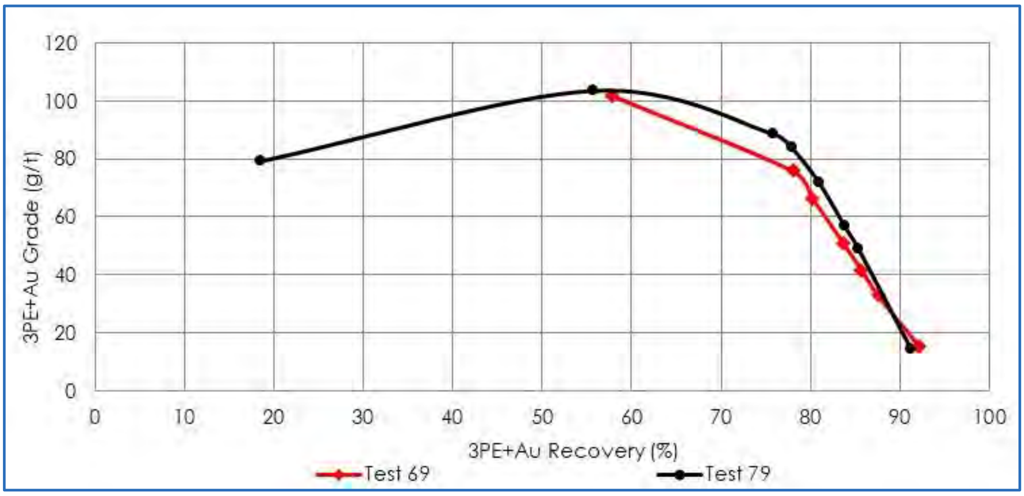
A series of optimisation tests (Tests 79, 88 and 91) were conducted on the remaining PFS Composite using the MF1 copper collector flow sheet. This testing found that increased depressant addition of 150 g/t in the roughers and 90 g/t in the cleaners, with the inclusion of a separate low-grade and scavenger cleaner circuit as presented in Figure 13.13, resulted in superior metallurgical response (Test 79). Test 54 represents the results of the initial copper collector testing conducted as part of the Platreef 2015 PFS using a flow sheet as presented in Figure 13.11 with the exclusion of the conditioning prior to flotation.

Tests 80 and 81 were conducted using the optimal conditions of Test 79, but at a coarser grind of 106 µm and 150 µm respectively. The coarser grind in these tests was found to result in reduced 3PE+Au recovery of 7.0% for the 106 µm grind and 5.6% for the 150 µm grind. The finer grind requires additional milling power, however the additional 3PE+Au recovery results in improved overall project economics as compared to a lower 3PE+Au recovery with reduced mill power consumption. This provided confirmation that the target of 80% passing 75 µm grind, as selected in the Platreef 2015 PFS, was optimal for the MF1 copper collector flow sheet.

Additional testing with lime addition to achieve a pH of ~11 in the roughers (Test 89) and cleaners (Test 90) with increased depressant in the cleaners was conducted. These tests were deemed unsuccessful. Test 89 indicated that operation at pH 11 in the roughers, resulted in reduced sulfur recovery and improved 3PE+Au grade, however this reduced 3PE+Au recovery due to increased losses to the rougher tails. Test 90 indicated that operation at pH 11 in the cleaners, resulted in similar sulfur recovery as achieved in Test 79 but at reduced first pass 3PE+Au and Ni recovery due to increased losses to the cleaner tails.

The results of the MF1 circuit optimisation Test 79, using a targeted copper collector indicated that the optimised flow sheet (Figure 13.12) could achieve a similar performance to Test 69 using, the oxalic acid and thiourea flow sheet as presented in the Platreef 2015 PFS (Figure 13.11). This comparison of performance is presented in Figure 13.13.

Figure 13.13 PGE Grade Recovery Curve for the Optimised MF1 Copper Collector Flow Sheet



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Based on the confirmation of similar metallurgical performance for the MF1 flow sheet using the copper collector reagent suite, compared to the MF1 oxalic acid and thiourea flow sheet, it was decided to base the remainder of the Platreef 2017 FS testing on the MF1 copper collector flow sheet.

The MF1 copper collector flow sheet is preferable to the oxalic acid and thiourea reagent suite for the following reasons:

- The oxalic acid and thiourea reagent suite requires extended conditioning time for optimum effectiveness, resulting in scale-up difficulties and increased operating cost and technical risk. The conditioning time is not required for the copper collector flow sheet.
- The oxalic acid and thiourea reagent suite have not been used in the South African platinum industry and use of the MF1 copper collector flow sheet mitigates the technical risk associated with the supply and use of this novel reagent suite.
- The MF1 circuit, using a reagent suite containing a targeted copper collector, has demonstrated in testwork to have the potential to achieve lower first pass sulfur recovery to final concentrate. This may result in a more marketable concentrate that is likely to attract improved third-party treatment terms.
- The copper collector reagent suite, during laboratory scale testwork, was noted to result in more stable cleaner circuit froth that allows for more stable operation and improved mass pull control.

13.7.2.2 Sulfur Optimisation Testing

Sulfur optimisation testing was conducted on Platreef 2017 FS Composite 1 (FS Composite) (1.2% sulfur) and Composite 2 (1.0% sulfur), to evaluate the effect of sulfur feed grade, cleaner solids concentration and depressant addition on sulfur recovery to final concentrate.

Reducing sulfur (pyrrhotite) recovery to final concentrate would result in reduced concentrate mass pull, and lower concentrate sulfur content, to produce a more marketable concentrate product that is likely to attract improved third-party treatment terms.

A summary of the flow sheets used in this testing is presented in Figure 13.12 (refer to Section 13.7.2.1). The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.12.

Table 13.12 Summary Results of Sulfur Optimisation Testing

Test No.	Sample	Rougher Solids % (w/w)	Scav. Cleaner Solids % (w/w)	Depressant		Mass (%)	Grade					Recovery				
				Rougher (g/t)	Cleaner (g/t)		3PE+ Au (g/t)	Cu (%)	Ni (%)	Fe (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	Fe (%)	S (%)
82	Composite 1	35	As Floated	150	90	3.66	61.04	3.97	6.88	23.68	19.57	68.47	77.58	63.21	9.04	59.70
82 Rpt	Composite 1	35	As Floated	150	90	3.00	71.47	6.03	7.69	22.24	15.14	63.54	83.55	56.06	6.92	37.90
84	Composite 1	35	As Floated	150	170	2.91	84.14	6.44	7.83	20.08	16.53	74.34	85.60	59.34	6.18	42.83
93	Composite 1	30	10	150	179	3.27	74.15	5.40	7.73	28.50	22.44	73.82	81.66	61.08	9.59	57.98
94	Composite 1	30	10	150	170	3.36	72.58	5.13	7.02	24.45	17.35	75.11	84.82	59.93	8.46	47.85
83	Composite 2	35	As Floated	150	90	3.69	79.60	5.28	5.87	19.79	17.24	74.49	84.92	55.61	7.59	58.56
85	Composite 2	35	As Floated	150	170	2.93	97.21	6.29	8.59	24.53	14.84	73.03	89.74	62.72	7.48	42.46

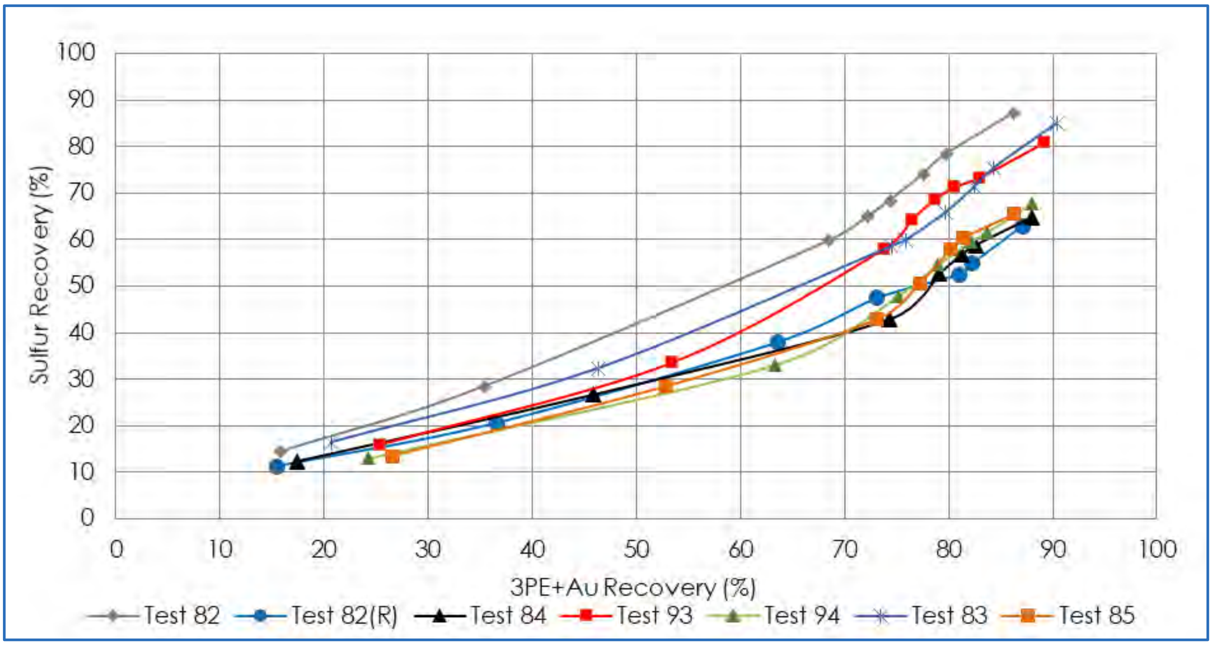
A series of optimisation tests (Tests 82, 84, 92, 93 and 94) were conducted on Composite 1 (1.2% sulfur) using the MF1 copper collector flow sheet. These tests indicated that increasing cleaner depressant addition from 90 g/t (Test 82) to 170 g/t (Test 84) resulted in a 16% reduction in first pass sulfur recovery to final concentrate. A repeat of Test 82, where a lower rougher and cleaner mass pull was targeted by adjusting flotation scrape rates, also resulted in reduced sulfur recovery. Test 82 (Rpt) resulted in significantly lower 3PE+Au recoveries, where the first pass 3PE+Au recovery to final concentrate was 10.8% lower than for Test 84, however this was as a result of increased rougher tails losses for Test 82 which would not be influenced by the cleaner depressant regime.

Bench-scale tests, conducted at a lower solids concentration of 30% (by mass) in the roughers and 10% (by mass) in the scavenger cleaner (Tests 93 and 94) did not result in improved 3PE+Au recovery or sulfur rejection relative to Test 84. Test 93, which considered the addition of a scavenger re-cleaner stage, was found to result in increased sulfur recovery.

Tests 83 and 85 were conducted on Composite 2, with a lower sulfur content of 1.0%. These tests confirmed the finding that increasing cleaner depressant addition from 90 g/t (Test 83) to 170 g/t (Test 85) resulted in a 16% reduction in first pass sulfur recovery to final concentrate.

Sulfur recovery in the sulfur optimisation testing is presented relative to 3PE+Au recovery in Figure 13.14.

Figure 13.14 Sulfur Recovery as Function of PGE Recovery in Sulfur Optimisation Testing



DRA, 2020

As part of the sulfur optimisation testwork, a locked cycle test was conducted on Composite 1 using the reagent regime with higher cleaner depressant addition as per open circuit Test 84. In this locked cycle test, depressant addition was adjusted based on the visual performance of the float, and a cleaner depressant addition rate of 130 g/t was achieved. This test achieved an overall 3PE+Au recovery of 81% and a sulfur recovery of 60%.

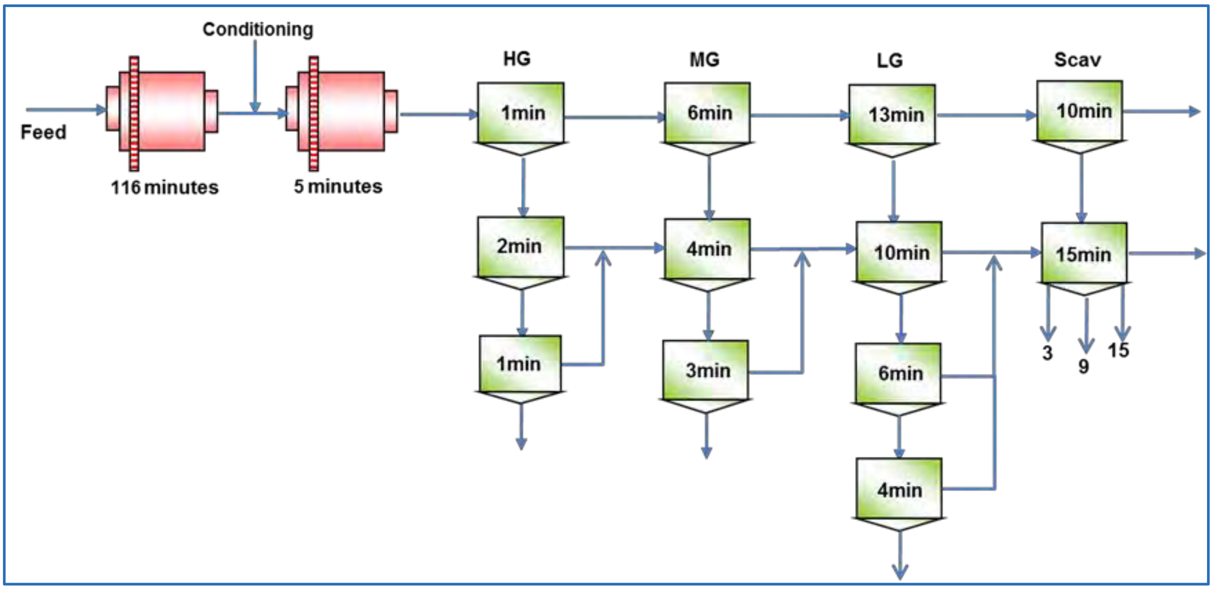
This lower than expected 3PE+Au recovery was attributed to high depressant addition in the cleaners, and further grade optimisation testing was initiated. This testing focused on optimising rougher depressant addition and evaluating the benefit of additional cleaning stages.

13.7.2.3 Concentrate Grade Optimisation

Concentrate grade optimisation testwork was conducted on the Platreef 2017 FS Composite 3 (3.46 g/t 3PE+Au), with the aim of increasing final concentrate 3PE+Au grade. This testwork evaluated the benefit of additional cleaning stages and optimisation of rougher depressant addition. After completion of the sulfur optimisation testing, Composite 1 was depleted. Further grade optimisation testing was thus conducted on Composite 3 with a similar 3PE+Au grade as Composite 1, however Composite 3 was selected to specifically target a lower sulfur grade in feed of 0.8%, that at the time of testing represented the expected sulfur grade in the early years of mining.

A summary of the flow sheets used in this testing is presented in Figure 13.15 and Figure 13.16. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.13.

Figure 13.15 Copper Collector Reagent Flow Sheet with HG Re-Cleaner



DRA, 2017

Figure 13.16 Copper Collector Reagent Flow Sheet with HG Re-Cleaner and MG/LG Column

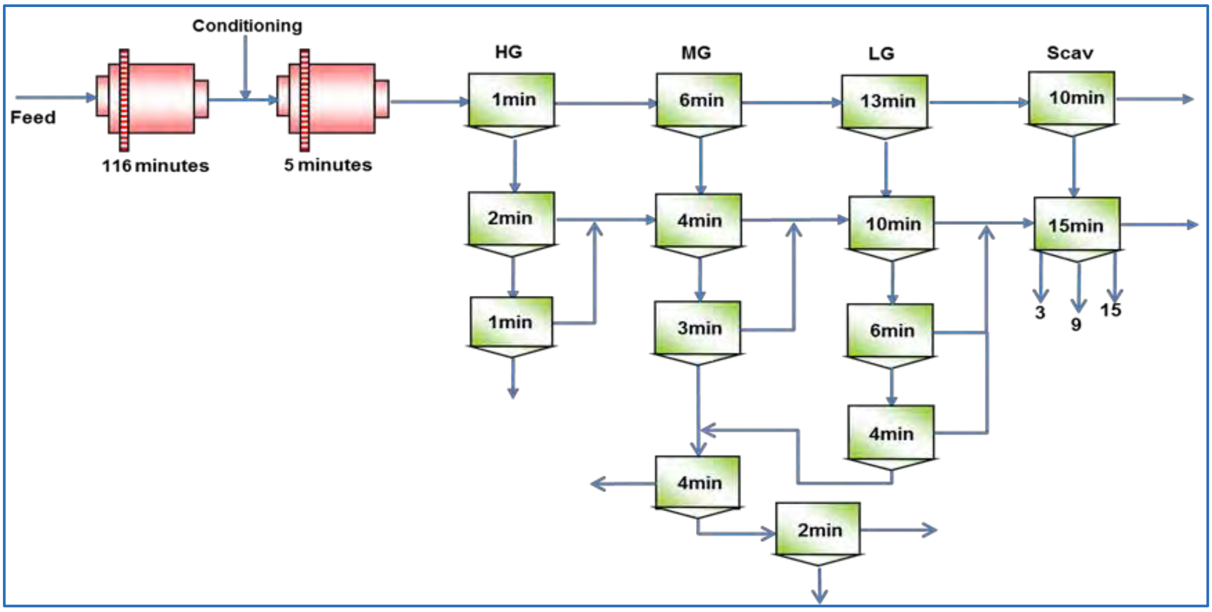


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Table 13.13 Summary Results of Concentrate Grade Optimisation Testing

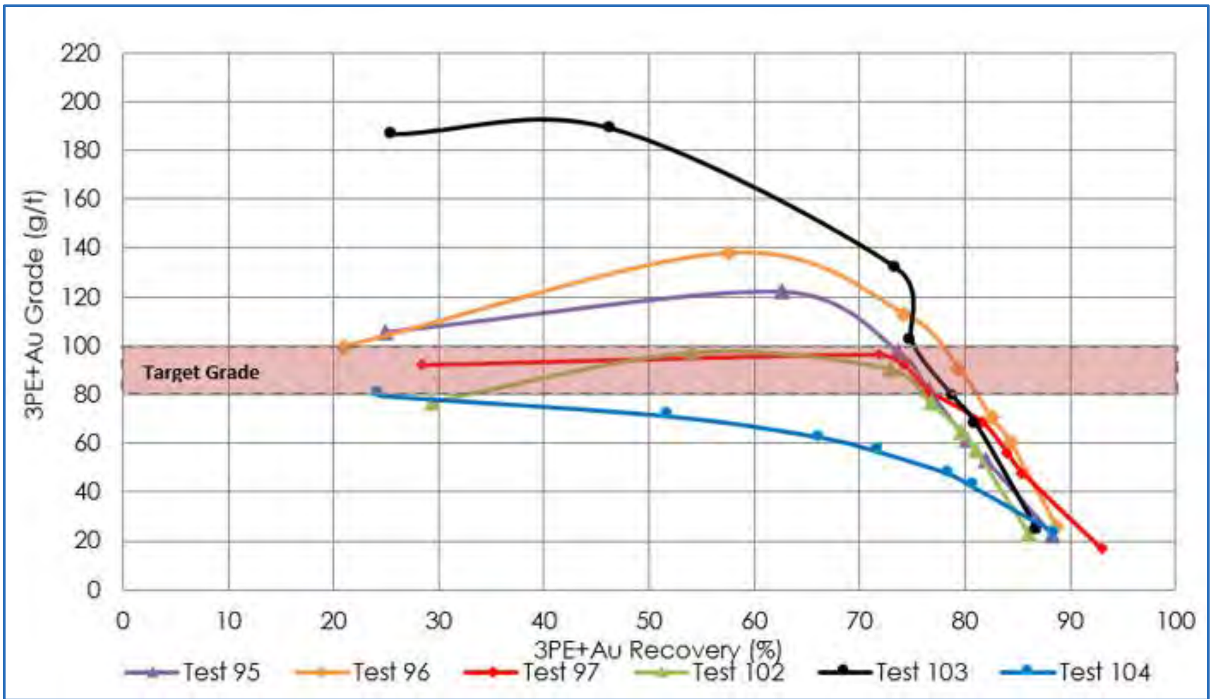
Test No.	Sample	Depressant			Mass (%)	Grade					Recovery				
		Rougher (g/t)	Cleaner (g/t)	Type		3PE+Au (g/t)	Cu (%)	Ni (%)	Fe (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	Fe (%)	S (%)
95	Composite 3	196	59	CMC	2.42	97.50	6.99	9.26	24.48	19.49	73.70	85.25	63.22	6.03	55.72
96	Composite 3	150	74	CMC	2.29	113.31	7.04	9.43	23.62	20.44	74.25	84.33	60.67	5.65	54.72
97	Composite 3	146	59	CMC	3.15	81.30	4.59	6.74	20.25	15.86	76.66	83.00	64.50	6.46	60.30
102	Composite 3	150	74	CMC	2.79	90.38	5.70	8.01	21.15	16.70	72.86	87.40	59.56	6.02	54.22
103	Composite 3	150	89	CMC	1.93	131.45	7.52	10.88	24.59	20.04	73.48	85.31	55.78	4.90	47.26
104	Composite 3	150	84	HC	3.88	62.45	4.07	4.12	13.95	8.65	66.37	86.08	46.72	5.39	41.93

This testing indicated that final concentrate grade could be improved with the addition of a high-grade (HG) concentrate re-cleaner stage (Tests 95 and 96), compared to testing with a single HG cleaner stage (Test 102). In Test 97, dilute two-stage cleaning of the combined medium-grade (MG) and low-grade (LG) concentrate was conducted in order to provide an indication of the expected performance of a column cell. This dilute two-stage cleaning, was found to allow for an increase in the final concentrate grade achieved in the combined MG/LG circuit. A column cell was thus included in the Platreef 2017 FS flow sheet, all further testing was conducted without the addition of this dilute two-stage cleaning, column simulation. The bench-scale column cell simulation is regarded as indicative only, and the true benefit associated with the inclusion of the column cell can only be accurately determined from pilot-scale testing.

Test 104, using a hemicellulose (HC) depressant, achieved lower PGE grades and recovery when compared to the results obtained with a carboxymethyl cellulose (CMC) depressant in the other tests.

Test 103, which considered the inclusion of a HG concentrate re-cleaner, CMC depressant dosage of 240 g/t and a target final mass pull of <2.0%, resulted in the optimal PGE upgrade profile, however recoveries were lower than for Test 96, within the target grade range. This is presented in Figure 13.17.

Figure 13.17 Concentrate Grade Optimisation Test Results



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13.7.2.4 Confirmation of Optimised Conditions for a High-Grade Sample

Further optimisation testwork was conducted on a high-grade composite (5.39 g/t 3PE+Au), which represented at the time of testing, the expected PGE feed grade during the first five years of production. The aim of this phase of testing was to evaluate the effect of solids concentration, depressant addition and to evaluate any further opportunity for concentrate recovery and grade improvement.

The effect of rougher feed solids concertation had previously been evaluated on Composite 1, where no improvement in 3PE+Au recovery or concentrate grade was demonstrated, however this was possibly due to the increased cleaner depressant addition in this testing. Mintek also indicated that at the time of the initial testing, control of the rougher operating conditions in a 5 L batch cell, at the lower solids concentration, had not been optimised.

A summary of the flow sheet used in this testing is presented in Figure 13.18. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.14.

Figure 13.18 Flow Sheet to Confirm Optimal Conditions for a High-Grade Sample

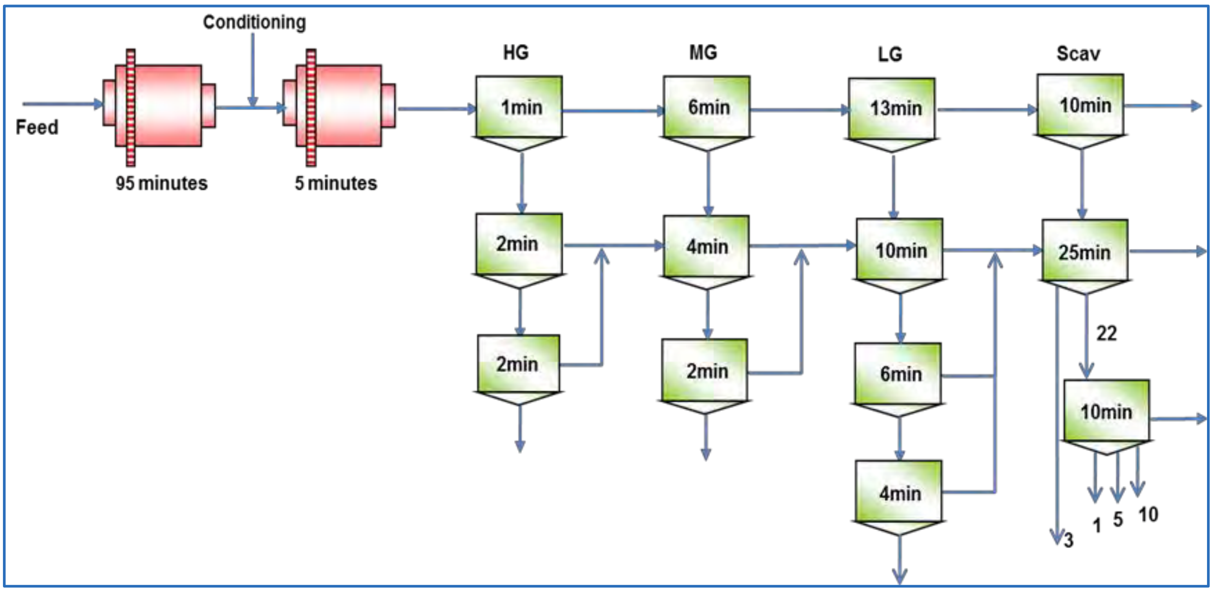

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Table 13.14 Summary Results for Optimisation Testing on a High-Grade Sample

Test No.	Sample	Rougher Solids % (w/w)	Depressant		Scavenger Cleaner Configuration	Mass (%)	Grade					Recovery				
			Rougher (g/t)	Cleaner (g/t)			3PE+Au (g/t)	Cu (%)	Ni (%)	Fe (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	Fe (%)	S (%)
106	Composite 5	35	176	106	15min Cleaner	3.11	120.2	5.91	7.82	28.72	20.24	73.43	85.21	57.10	8.79	65.64
107	Composite 5	35	150	190	15min Cleaner	3.15	123.6	5.40	8.03	29.81	20.00	74.01	81.98	59.52	8.99	66.65
111	Composite 5	28	150	90	15min Cleaner	3.43	111.2	4.83	8.38	26.74	20.07	75.90	80.62	61.93	9.46	70.73
112	Composite 5	28	250	90	15min Cleaner	3.52	113.6	–	–	–	17.99	77.11	–	–	–	65.73
113	Composite 5	28	250	90	25min Cleaner	3.55	112.2	5.07	7.39	26.37	18.65	80.16	85.15	59.53	9.30	68.26
114	Composite 5	28	250	96	25min Cleaner 10min Re-Cleaner	3.68	121.1	4.86	7.30	25.16	18.60	79.52	85.21	62.16	8.81	68.20

*There was insufficient high-grade re-cleaner concentrate mass to allow for Cu, Ni and Fe assays in Test 112.

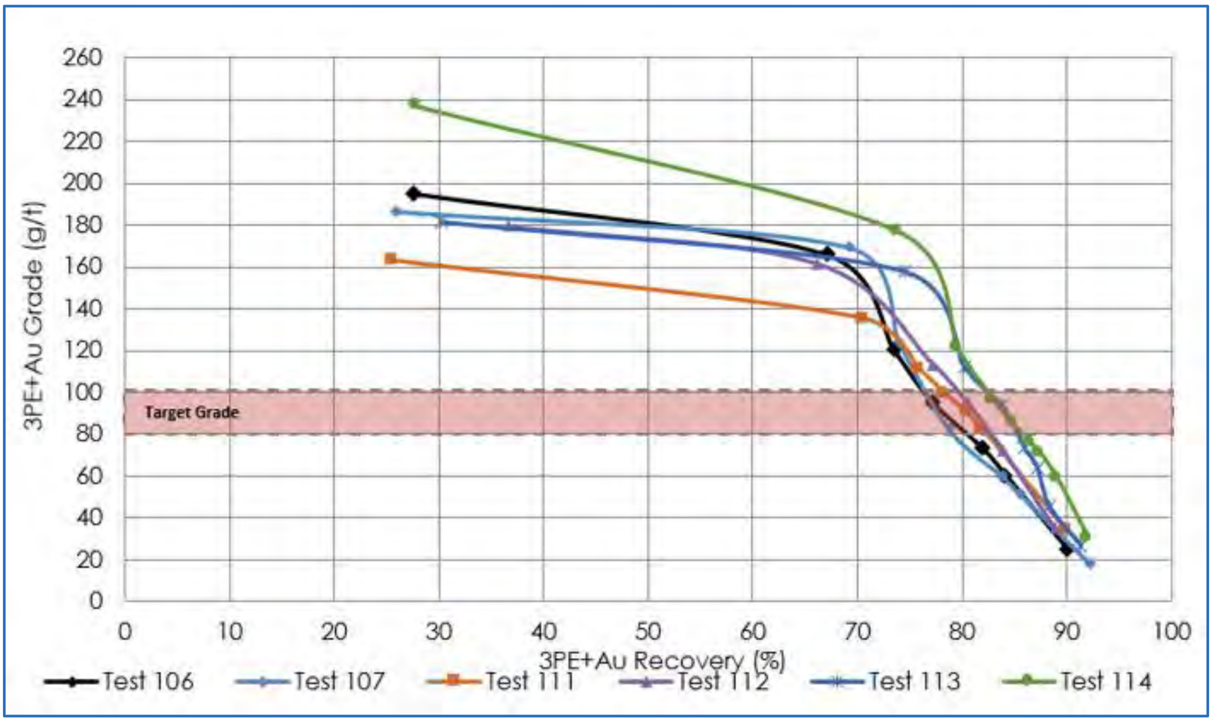
The testwork indicated that final concentrate PGE grades of 110–120 g/t could be achieved at a PGE head grade of 5.4 g/t.

Test 111 conducted at 28% solids (by mass) showed evidence of minor improvement in the grade recovery profile within the target operating range as compared to Test 106 with a similar depressant addition regime but conducted at 35% solids (by mass) as presented in Figure 13.19.

Test 113 was conducted with a 25 minute residence time in the scavenger cleaner, this test indicated that the first concentrate produced after three minutes in the scavenger cleaner was of comparable grade (~20 g/t) to the final low grade concentrate (~23 g/t). It was subsequently decided that this concentrate should be routed to the final low grade re-cleaner feed in locked cycle testing.

A further test (Test 114) was then conducted to evaluate if the scavenger concentrate produced after the first three minutes could be upgraded to the low-grade concentrate grade with the inclusion of a re-cleaner stage. This proved unsuccessful, and a grade of ~11 g/t was achieved.

Figure 13.19 Evaluation of the Effect of Rougher Feed Solids Concentration



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The test conditions and flow sheet as employed in Test 113 was selected for the open circuit variability testing and further locked cycle testwork.

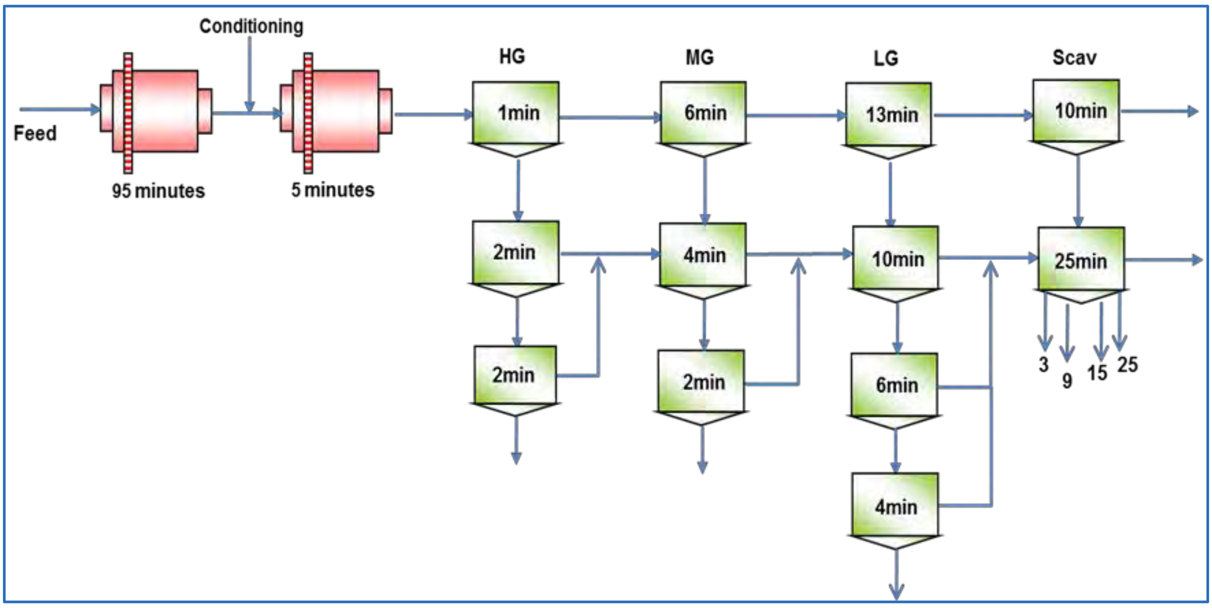
13.8 Flotation Open Circuit Variability Testwork

As part of the Platreef 2017 FS testwork programme, an extensive bench-scale variability test programme was conducted. As part of this programme, 80 open circuit tests were conducted on samples representing T1, T2U, T2L, CZ and various ore blends as follows:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example, T1 from drill core hole UMT104D1.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example, a blend of T1, T2U, T2L and CZ from drill core hole UMT166D1.
- Blend composites representing blends of geometallurgical units from various drill core samples. For example, a blend of 14% T1, 51% T2U, 30% T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

The open circuit variability flow sheet is presented in Figure 13.20.

Figure 13.20 Optimised Platreef 2017 FS Flow Sheet for Open Circuit Variability Testing



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13.8.1 Summary of Open Circuit Flotation Variability Results

13.8.1.1 3PE+Au Recovery and Grade Relationships

Open circuit flotation test data from the 2017 FS was used to derive an estimate for the expected locked cycle test recovery and concentrate grade as follows:

- Locked cycle test mass pull was, on average observed to be approximately 20% higher than the mass pull achieved in open circuit testing.

- The expected 3PE+Au and sulfur recovery was based on the recovery to final concentrate and the average of the recovery to scavenger cleaner concentrate 2 and 3.
- The expected copper and nickel recovery were based on the reported recovery to final concentrate and the recovery to scavenger cleaner concentrate 4.
- The expected final concentrate grades where then calculated based on the test mass pull and recovery data.

This method was derived based on comparison of open circuit and locked cycle mass pull and recovery data for testing conducted on Platreef samples. This method provides a reasonable approximation of the achievable recovery and concentrate grade based on open circuit test results; however, it does not provide a definitive representation of locked cycle recovery. This method as applied to all variability test data, is considered adequate for the purpose of comparing the open circuit test results.

The 3PE+Au head grade-recovery curves derived from the variability testing conducted on domain point samples and blend composites are presented in Figure 13.21 and Figure 13.22.

Figure 13.21 Variability Testing Head Grade Recovery Relationships for Domain Point Samples

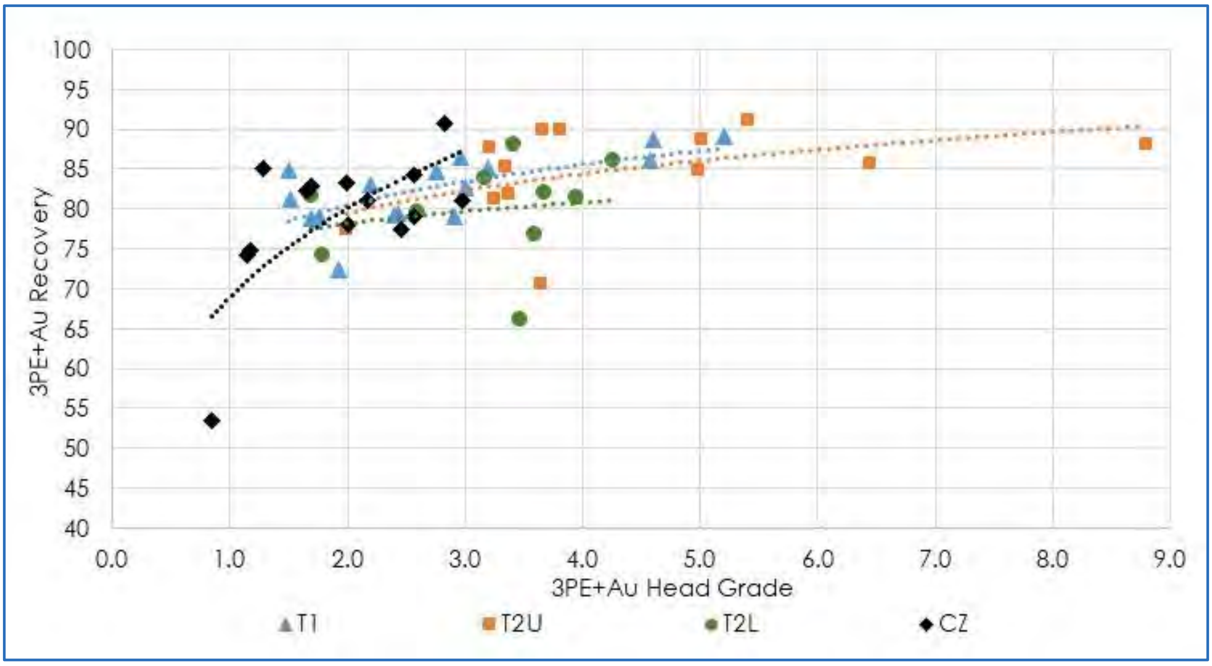

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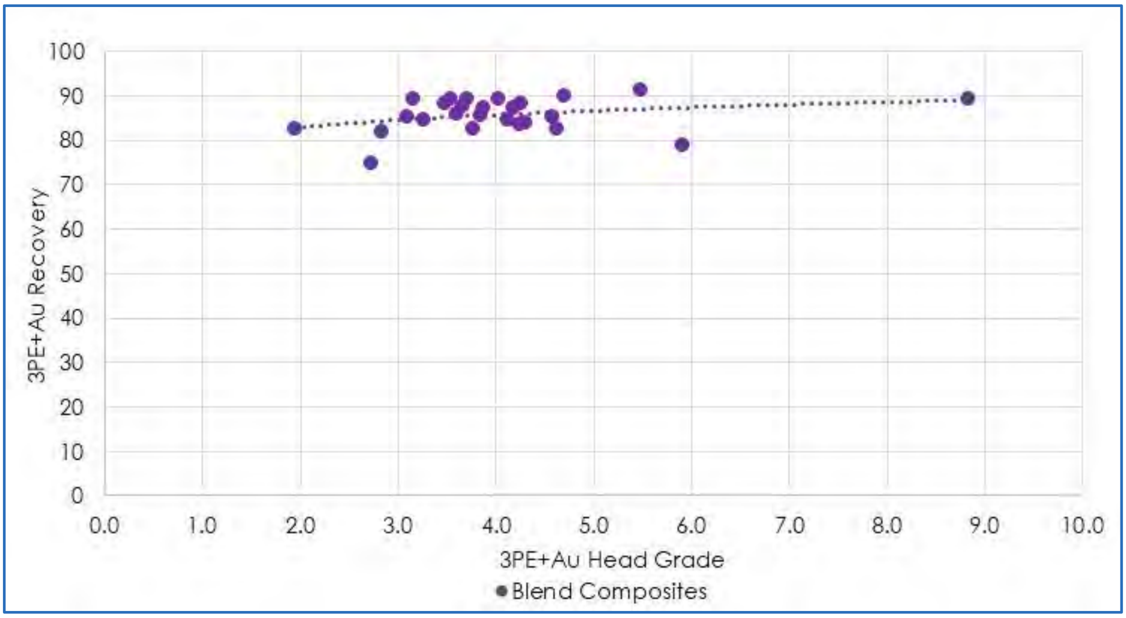
Figure 13.22 Variability Testing Head Grade - Concentrate Grade Relationships for Blend Composites



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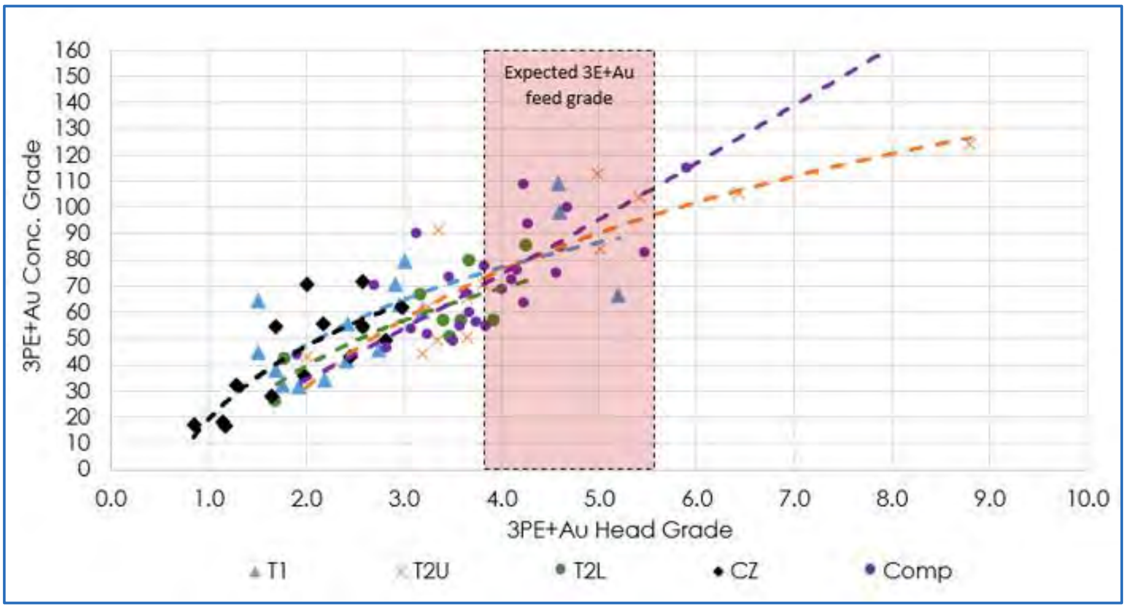
There was a positive correlation between head grade and recovery with an increase in 3PE+Au recovery as 3PE+Au head grade increased for all ore types tested. The final concentrate 3PE+Au concentrate grade also increased with increasing head grade as presented in Figure 13.23.

Figure 13.23 Variability Testwork Concentrate Grade as a Function of Head Grade



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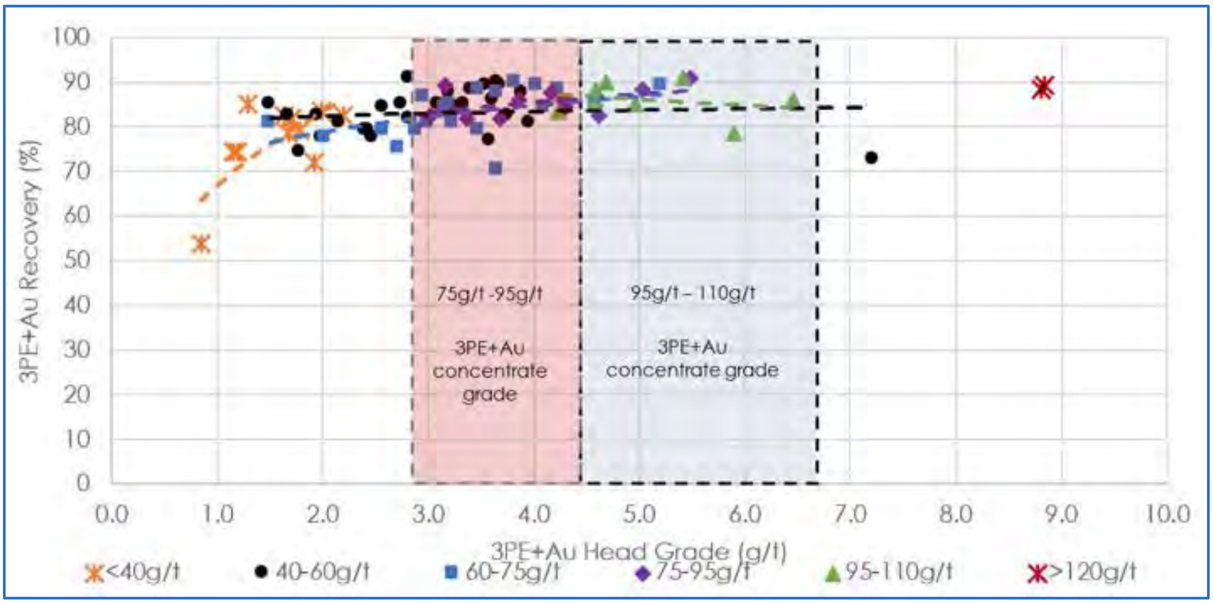
Further to this, 3PE+Au recovery was plotted relative to head grade and grouped according to the final concentrate grade estimate as presented in Figure 13.24. The test data showed only minor variation between the performance for the individual domain point samples and the blend composites, rather the recovery and achievable concentrate grade was primarily dependant on the 3PE+Au head grade.

The achievable concentrate grade was shown to have a dependence on 3PE+Au head grade for all ore types tested. Based on the variability testwork, the minimum 3PE+Au concentrate target grade of 85 g/t can be achieved for head grades in the range 3.2–4.6 g/t. 3PE+Au concentrate grades in excess of 100 g/t can be achieved for head grades in the range 4.6–8.8 g/t.

Figure 13.24 Variability Testwork Recovery as a Function of Head Grade

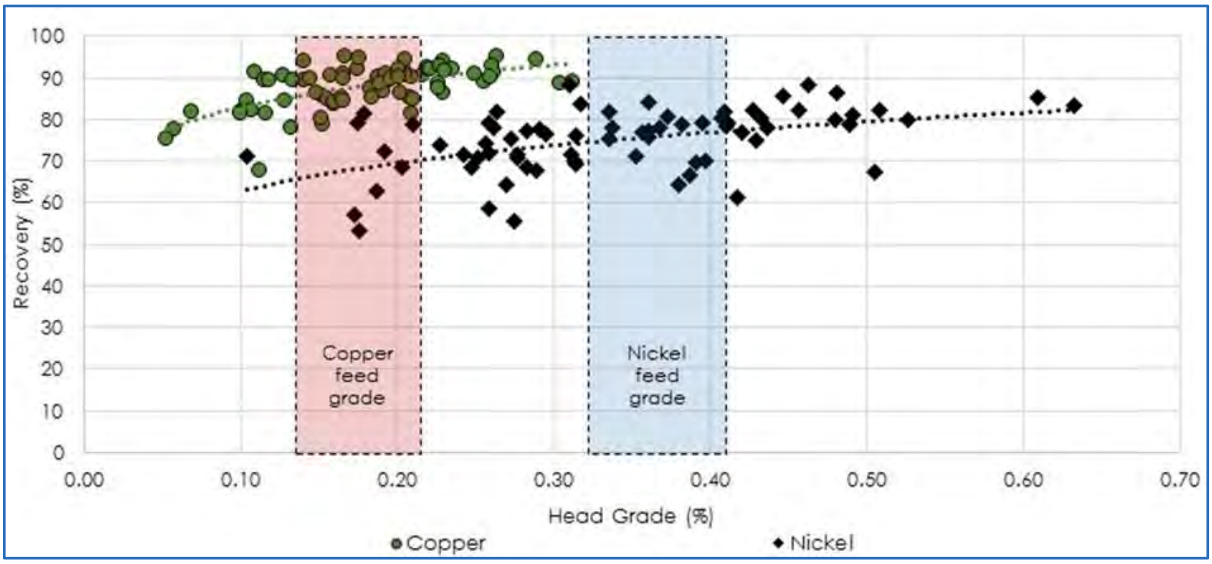


DRA, 2017

13.8.1.2 Copper and Nickel Recovery and Grade Relationships

The nickel and copper head grade-recovery curves derived from the variability testwork conducted on domain point samples and blend composites are presented in Figure 13.25.

Figure 13.25 Variability Testwork Recovery as a Function of Head Grade

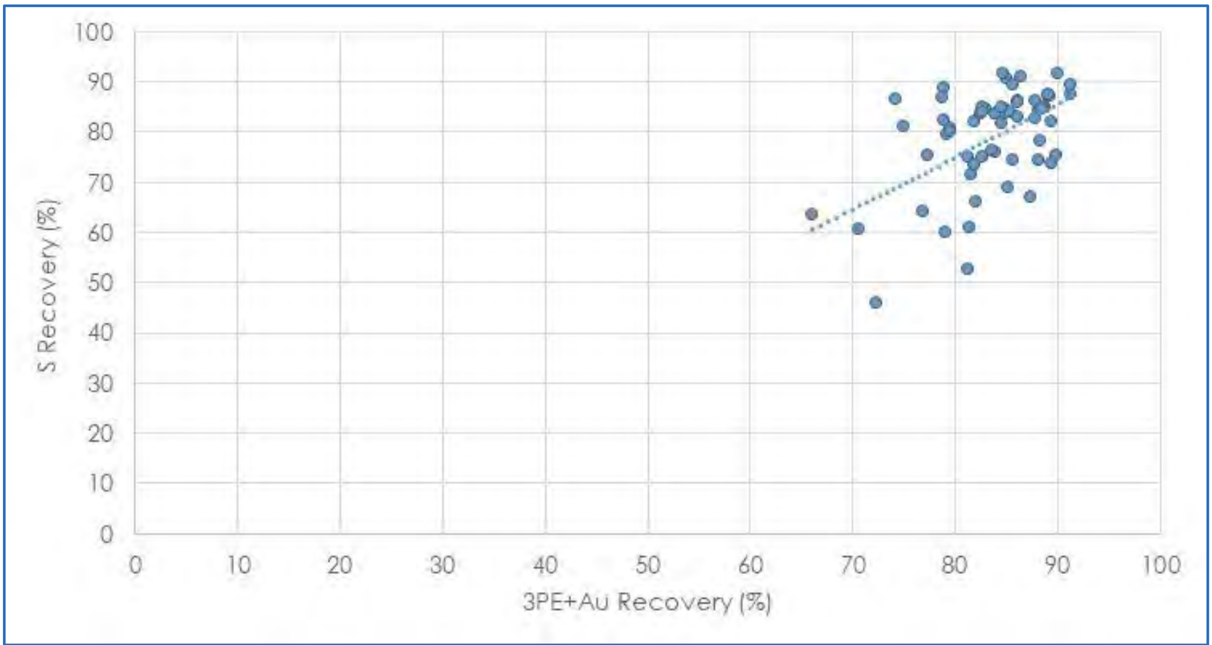


DRA, 2017

13.8.1.3 Sulfur Recovery

A plot of the sulfur recovery as a function of 3PE+Au recovery (Figure 13.26) for all test data, indicated increased sulfur recovery as 3PE+Au recovery increased, albeit with significant scatter in the data.

Figure 13.26 Variability Testwork Sulfur Recovery as a Function of 3PE+Au Recovery



DRA, 2017

13.9 Platreef 2022 FS Open Circuit Flotation Testwork

During the Phase 8 testwork at SGS, bench scale flotation testwork was conducted to characterize flotation performance of the Platreef 2022 FS composite using a flowsheet and reagent regime for two potential MF2 toll processing options (Toll option #1 and #2) as compared to the baseline MF1 flowsheet used in the 2017 FS.

The testwork which was aimed at evaluating the toll processing options, was primarily focused on evaluating a number of potential configurations for Toll option #1 as this was considered to reflect the preferred processing option.

Further open circuit flotation test work was conducted as part of Phase 9 at Mintek in 2021. The aim of this test work was to:

- Further characterise the flotation response of the Platreef 2022 FS composite samples;
- Evaluate the potential for Jameson cell technology in the cleaner circuit;
- Determine the effect on flotation response when using site water;

- Conduct preliminary pilot scale test work which was primarily aimed at producing bulk concentrate samples for settling, filtration and Kell hydrometallurgical refining test work. Additionally, the intention was to derive process design information to supplement the design data as derived from bench scale flotation test work.

The Platreef 2022 test work which was conducted at Mintek included bench-scale flotation test work at Mintek, however, all samples were sent to SGS South Africa for analysis

13.9.1 Platreef 2022 FS Flotation Testwork Samples

The head grades of the Platreef 2022 FS composite samples are presented in Table 13.15.

Table 13.15 Platreef 2022 FS Samples Measured Head Assays

Test ID	Grade									
	3PE+ Au	Pt	Pd	Rh	Au	Cu	Ni	Total S	MgO	Fe
	(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)
2020 SGS PDP Composite	4.89	2.44	2.00	0.14	0.31	0.18	0.36	0.96	23.4	9.51
2021 Mintek HG Core Composite	4.70	2.10	2.16	0.11	0.33	0.20	0.40	1.05	24.4	10.1
2021 Mintek LoM Core Composite	3.67	1.46	1.82	0.13	0.26	0.18	0.36	0.61	23.5	12.9
2021 Mintek Shaft Intercept Sample #1	2.90	1.57	1.07	0.07	0.19	0.11	0.24	1.11	24.5	13.9
2021 Mintek Shaft Intercept Sample #2	3.80[1]	1.82	1.76	-	0.22	0.20	0.41	-	13.1	8.45

Note1: PGE assay data for Shaft Intercept Sample #2 reflects 2E+Au assay results

A comparison between the Platreef 2022 FS samples tested at SGS and Mintek relative to the 2017 FS variability blend composites is presented in Table 13.16

Table 13.16 Summary of the Platreef 2022 FS Sample Composition Relative to FS Variability Composites

Test ID	Head Grade			Blend Ratio			
	3PE+Au (g/t)	Cu (%)	Ni (%)	T1 (%)	T2U (%)	T2L (%)	CZ (%)
2017 Variability Blend Composites	2.5 – 5.7	0.16 – 0.20	0.30 - 0.46	10 - 22	47 - 52	26 - 33	5
2020 SGS PDP Composite (Phase 8)	4.89	0.18	0.36	22	47	26	5
2021 Mintek HG Core Composite	4.70	0.20	0.40	17	39	35	9
2021 Mintek LoM Core Composite	3.67	0.18	0.36	24	40	29	8
2021 Mintek Shaft Intercept Sample #1	2.90	0.11	0.24	No Data			
2021 Mintek Shaft Intercept Sample #2	3.80[1]	0.20	0.41	No Data			

Note1: PGE assay data for Shaft Intercept Sample #2 reflects 2E+Au assay results.

The 3PE+Au grade of the 2020 SGS PDP composite (4.9g/t), Mintek HG core composite (4.7g/t) and shaft intercept sample #2 (3.8g/t) are aligned to the expected grade range of 3.8-6.1 g/t averaging 4.4 g/t in the Platreef 2022 FS mine plan.

All the Platreef 2022 FS samples have a 3PE+Au grade lower than the 5.0 – 6.1 g/t that is expected in the first five years of mining.

The composite samples tested at Mintek in 2021 as part of the Platreef 2022 FS contained higher ratios of CZ (8-9%) as compared to the 2017 FS variability blend composites (~5%).

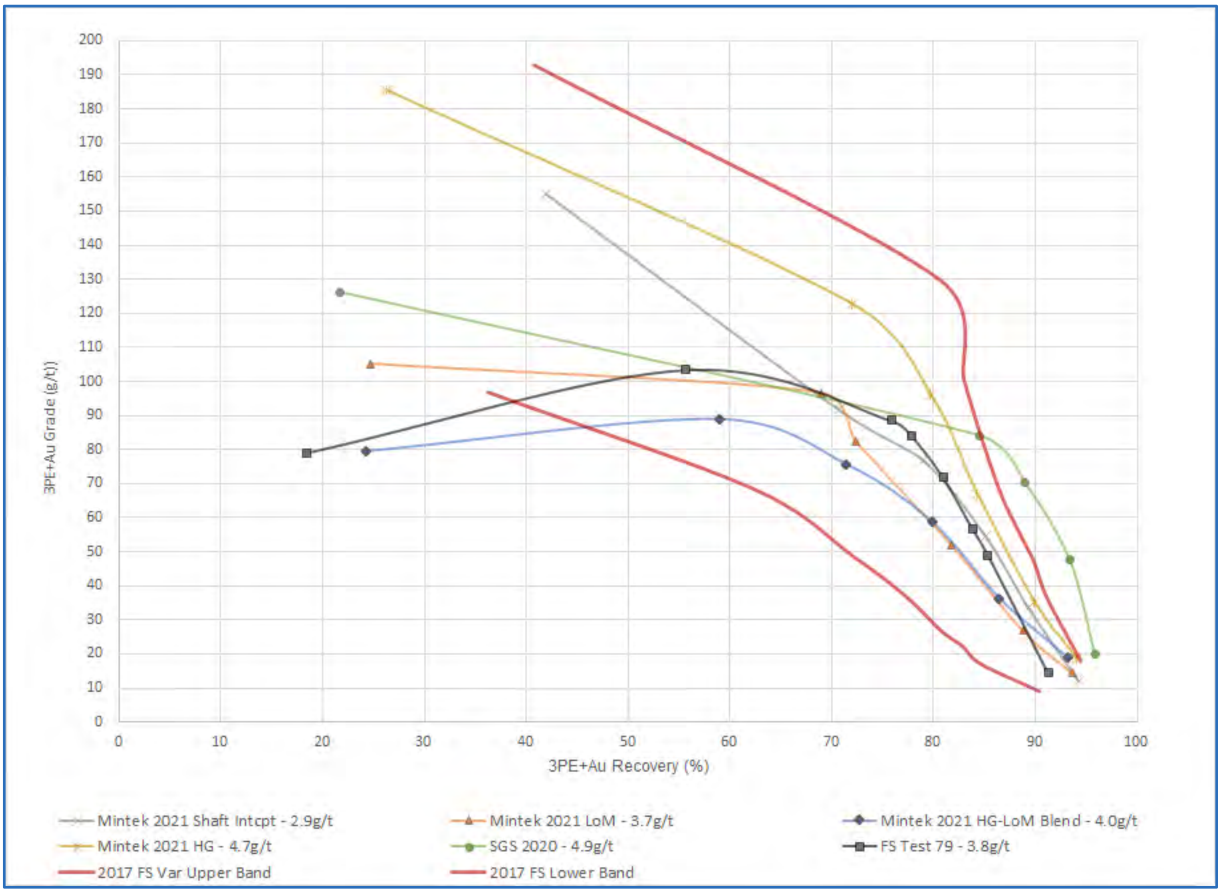
The core composites that were used for the Platreef 2022 FS testwork campaigns reflect un-crushed ½ and ¼ drill core remainders from the 2017 FS drilling campaign.

13.9.2 Platreef 2022 FS Open Circuit Flotation Testwork Results

The Platreef 2022 FS flotation testwork at Mintek and SGS included open circuit baseline tests using the open circuit test conditions as derived during the 2017 FS (Refer Figure 13 20). A comparison of the optimal 3PE+Au open circuit upgrade profile for each of the Platreef 2022 FS samples using the 2017 baseline conditions is presented in Figure 13.27.

Figure 13.27 also shows the minimum and maximum upgrade profiles as achieved in the previous 2017 FS open circuit variability testwork for sample feed grades of less than 5 g/t (3PE+Au) for reference.

Figure 13.27 Platreef 2022 FS Testwork Optimal Open Circuit 3PE+Au Upgrade Profiles



DRA, 2022

The 3PE+Au upgrade profiles for the optimal tests from the Platreef 2022 FS campaigns at both SGS and Mintek fall within the expected range observed in the historical 2017 FS test work at Mintek. A similar trend was observed for Cu and Ni.

The open circuit baseline test at SGS was conducted using a proxy copper collector reagent mixed according to the reagent vendor specifications. This was due to an inability to source the C7160 copper collector due to international COVID-19 travel restrictions. This test achieved an optimal open circuit 2E+Au recovery of 85% at a concentrate grade of 84 g/t.

The optimal Mintek results for the LoM core composite (3.7 g/t 3PE+Au) and HG-LoM core composite blend (4.0 g/t 3PE +Au) were noted as being on the lower end of the range achieving an open circuit 3PE+Au recovery of 70% to 72% at 80 g/t grade as compared to approximately 80% 3PE+Au recovery at 96 g/t and 77 g/t for the HG (4.7 g/t 3PE+Au) and shaft intercept #1 (2.9 g/t 3PE+Au) samples respectively.

In addition to the baseline open circuit testwork aligned to the 2017 FS flowsheet, Bench-scale Jameson cell scouting test work was conducted at SGS in 2020 and Mintek in 2021. These tests at, both at laboratories, indicated the potential for a significant reduction in first pass cleaner circuit 3PE+Au recovery relative to the baseline open circuit cleaner tests. A similar result was observed for Cu and Ni. The equipment vendor has subsequently indicated that the test work was conducted using an outdated procedure which may have impacted on the results. Repeat test work using the updated vendor procedure is scheduled to take place in the first quarter of 2022.

Comparative test work on the 2021 Mintek LoM and HG core composites using filtered site water representative of the water quality expected from the Masodi grey water system achieved a similar 3PE+Au upgrade profile as the baseline tests on these samples. A similar trend was observed for Cu and Ni.

During the Mintek Platreef 2022 FS testwork campaign notable variances were observed for tests performed in duplicate and four tests out of the fourteen conducted achieved poor test accountability and were rejected. The reasons for the poor test accountability and variance in duplicate testing are not fully understood however it is possible that sample variability, assay variances, procedural variances and non-optimized mass pull may have contributed.

13.10 Flotation Locked Cycle Testwork

A total of seventeen (17) locked cycle tests were undertaken during the 2017 FS and three (3) additional locked cycle test was conducted at SGS in 2020 as part of the Platreef 2022 FS test work programme as follows:

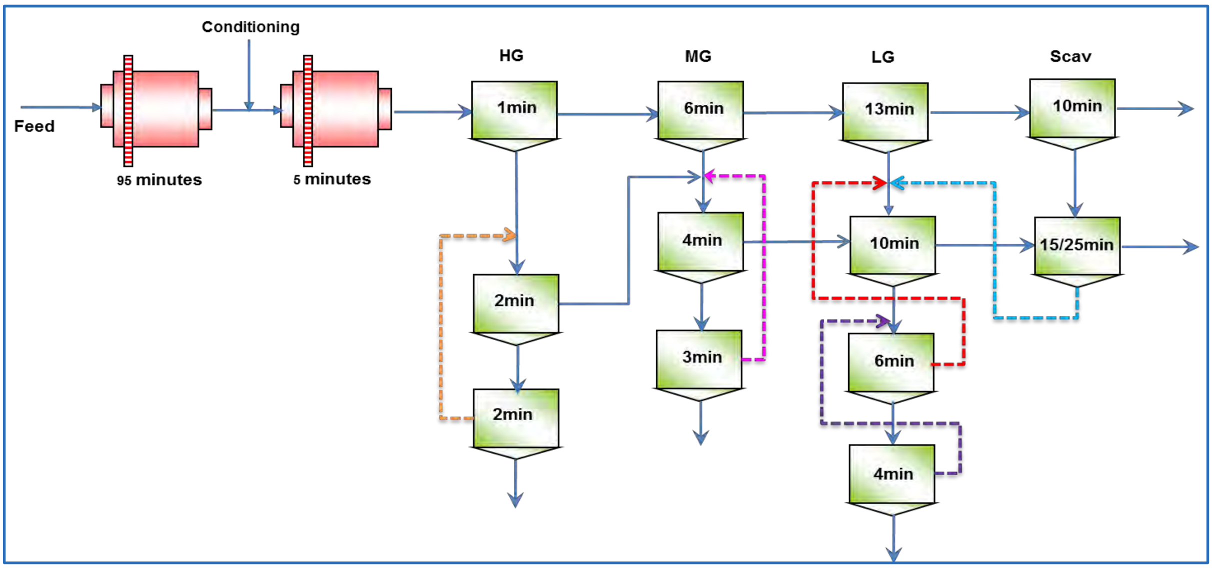
- Mintek 2017 FS: Five (5) locked cycle test work was conducted on four development composites representing the PFS mine blend ratio. Twelve (12) locked cycle tests were conducted on seven variability composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore blends, with the focus placed on testing an ore blend representing the first 5 years of mining.
- SGS 2020: Three (3) locked cycle tests were conducted on the Platreef 2022 FS Phase 8 blend composite representing a blend containing 22% T1, 47%T2U, 26%T2L and 5% CZ. Only one of these tests reflects the Platreef flowsheet while the other two tests reflect testwork which was aimed at assessing the performance of potential toll treatment options. The toll treatment options have been discarded based on the outcome of the 2020 PEA.

13.10.1 Summary of Locked Cycle Test Flow Sheets

A summary of the locked cycle test flow sheets used during the initial development and optimisation testwork phase are presented in the optimised Platreef 2017 FS locked cycle test flow sheet is presented in Figure 13.28

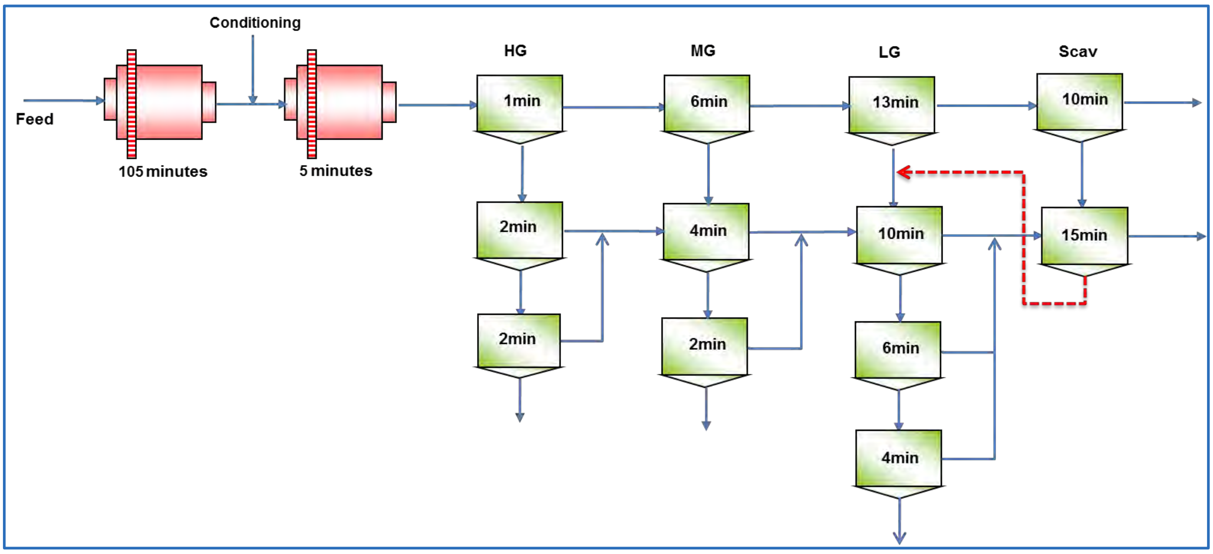
Additionally, the toll processing flowsheets as assessed during the SGS PDP test work are presented in Figure 13.28, Figure 13.29, Figure 13.30, Figure 13.31 and Figure 13.32.

Figure 13.28 Development Testing Locked Cycle Test Flow Sheet 1 (Flow Sheet 1)



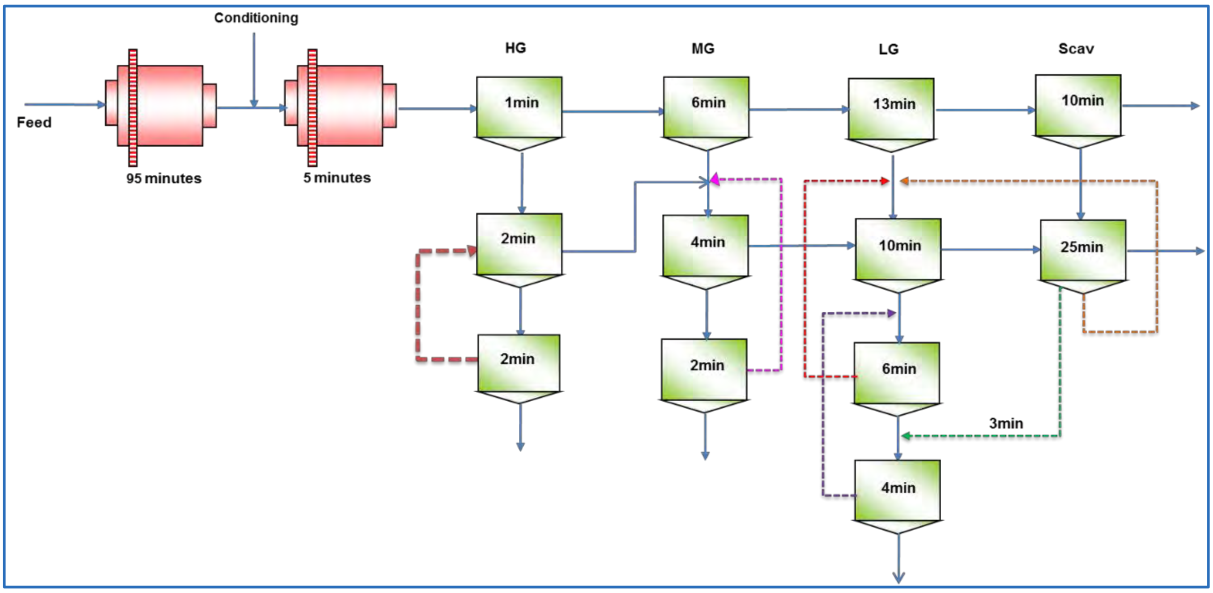
DRA, 2017

Figure 13.29 Development Testing Locked Cycle Test Flow Sheet 2 (Flow Sheet 2)



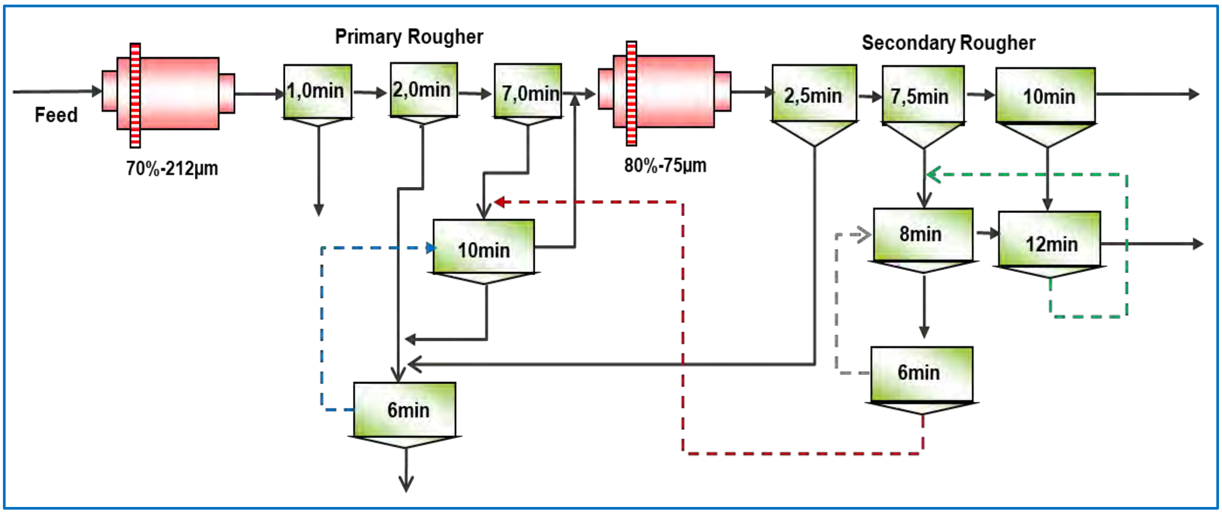
DRA, 2017

Figure 13.30 Optimised Platreef 2017 FS Locked Cycle Test Flow Sheet (Flow Sheet 3)



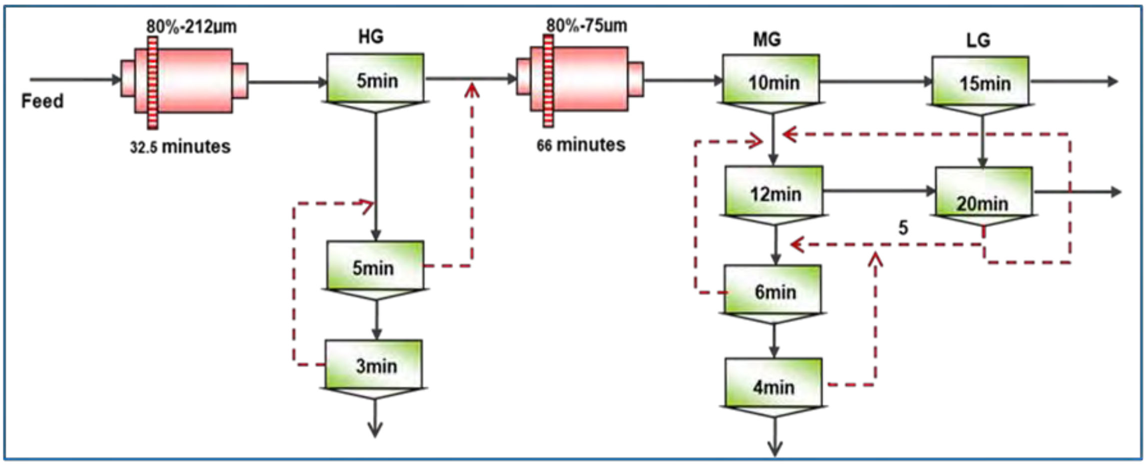
DRA, 2017

Figure 13.31 SGS PDP Toll Processing Test Flow Sheet A (Flow Sheet 4)



DRA, 2021

Figure 13.32 SGS PDP Toll Processing Test Flow Sheet A (Flow Sheet 5)

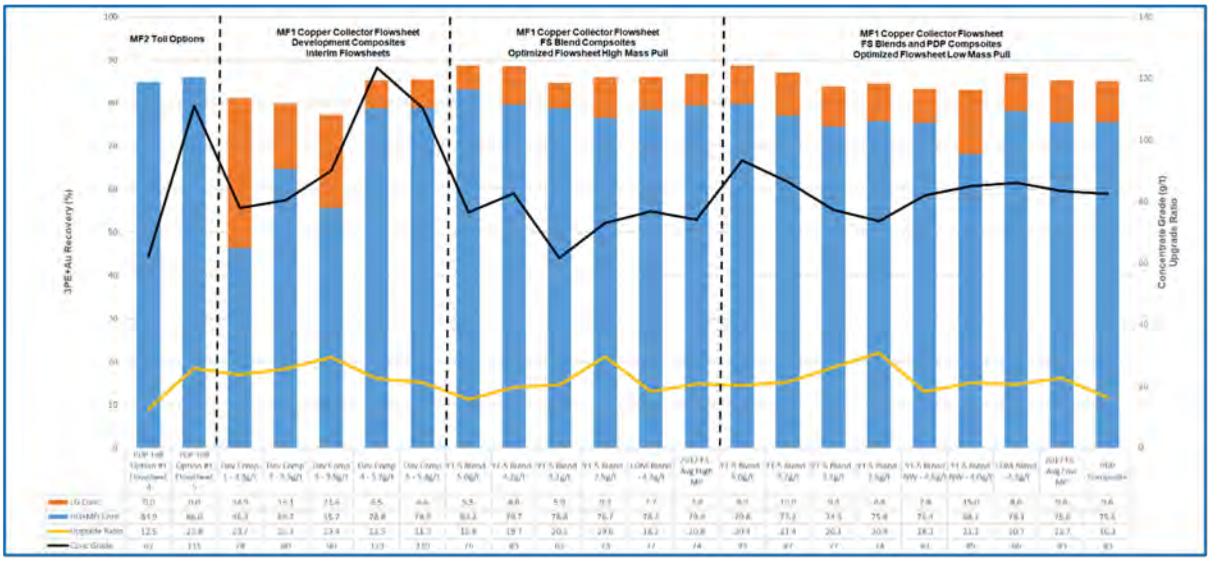


DRA, 2021

13.10.2 Summary of Locked Cycle Test Results

The head grade and domain blends of the locked cycle test samples are presented in Table 13.17. A summary of the locked cycle test results is presented in Figure 13.33, while the grade and recovery achieved for the final combined concentrate (high grade, medium grade and low grade) is presented in Table 13.18.

Figure 13.33 Summary of Locked Cycle Test Results



DRA, 2021

Table 13.17 Locked Cycle Test Measured Sample Head Assays

Test ID	Approximate Ore Blend	Grade									
		3PE+Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
Development Testwork Domain Blends											
Composite 1	12%T1, 32%T2U, 50%T2L, 6%CZ	3.25	1.34	1.62	0.06	0.23	0.20	0.39	1.28	23.02	9.15
Composite 3		3.48	1.53	1.61	0.10	0.24	0.18	0.36	0.81	27.56	10.15
Composite 4		5.71	2.49	2.70	0.19	0.33	0.21	0.46	0.97	27.06	10.40
Composite 5		5.39	2.24	2.70	0.15	0.30	0.21	0.46	0.97	25.07	9.77
2017 FS Domain Blends											
Year 1–5 Blend 1	14%T1, 51%T2U, 30%T2L, 5%CZ	5.05	2.36	2.17	0.13	0.39	0.22	0.40	1.02	24.24	9.53
Year 1–5 Blend 2		4.20	1.91	1.90	0.11	0.28	0.20	0.38	1.00	25.01	9.62
Year 1–5 Blend 3		3.21	1.39	1.42	0.10	0.30	0.20	0.37	1.19	25.15	9.72
Year 1–5 Blend 4		2.49	1.05	1.16	0.09	0.20	0.19	0.36	1.09	24.79	9.68
Year 1–5 Blend 5	10%T1, 52%T2U, 33%T2L, 5%CZ	4.56	1.93	2.20	0.15	0.28	0.18	0.34	0.81	24.65	9.31
Year 1–5 Blend 6		3.98	1.69	1.89	0.12	0.28	0.16	0.30	0.80	22.85	9.48
LOM Blend	22%T1, 47%T2U, 26%T2L, 5% CZ	4.27	1.90	1.96	0.15	0.26	0.20	0.43	1.00	24.07	9.68
2022 FS Domain Blends											
SGS PDP Comp	22%T1, 47%T2U, 26%T2L, 5% CZ	4.89	2.44	2.00	0.14	0.31	0.18	0.36	0.96	23.4	9.51

Table 13.18 Summary of Locked Cycle Test Results

Test ID	Flow /Sht	Mass Pull	Depressant		Head Grade	Conc Mass (%)	Concentrate Grade						Recovery					
			Rough (g/t)	Clean (g/t)	3PE+Au (g/t)		3PE+Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (g/t)	Fe (%)	3PE+Au (%)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
Development Locked Cycle Testwork (MF1)																		
Composite 1	1	Low	150	130	3.25	3.4	77.9	5.7	7.8	20.6	9.4	27.8	81.2	91.0	68.8	59.2	1.4	9.9
Composite 3	1	Low	150	90	3.48	3.1	80.4	5.1	8.1	17.1	12.7	22.1	79.8	88.9	70.9	65.2	1.5	7.0
Composite 3	2	Low	150	90	3.48	2.6	90.0	5.6	7.5	19.6	13.1	23.5	77.3	84.8	53.7	67.7	1.3	6.3
Composite 4	1	Low	150	90	5.71	3.8	123.4	4.7	8.0	18.4	11.8	30.6	85.3	90.2	71.0	82.0	1.7	11.2
Composite 5	1	Low	250	90	5.39	4.0	110.3	4.2	7.3	18.9	12.9	25.8	85.5	87.9	71.2	81.1	2.0	10.1
Optimised FS Flow Sheet Locked Cycle Testwork (MF1)																		
Year 1–5 Blend 1	3	Low	250	90	5.05	4.4	93.4	4.6	7.7	18.2	9.9	25.8	88.7	93.5	83.3	82.4	1.9	11.5
Year 1–5 Blend 1	3	High	–	90	5.05	5.6	76.5	3.7	5.6	15.5	15.7	20.3	88.7	92.2	81.9	82.6	3.8	11.5
Year 1–5 Blend 2	3	Low	250	90	4.20	4.1	86.5	4.3	7.1	16.6	15.7	19.7	87.2	89.6	74.1	69.2	2.5	8.2
Year 1–5 Blend 2	3	High	125	90	4.20	4.5	82.7	4.1	7.1	15.8	13.9	22.2	88.6	95.4	80.2	74.7	2.6	10.7
Year 1–5 Blend 3	3	Low	250	90	3.21	3.2	77.3	5.9	8.8	20.6	11.9	25.8	83.8	90.2	73.9	66.2	1.5	8.3
Year 1–5 Blend 3	3	High	125	90	3.21	4.1	61.7	4.4	7.0	19.5	11.4	25.9	84.7	89.8	74.5	79.0	1.9	10.6
Year 1–5 Blend 4	3	Low	250	90	2.49	2.7	73.6	6.1	8.6	21.7	10.0	26.4	84.6	88.5	70.7	70.6	1.1	7.5
Year 1–5 Blend 4	3	High	125	90	2.49	2.9	73.0	5.9	9.0	21.9	8.8	27.5	86.0	93.7	74.1	71.9	1.0	8.1
Year 1–5 Blend 5	3	Low	250	90	4.56	4.6	82.0	3.4	4.8	12.7	18.1	17.5	83.2	87.2	66.2	73.3	3.2	8.7
Year 1–5 Blend 6	3	Low	250	90	3.98	3.9	85.0	3.8	5.7	13.6	14.3	20.4	83.1	86.2	72.3	68.9	2.5	8.3
LOM Blend	3	Low	250	90	4.27	4.2	86.1	4.9	8.4	19.4	11.5	24.8	86.9	93.4	80.6	79.5	2.0	10.6
LOM Blend	3	High	125	90	4.27	4.7	76.8	4.1	6.8	14.4	16.1	20.5	86.1	95.3	78.2	70.4	3.1	9.7
SGS PDP Comp	3	Low	125	90	4.89	5.2	81.9	3.6	5.3	11.8	15.1	18.5	84.9	90.7	71.2	61.6	3.3	3.6
Toll Processing Flow Sheet Locked Cycle Testwork (MF2)																		
SGS PDP Comp	4	High	-	190	4.89	7.1	61.2	2.6	4.0	9.2	18.1	15.8	85.3	96.5	76.3	65.1	5.7	12.3
SGS PDP Comp	5	Low	-	170	4.89	3.4	113	5.0	7.6	15.9	12.2	21.3	85.7	89.8	74.1	60.9	1.9	7.6

13.10.2.1 Locked Cycle Testwork on Development Composites

Locked cycle testwork on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries in the range 77.3–85.5% at a final concentrate grade of approximately 80–120 g/t.

The locked cycle test on development Composite 1 was conducted with a high cleaner depressant addition rate of 130 g/t with the objective of reducing sulfur recovery. In this test using flow sheet 1 (refer to Figure 13.28), it was found that this higher cleaner addition rate negatively impacted on recovery. In this test only 46% 3PE+Au recovery was achieved in the combined high grade and medium grade concentrate with further 35% 3PE+Au recovery via the low-grade cleaner circuit resulting in a total overall recovery of 81%. The large portion of misplaced PGE's in the low-grade cleaner circuit was attributed to the high depressant addition and cleaner depressant addition was reduced to 90 g/t in all further locked cycle tests.

Two locked cycle tests were conducted on development Composite 3. The first test, using flow sheet 1 (refer to Figure 13.28) with a reduction in cleaner depressant dosage achieved 65% 3PE+Au recovery in the combined high grade and medium grade concentrate with further 15% 3PE+Au recovery via the low grade cleaner circuit resulting in a total overall recovery of 80%. This confirmed that lower cleaner depressant addition allowed for maximum 3PE+Au recovery via the high grade and medium grade cleaner circuit. A second test on Composite 3, was conducted using flow sheet 2 (refer to Figure 13.29) with the aim of targeting a lower overall mass pulled higher concentrate grade. This test achieved a lower mass pull of 2.6% as compared to 3.1% for the first test. Concentrate grade improved by 10 g/t, however, overall 3PE+Au recovery was 2.5% lower. Thus, no further locked cycle testing of flow sheet 2, was conducted.

A further two locked cycle tests were conducted on high grade Composites 4 (5.7 g/t 3PE+Au) and Composite 5 (5.4 g/t 3PE+Au) using flow sheet 1 (refer to Figure 13.28) with a cleaner circuit depressant addition rate of 90 g/t. These tests achieved a PGE recovery of approximately 85% at a smelter-grade final concentrate grade of 120 g/t and 110 g/t for Composite 4 and 5 respectively.

13.10.2.2 Locked Cycle Testwork on Platreef 2017 FS Blend Composites

Locked cycle testing was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore blends, with the focus placed on testing of an ore blend representing the first five years of mining. These tests were conducted using flow sheet 3 (refer to Figure 13.30).

For the Platreef 2017 FS blend sample and Year 1 to Year 5 Blends 1–4, two locked cycle tests were conducted. The first targeting a low mass pull and the second targeting a high mass pull. For the Year 1 to Year 5 Blend 5 and 6 a single locked cycle test, targeting a low mass pull was conducted. The locked cycle testing was conducted on ore blend composites representing 3PE+Au head grades in the range 2.5–5.1 g/t.

The low mass pull tests were conducted using the optimised depressant regime as derived in the development testing phase. In the tests targeting a high mass pull, the rougher depressant addition was reduced based on the visual performance of the float. Initially for the high mass pull test on the 5.1 g/t composite no rougher depressant was added, however after reviewing the test results it was decided to increase the rougher depressant to 125 g/t and adjust cleaner scrape rates in the remaining tests targeting a high mass pull.

This locked cycle flotation testwork achieved recoveries in the range 83.1–88.7% at a smelter-grade final concentrate of approximately 60–95 g/t. In, this locked cycle testwork, concentrate mass pull was found to be dependent on the 3PE+Au concentrate grade relative to the 3PE+Au head grade (upgrade ratio). When a low mass pull was targeted, higher concentrate grades were achieved (high upgrade ratio). Conversely when a high mass pull was targeted lower concentrate grades were achieved (low upgrade ratio) as summarised in Table 13.19.

Table 13.19 Summary of Average 2017 FS Locked Cycle Test Results at High and Low Mass Pull

Averaged Results	Head Grade	Mass Pull	UGR	Concentrate Grade			Recovery		
	3PE+Au (g/t)	Mass %	Conc /HG	3PE+Au (g/t)	Cu (%)	Ni (%)	3PE+Au (%)	Cu (%)	Ni (%)
Low MP	3.8	3.7	21	83	4.7	7.6	86.2	91.0	76.5
High MP	3.8	4.4	24	74	4.4	7.1	86.8	93.3	77.8

With the exception of the 5.1 g/t Year 1 to Year 5 blend composite and the 4.3 g/t LOM blend composite, 3PE+Au recovery was found to be lower for tests in which a higher concentrate grade was targeted. This was attributed to the fact that the lower mass pull tests on these two composites achieved near optimal recovery and at an upgrade ratio of approximately 21, which is lower than the target upgrade ratio of approximately 25 for the low mass pull operating conditions. Attempts to improve recovery at higher mass pull for these composites only resulted in grade dilution due to increased MgO recovery via the low-grade cleaner circuit.

In the locked cycle tests on the FS ore blend composites, the high grade and medium grade cleaner circuit was found to produce a high-grade concentrate of approximately 100–130g/t, at a 3PE+Au recovery of 68–83%. The low-grade cleaner circuit produced a low-grade concentrate of 15–35g/t and allowed for additional 3PE+Au recovery of 5–15%. It is thus evident that it is possible to achieve high concentrate grades of 100–130g/t for the Platreef ore, however this would result in a lower 3PE+Au recovery as it would not be possible to achieve optimal 3PE+Au recovery from the low grade cleaner circuit within the concentrate grade constraints.

13.11 Platreef 2022 FS Locked Cycle TestWork

A single locked cycle test on the Platreef 2022 FS Composite at SGS using the optimized 2017 FS flowsheet (flowsheet 3) achieved a 3E+Au recovery of 85% at a smelter-grade final concentrate of approximately 82 g/t. A copper recovery of 91% was achieved, while for nickel a recovery of 71% was achieved. This result is aligned to the performance achieved in the original 2017 FS, albeit on the lower end of the range. As previously noted the 2020 SGS testwork was conducted using a proxy copper collector which may have impacted on performance.

Locked cycle test work to evaluate the expected performance for an MF2 toll processing plant achieved a 3E+Au recovery of 85% at a smelter-grade final concentrate of approximately 61 g/t at a high mass pull of 7.1%. A copper recovery of 97% was achieved, while for nickel a recovery of 76% was achieved. A further locked cycle test using a modified flowsheet to target a lower mass pull of 3.4% achieved a 3E+Au recovery of 86% at a smelter-grade final concentrate of approximately 113 g/t. A copper recovery of 90% was achieved, while for nickel a recovery of 74% was achieved. The grade achieved in this test is higher than the target grade of 80 g/t to 85 g/t and it could be argued that targeting a lower grade would potentially offer the opportunity for recovery improvement.

13.12 Preliminary Mini-Pilot Plant Testwork

During the Platreef 2022 FS testwork campaign, an initial mini-pilot plant commissioning run was conducted at Mintek in June 2021 with a further commissioning run in November 2021. The aim of the mini pilot plant programme was to produce bulk concentrate samples for concentrate Kell hydrometallurgical refining test work and concentrate de-watering test work. Additionally, the intention was to derive process design information to supplement the design data as derived from bench scale flotation test work. The latter objectives were only partially achieved as the MINTEK mini-plant was not adequately commissioned, stabilized and optimized due to a number of operational challenges which included stoppages due to power interruptions, inability to consistently dose copper collector reagent on a continuous basis, mechanical breakdowns and lack of assay data for operational control.

In order to meet the timelines required for Kell hydrometallurgical refining testwork, the concentrates generated during the June 2021 pilot run were re-floated in an 80 L flotation cell in batch mode in order to produce timed kinetic concentrate samples. These samples were dispatched for concentrate Kell testwork and concentrate de-watering testwork.

The November 2021 commissioning run of the Platreef circuit was conducted on a bulk shaft intercept sample with a measured 2PE+Au head grade of 3.8 g/t. The run achieved stable mass flows however large variances in final concentrate mass pull resulted in combined final concentrate grades of approximately 50 g/t to 78 g/t 2PE+Au at a mass pull of 5% to 8%. The averaged metallurgical projection data indicates that the mini-pilot run achieved an average 2PE+Au concentrate grade of 57 g/t at a recovery of 87% and 5.3% mass pull. A copper recovery of 86% was achieved and a nickel recovery of 80% was achieved. These results are indicative only as they reflect Mintek 2E+Au assay results and there were notable variances in the measured sub-sampled flotation feed grades during the run.

Further mini-plant optimization test work will be required to achieve the target concentrate grade of 85 g/t by targeting a lower mass pull aligned to optimized bench scale test locked cycle work. Based on previous locked cycle test work at Mintek in 2017 and SGS in 2020 it is expected that a reduction in mass pull will allow for increased concentrate grade with a 1% - 2% reduction in recovery.

The mini-pilot testwork included trials of an SIBX reagent suite with preliminary data indicating this to be a viable alternative to the copper collector reagent suite with no froth stability challenges evidenced during the November 2021 mini-pilot plant run.

The Platreef FS design includes an additional column cell to treat the combined medium grade and low-grade concentrate. The column has not been included in the locked cycle or mini-pilot plant flowsheet assessments. The inclusion is based on previous open circuit bench scale testwork during the 2017 FS which indicated the potential for upgrading this concentrate by a factor of 1.3 while achieving a high column stage recovery of up to 90%. Pilot scale column test work is recommended to confirm the additional upgrade potential in a column cell.

13.13 Concentrate De-Watering Testwork

Metso Outotec South Africa (MO) conducted test work on three concentrate samples generated from bulk batch flotation tests conducted during the June 2021 mini pilot run.

The sample D_{90} particle size distribution (PSD) as determined by laser diffraction method was in the range 40 to 80µm and the D_{50} was in the range 13 to 24µm.

The aim of the testwork was to determine concentrate thickener and filter design and operating parameters. The thickening test work included flocculant scouting tests and dynamic test runs using Metso Outotec's 99mm dynamic rig. The filtration test work included Larox filtration tests using Metso Outotec's Labox 100 test unit.

The results of the Metso Outotec concentrate thickening testwork are presented in Table 13.20.

Table 13.20 Summary of Optimal Concentrate Thickening Test Results

Parameter	Value
Thickener Type	High Rate
Feed Solids Concentration	10% (m/m)
Flocculant Type	Kemira A120
Flocculant Dosage	25 - 40g/t

The bench-top dynamic thickening tests indicated a decrease in underflow solids concentration (weight) from 62 - 64% reducing to 57 - 58% as the solids flux rate increased from 0.30 to 0.80 t/m2/h. The overflow clarities achieved were 250 to 300 mg/l for all the flux rates tested.

The results of the Metso Outotec concentrate filtration test work are presented in Table 13.21.

Table 13.21 Summary of Optimal Concentrate Filtration Test Results

Parameter	Value
Filter Type	Larox
Feed Solids Concentration	60 - 64% (m/m)
pH	8.57 – 8.85
Filter Flux	594 – 618 kg/m2h
Cake Moisture	11.0 – 13.3% (m/m)

The bench scale filtration test work indicated that the optimal operating times would be 2 minutes feeding time, 1.5 minutes pressing time and 2.5 minutes drying time. The test work achieved a high flux of approximately 600 kg/m^2h for all samples tested and confirmed the potential to achieve a final concentrate moisture of <14% (w/w).

13.14 Tailings De-Watering Testwork

13.14.1 Tailings Thickening Testing at Vietti Slurrytec

Vietti Slurrytec conducted testwork on one tailings composite sample representing the combined rougher and cleaner flotation tailings from testing on FS Composites 1–4. The objective of the testing was to obtain design and operating parameters. The testwork included material characterisation, slurry conditioning, static sedimentation and bench-top dynamic thickening testwork.

The material characteristics and operating parameters obtained from the Vietti Slurrytec testwork are presented in Table 13.22 and Table 13.23.

Table 13.22 Summary of Material Characterisation Results Vietti Slurrytec

Parameter	Value
Tailings pH	8.6
Process Water pH	7.6
Tailings D_{80}	82 microns
Specific Gravity	3.12

Table 13.23 Summary of Optimal Tailings Thickening Test Results Vietti Slurrytec

Parameter	Value
Thickener Type	High Rate
Feed Solids Concentration	12.5% (m/m)
Flocculant Type	Magnafloc 5250
Flocculant Dosage	30g/t

The bench-top dynamic thickening tests indicated a decrease in underflow solids concentration from 66–57% as the solids flux rate increased from 0.30–0.90 t/m^2/h, with good overflow clarity achieved.

It is recommended that pipe loop and pump de-rating testwork be conducted prior to the detailed design phase, to confirm the tailings pumping system design and operating parameters.

13.14.2 Golder Tailings Thickening and Filtration Testwork for Backfill Plant Design

In addition to the tailings settling testwork conducted at Vietti Slurrytec, further testwork was undertaken by Golder on Platreef Mine (Platreef) tailings during the FS in 2016. The data of this test work campaign was utilised for the paste plant design to DFS level. For detail laboratory test results refer to DRA Report Number: DRA-J0283-STU-REP-909, Section 6.2 (Ivanplats Report Number: 1051-EV-00-209).

13.14.2.1 Testwork Samples

Two samples were received in dry form: "1530678 Combined Final Tails" and "1530678 Combined Final Tails - S2". Sample "1530678 Combined Final Tails" was used for all materials property testing in the laboratory. Process water was also received.

The tailings and process water were assessed individually for concentrations of volatile organic compounds, hydrogen cyanide gas, hydrogen sulfide gas, heavy metals, and dissolved cyanide. All concentrations were found to be within acceptable levels.

The testwork included material characterisation, settling tests and filtration tests. It should be noted that the Golder testwork was done to obtain a maximum underflow solids concentration to produce a vacuum filter feed for the paste plant while the Vietti Slurrytec testwork was done to obtain a suitable underflow solids concentration for the pumping of tailings to the TSF.

13.14.2.2 Material Characterisation

The material characteristics of the samples as tested are presented in Table 13.24.

Table 13.24 Summary of Material Characterisation Results Golder

Parameter	Value
Tailings pH	8.9–9.0
Process Water pH	6.9 which was adjusted to pH 9.0 prior to testing
Tailings D_{80}	105 microns
Specific Gravity	3.06–3.08

Mineralogy and chemistry tests were conducted on the tailings solids for the following purposes:

- To confirm that the sample as tested falls within anticipated mineralogical variations of the ore.
- To allow comparison, should future testing occur.

For the detailed mineralogy and geochemical results refer to the test report. It should be noted that since the two tailings samples received had nearly identical characteristics, were therefore deemed "the same" and only one tailings sample was used for all remaining laboratory testing.

13.14.2.3 Testwork Samples

The settling tests for the 1530678 Combined Final Tails were completed in an iterative fashion to determine optimum values involving flocculant type, flocculant dosage, and feed slurry solids concentration. Different flocculant dosage and feed rates were evaluated in 500 ml, 1 l and 4 l vessels.

The dewatering and settling results are summarised in Table 13.25 and Figure 13.34.

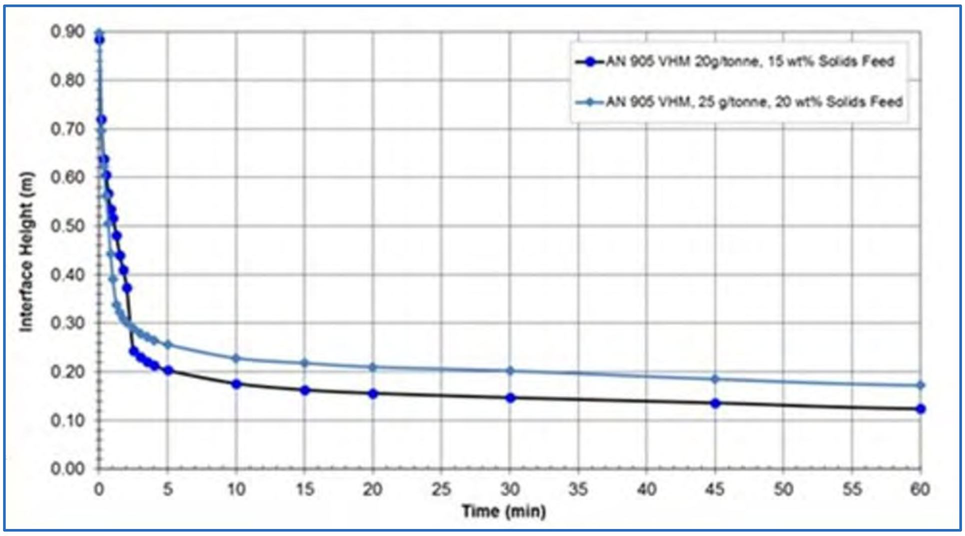
Table 13.25 Summary of Tailings Thickening and Filtration Test Results for Backfill Plant Design

Parameter	Value
Thickener Type	High Rate
Feed Solids Concentration	15–20% (m/m)
Flocculant Type	AN 905 VHM
Flocculant Dosage	20–25g/t
24 Hour Underflow Solids	69% (m/m)
Maximum Underflow Solids	78% (m/m)

The settling tests, as presented in Figure 13.34 provided an observation regarding the rapidity of settling that will be an important consideration in the flocculation process and the operation of a thickener. The settling curve indicates very rapid settling into the compression zone within one minute of flocculation with a dosage rate of 25 g/t and 20 wt% solids feed. An accidental overdose of flocculant above the recommended maximum of 25 g/t range will greatly increase the settling rate and could be problematic in "bogging" the thickener's rake mechanisms. Consideration is warranted in retaining good flocculant addition control.

As a general summary, the tailings will dewater acceptably using available thickening technology. A further improvement of as much as 5 wt% in underflow density is often observed in production scale thickeners when compared to the bench scale 4 l test 24-hour results.

Figure 13.34 Settling Curve, 15–20% Solids Feed, 20–25 g/t



Golder, 2017

13.14.3 Tailings Filtration Testwork

Vacuum filtration tests, to simulate the results associated with disc filters, were conducted at a vacuum level of 557 mm (22") Hg. Typical cycle times (a full rotation) for vacuum disc filters are in the range of 90–180 seconds.

The vacuum filtration testwork programme was performed with a slurry feed solid concentration of 69 wt% which is the underflow value obtained in the 4 l tests. An additional series of filtration tests were also performed at 64 and 74 wt% solids.

The filter leaf was equipped with a small section of industrial grade polypropylene felt filter cloth. The leaf was immersed into the slurry and simulated production scale vacuum filters where the sectors dipped into the slurry in an agitated filter tank as the disc rotated. Proper technique and cycle times simulating continuous filters provide an estimate of cake loading, moisture and discharge characteristics.

Since the test was performed in the laboratory, under ideal conditions, actual loading is multiplied by 0.7 to reflect variable or upset conditions which may occur in plant operations.

The following parameters were used for testing:

- Elevation:1,109 masl (1).
- Vacuum Level: 557 mm (22") Hg.
- Temperature: 18.1˚C.
- Filter Cloth: Industrial grade polypropylene felt 133-03 (25–40 cfm rating).
- Apparatus: 100 mm (4 inch) diameter, dip style filter head.
- The Process water pH was adjusted to 9.0 prior to filtration testing.

The results are presented in Table 13.26, Table 13.27 and Table 13.28 as well as on Figure 13.35.

Generally, the higher the solids content feed to the vacuum disc filters, the higher the loading capacity. Loading capacity from 64–74 wt% solids feed provides a range of values to consider for disc filter sizing purposes. Based on the settling and filtration results, 609 kg/m²/hr cake loading rate at 69 wt% solids with 90s cycle time is chosen for disc filter sizing.

Table 13.26 Tailings Filtration Results – 74 wt% Solids

Vacuum Level (mm Hg)	Cycle Time (sec)	Cake Thickness (mm)	Cake Loading (kg/m²/hr)	Cake Moisture (wt%)	Final Density (wt% Solids)
557	30	10.5	1535	18.3	81.7
	45	11.7	1221	18.1	81.9
	90	14.9	769	17.4	82.6
	135	17.8	596	17.1	82.9
	180	18.9	504	17.1	82.9

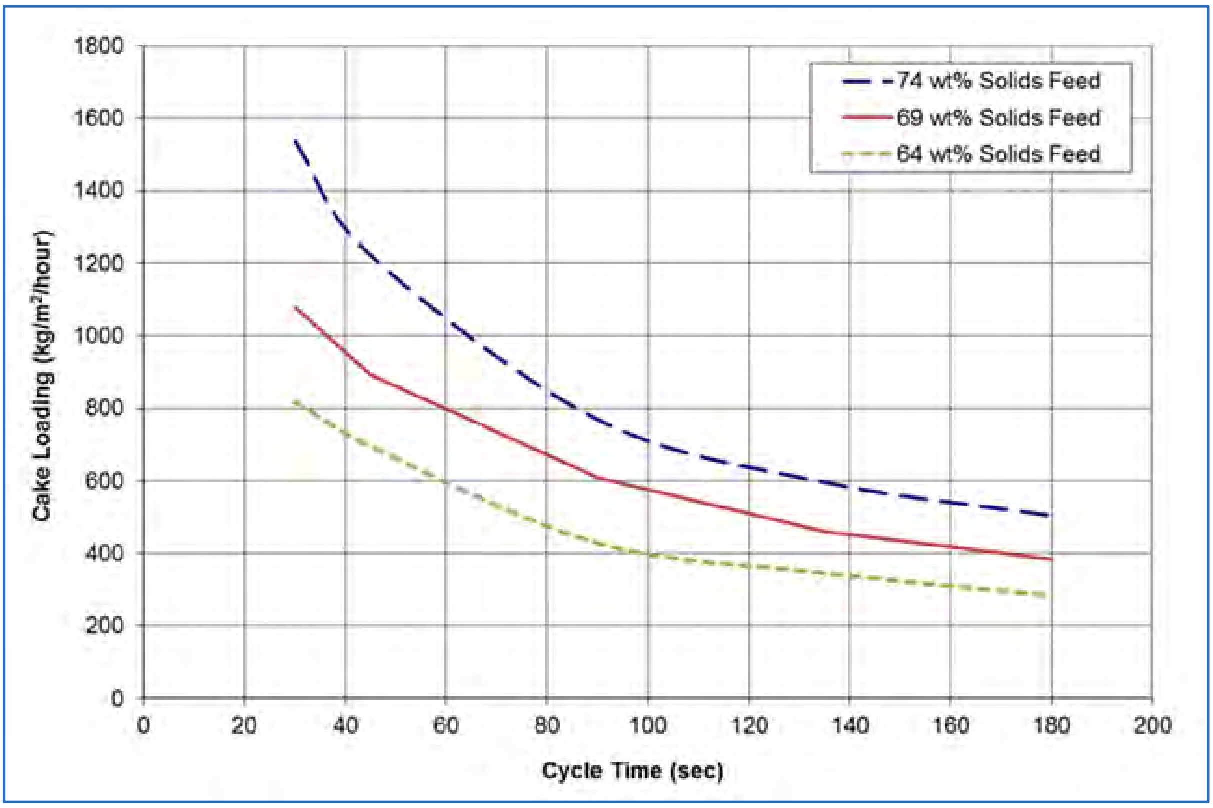
Table 13.27 Tailings Filtration Results – 69 wt% Solids

Vacuum Level (mm Hg)	Cycle Time (sec)	Cake Thickness (mm)	Cake Loading (kg/m²/hr)	Cake Moisture (wt%)	Final Density (wt% Solids)
557	30	7.5	1079	18.0	82.0
	45	9.4	892	17.1	82.9
	90	12.6	609	16.7	83.3
	135	14.1	460	16.9	83.1
	180	15.8	383	16.7	83.3

Table 13.28 Tailings Filtration Results – 64 wt% Solids

Vacuum Level (mm Hg)	Cycle Time (sec)	Cake Thickness (mm)	Cake Loading (kg/m²/hr)	Cake Moisture (wt%)	Final Density (wt% Solids)
557	30	5.7	818	17.2	82.8
	45	7.5	694	17.8	82.2
	90	8.8	428	17.2	82.8
	135	11.5	346	16.7	83.3
	180	12.4	283	16.6	83.4

Figure 13.35 Cake Loading vs. Cycle Time – 1530678 Combined Final Tails



Golder, 2017

13.14.4 Conclusions

From the laboratory assessment conducted on the feasibility paste fill samples it was concluded that the paste fill tailings can be initially dewatered through the use of available thickener and filtration technology.

13.15 Plant Recovery Estimate

A nominal head grade recovery relationship was derived based on 2017 locked cycle test results conducted using the MF1 copper collector reagents suite and the optimised Platreef 2017 FS flow sheet (Flow sheet 3) as presented in Figure 13.30.

13.15.1.1 Summary of Test Data Used

The Platreef recovery regression models, as presented, have been derived using the 2017 FS locked cycle test results for domain composites, comprising of the expected mine blend in the first five years of production and the overall life of mine blend.

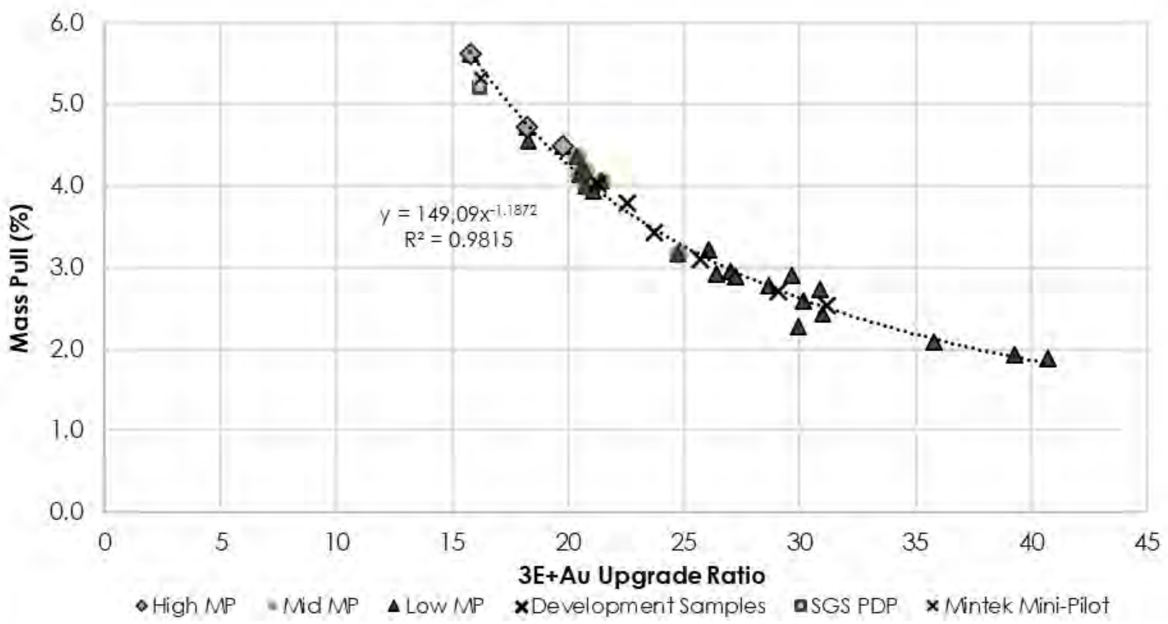
The locked cycle test data for the initial development samples were based on the Platreef 2015 PFS mine blend using interim development flow sheets and have thus not been used in the recovery modelling. Additionally, the 2020 locked cycle test work undertaken as part of the SGS PDP test program was also not used as it reflects test work results using a proxy copper collector reagent. The mini pilot plant data from the June 2021 and November 2021 runs have not been used for recovery modelling as this data is considered to reflect preliminary commissioning results. All of these results, however, have been compared to the recovery model predictions based on the 2017 FS locked cycle test results and were found to produce a reasonably good fit to the recovery models.

All comparative grade and recovery data for the mini-pilot plant in the plots that follow reflect 2PE+Au data and not 3PE+Au data.

13.15.1.2 Derivation of 3PE+Au Recovery Models

Locked cycle testwork showed a strong relationship between concentrate mass pull, achievable upgrade ratios and consequently final concentrate grades. Mass pull was dependent on the PGE upgrade ratio targeted in the locked cycle testing as presented in Figure 13.36. This curve illustrates that as higher concentrate grades are targeted (High upgrade ratio), the overall concentrate mass pull will decrease.

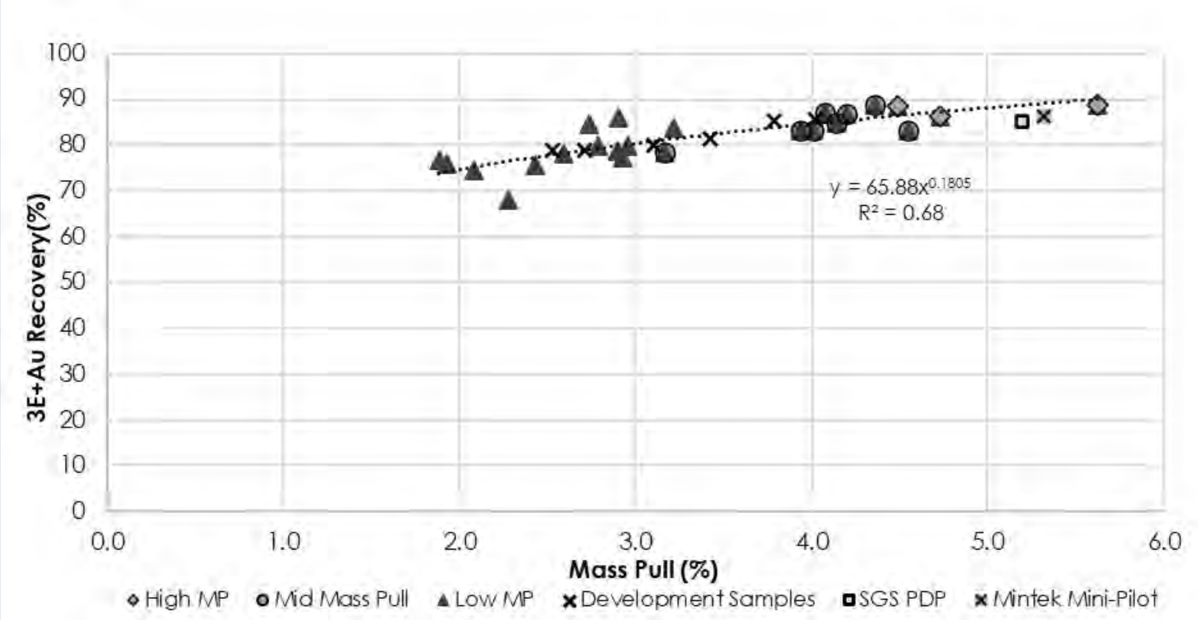
Figure 13.36 Locked Cycle Test Mass Pull as a Function of PGE Upgrade Ratio



DRA, 2021

In turn, testwork showed that metal recoveries were dependent on mass pull as presented in Figure 13.37. It is noted that some scatter exists between the results, which is believed to be attributed to sample variances, test operator inconsistencies, assay uncertainty and variances in the final concentrate grade achieved.

Figure 13.37 3PE+Au Recovery as A Function of Mass Pull



DRA, 2021

The combination of Figure 13.36 and Figure 13.37 thus illustrates that as higher 3PE+Au concentrate grades are targeted (High upgrade ratio) the overall concentrate mass pull will decrease, and consequently the 3PE+Au recovery would be lower than when targeting a lower concentrate grade (low upgrade ratio).

These relationships between mass pull and 3PE+Au recovery, as a function of upgrade ratio, allow for modelling of the expected recoveries based on feed grade and target concentrate grade and form the basis of the Platreef recovery derivation.

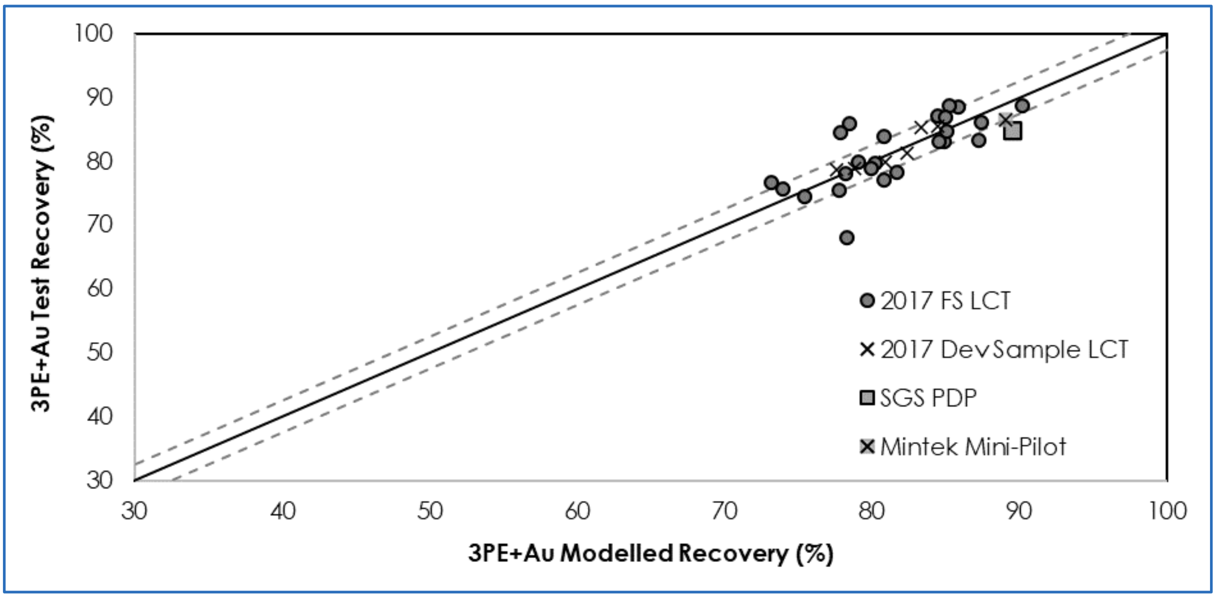
The recovery model has been derived as follows:

- Mass pull is calculated based on the targeted 3PE+Au upgrade ratio (concentrate grade:feed grade).
- Individual Pt, Pd, Rh and Au recoveries are calculated based on the mass pull recovery curves derived per element.

13.15.1.3 3PE+Au Recovery Model Verification

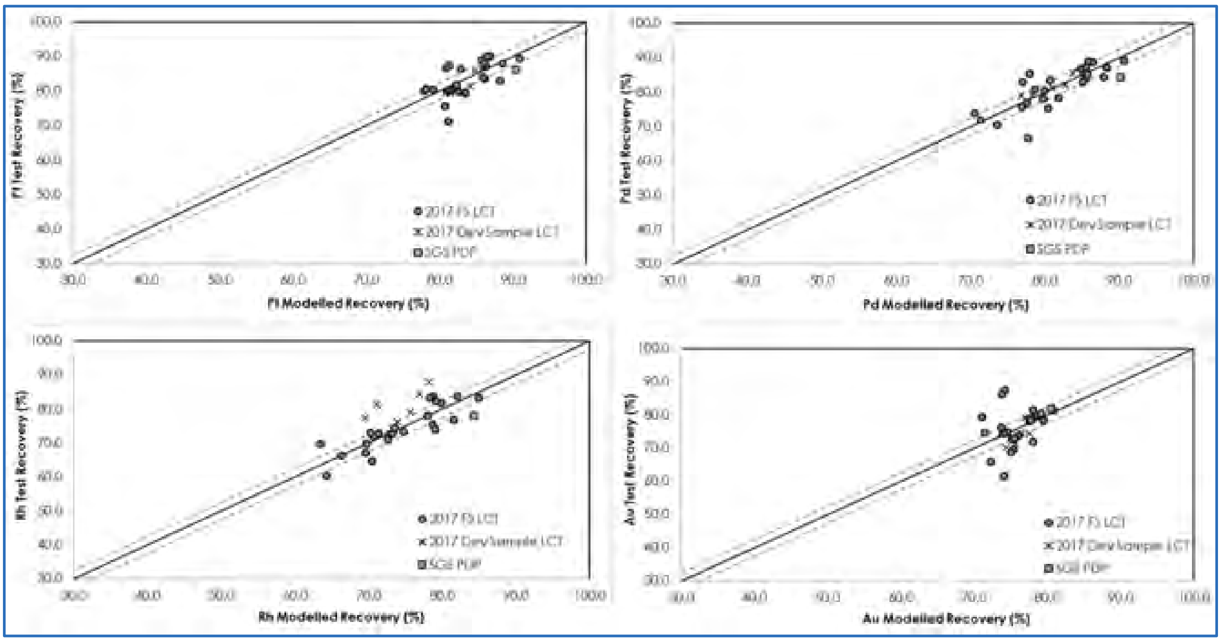
The PGE predicted recovery was compared to actual test results in order to verify the accuracy of the models. The modelled PGE, Pt, Pd, Rh and Au recovery as compared to actual test recoveries are presented in Figure 13.38 and Figure 13.39.

Figure 13.38 Combine PGE (3PE+Au) Recovery Model Verification



DRA, 2021

Figure 13.39 Individual, Pt, Pd, Rh and Au Recovery Model Verification



DRA, 2021

The modelled recovery data was found to fit the actual test data well, the upper and lower lines in these curves represent the bounds associated with modelled vs actual recovery variance. For Pt, Pd, Rh and Au, the variance was found to be within ±2.5% in the vast majority of instances.

The 2020 SGS PDP locked cycle test was found to exhibit a higher degree of variance for Pt, Pd and Rh however the data falls within the range evidenced in the 2017 locked cycle test work. The use of a proxy copper collector reagent, sample variances, test operator inconsistencies, assay uncertainty and variances in the actual final concentrate grade achieved are a likely contributing factor for the variance evidenced.

13.15.1.4 Derivation of Copper, Nickel and Sulfur Recovery Models

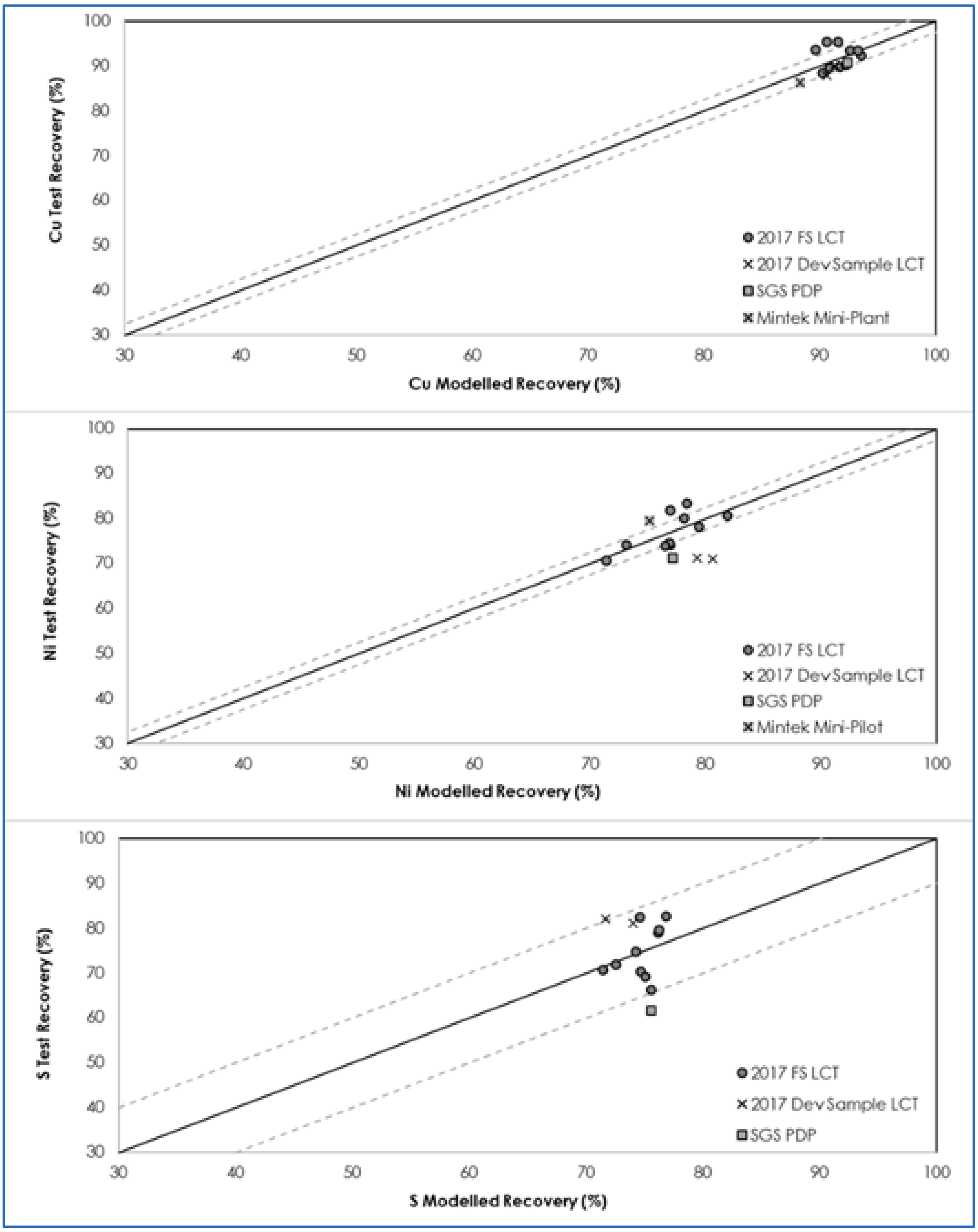
During commercial operation, the final concentrate mass pull and 3PE+Au recovery will be determined by the upgrade ratio required to achieve a minimum final concentrate grade of 85 g/t (3PE+Au). For the range of head grades tested, and the expected mass pull range as determined based on the Platreef 2017 FS mining schedule, the nickel, copper and sulfur recoveries are best described by head grade versus recovery relationships.

The copper, nickel and sulfur test recoveries achieved to the combined final concentrate (HG+MG+LG) in locked cycle testing were plotted relative to the respective test head grade in order to derive head grade-recovery curves.

13.15.1.5 Base Metal and Sulfur Recovery Model Verification

The method as outlined was used to derive recovery correlations for copper, nickel and sulfur. The recovery predictions were compared to laboratory test results in order to verify the model accuracies as presented in Figure 13.40.

Figure 13.40 Copper, Nickel and Sulfur Recovery Model Verification



DRA, 2021

The modelled recovery data was found to fit the actual Platreef locked cycle test data well for copper and nickel, where the variance was found to be within ±2.5% in the majority of instances. The variance in the modelled vs actual sulfur recovery was found to be within ±10.0%.

The higher variance for sulfur is possibly due to the analytical methods employed, however this would not affect the overall project economics as sulfur is not a payable metal. The effect of the uncertainty is discussed and illustrated in Sections to come.

13.15.1.6 Recovery Algorithms

The Platreef 2022 FS Recovery Algorithm is the same as the Platreef 2017 FS. The final recovery model (as a function of feed grade and targeted concentrate grade) for the individual metals are summarised in Table 13.29.

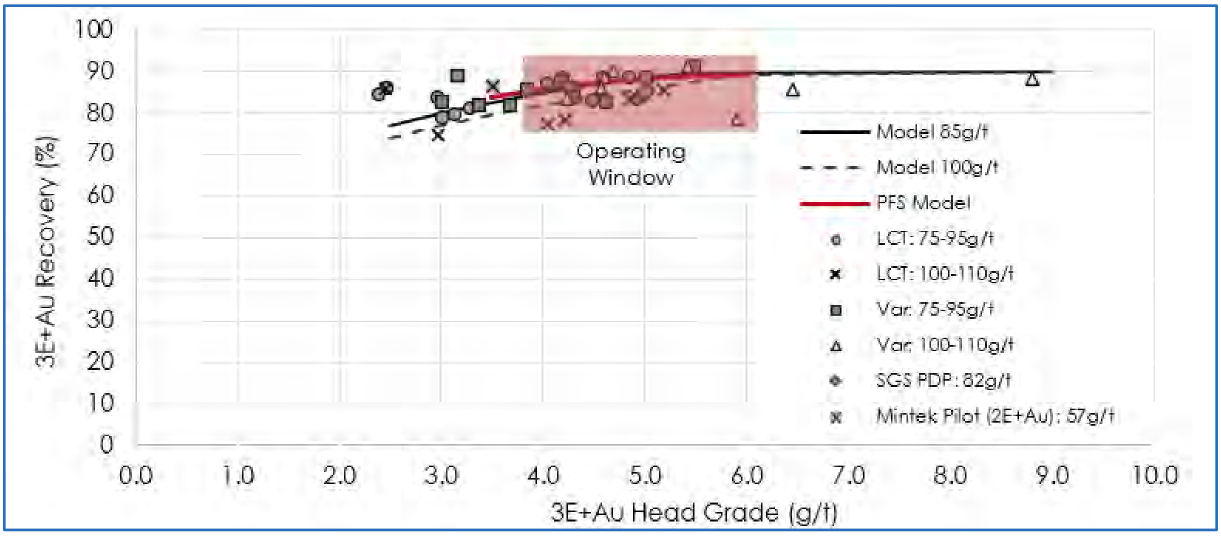
Table 13.29 Platreef 2022 FS Recovery Algorithms

Element/ Metal	Recovery Algorithm	Head Grade Range	Max Recovery
Platinum	Recovery = $e^{4.1501+0.15166\times\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}$	1.0–2.7g/t	90.9
Palladium	Recovery = $e^{4.8401-\frac{0.79022}{\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}}$	1.2–2.7g/t	90.7
Rhodium	Recovery = $52.042 + 19.041 \times Ln\left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)$	0.06–0.19g/t	84.9
Gold	Recovery = $66.424 \times \left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.1137}$	0.18–0.35g/t	80.8
Copper	Recovery = $129.78 \times Cu\ Head\ Grade^{0.2174}$	0.15–0.23%	93.6
Nickel	Recovery = $125.33 \times Ni\ Head\ Grade^{0.5145}$	0.30–0.44%	82.2
Sulfur	Recovery = $75.569 \times S\ Head\ Grade^{0.328}$	0.78–1.05%	76.8

By using these recovery algorithms for each individual element, it is possible to calculate the expected overall 3PE+Au recovery based on the 3PE+Au target concentrate grade.

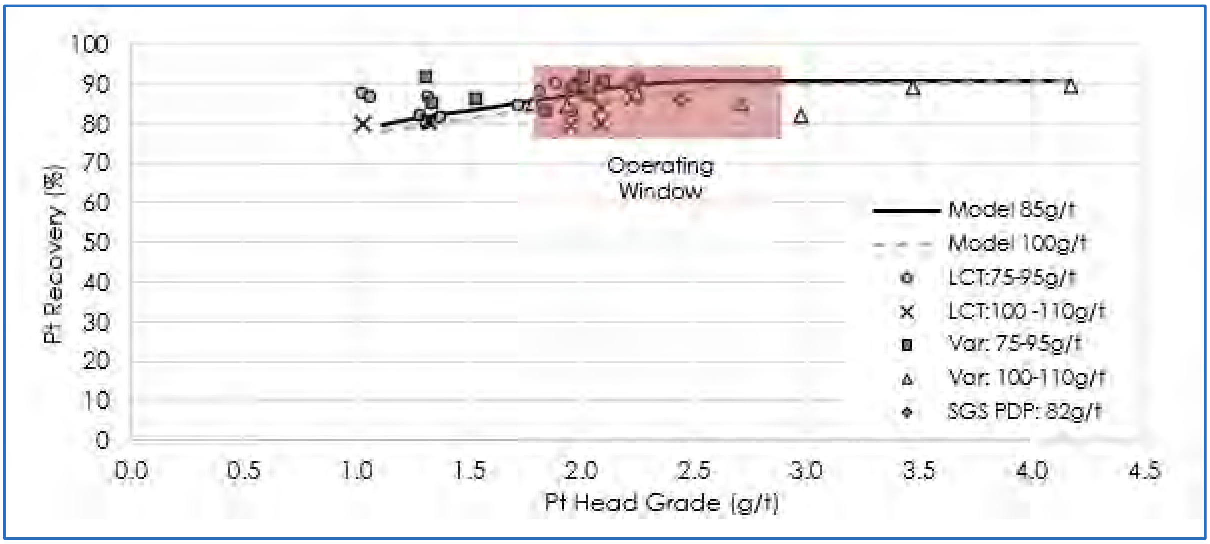
The PGE (3PE+Au) recovery models have been plotted relative to the locked cycle test results and expected closed circuit recoveries derived from the open circuit variability testwork in Figure 13.41 to Figure 13.45. As previously noted, the recovery estimates as derived from open circuit variability results are not definitive, however these results do provide an indication of the expected variability. It should be noted that the Platreef 2022 FS recovery estimates compare well with models developed during the Platreef 2015 PFS phase, highlighting the consistency between the various campaigns. Variances when compared to test results are exaggerated due to the fact that variances exist in the final concentrate grade achieved during testwork and modelled.

Figure 13.41 3PE+Au Recovery as a Function of Feed Grade and Targeted Concentrate Grade



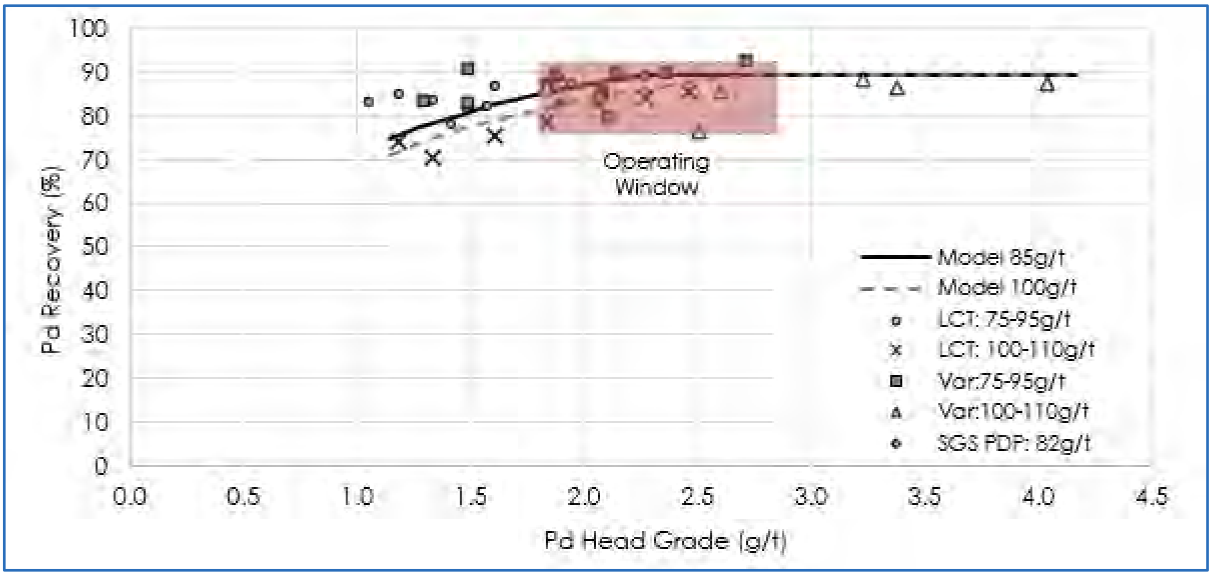
DRA, 2021

Figure 13.42 Platinum Recovery as a Function of Feed Grade and Targeted Concentrate Grade



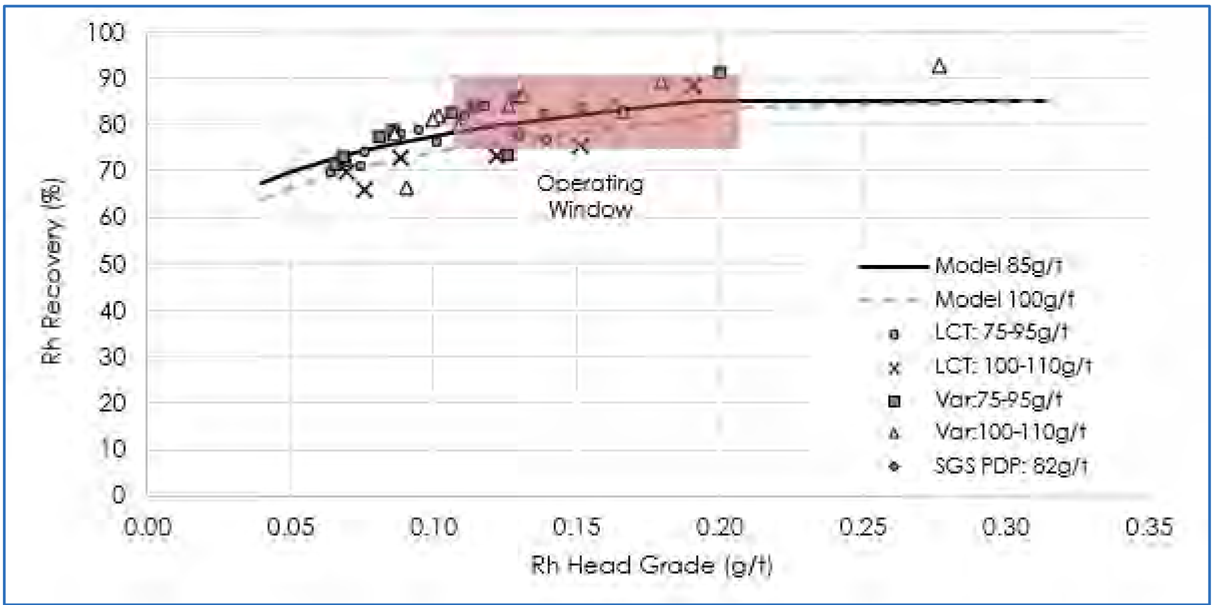
DRA, 2021

Figure 13.43 Palladium Recovery as a Function of Feed Grade and Targeted Concentrate Grade



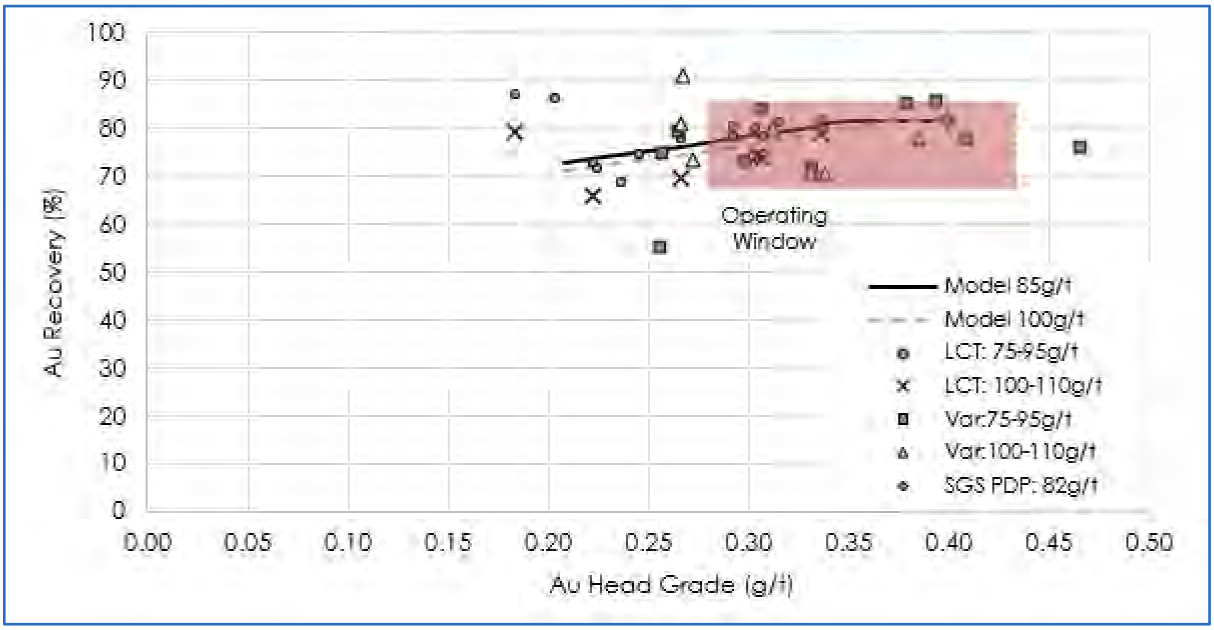
DRA, 2021

Figure 13.44 Rhodium Recovery as a Function of Feed Grade and Targeted Concentrate Grade



DRA, 2021

Figure 13.45 Gold Recovery as a Function of Feed Grade and Targeted Concentrate Grade

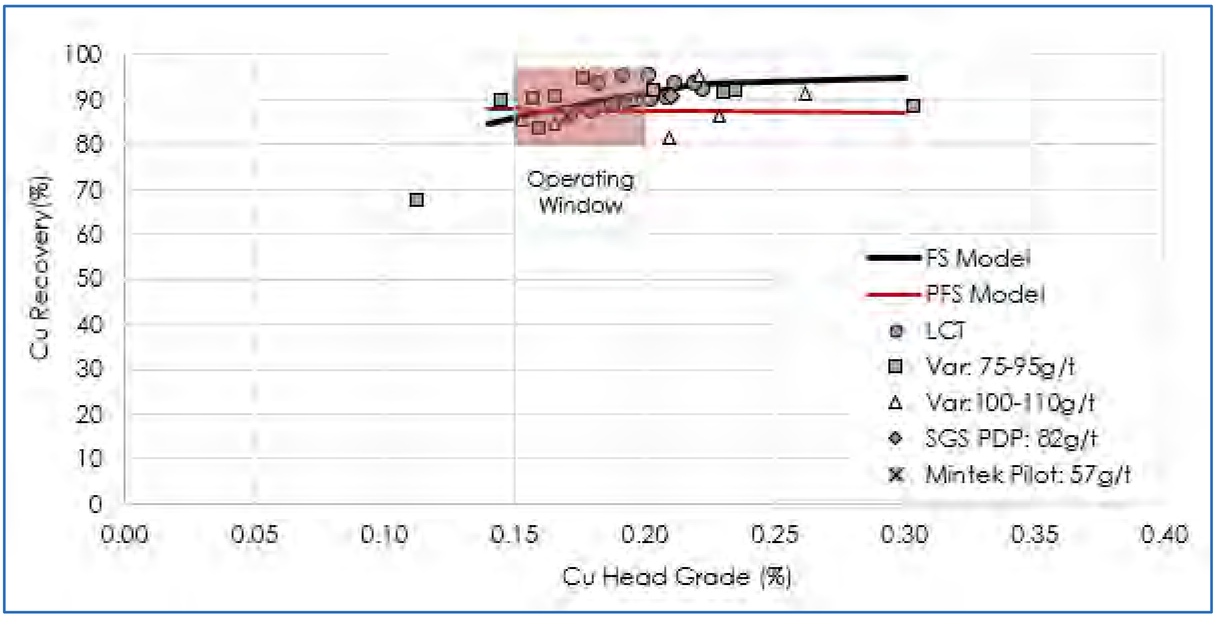


DRA, 2021

The copper and nickel recovery models have been plotted relative to the locked cycle test results and expected closed circuit recoveries derived from the open circuit variability testwork in Figure 13.46 and Figure 13.47.

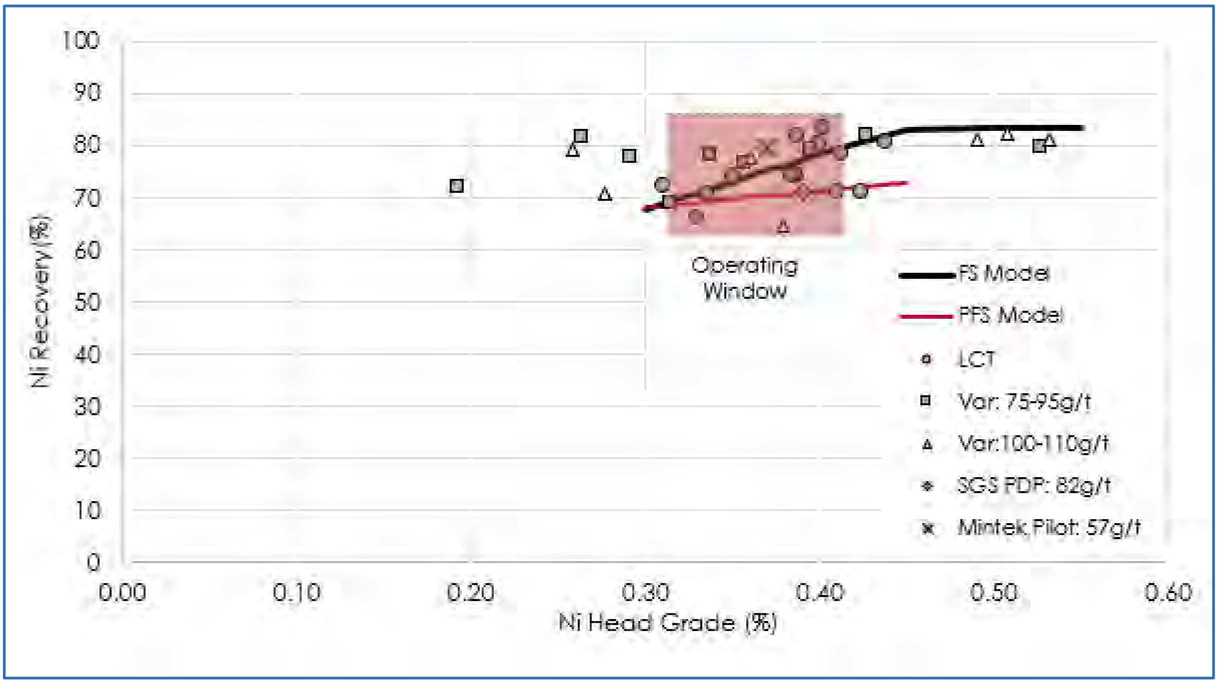
As previously noted, the recovery estimates as derived from open circuit variability results are not definitive, however these results do provide an indication of the expected variability. It should be noted that the Platreef FS recovery estimates derived from the 2017 locked cycle testwork data have indicated improved copper and nickel recovery as compared to Platreef 2015 PFS testing at similar head grades. This is possibly as a result of the optimised Platreef 2017 FS flotation conditions, targeting high mass pull using the copper collector reagent suite.

Figure 13.46 Copper Recovery as a Function of Feed Grade



DRA, 2021

Figure 13.47 Nickel Recovery as a Function of Feed Grade

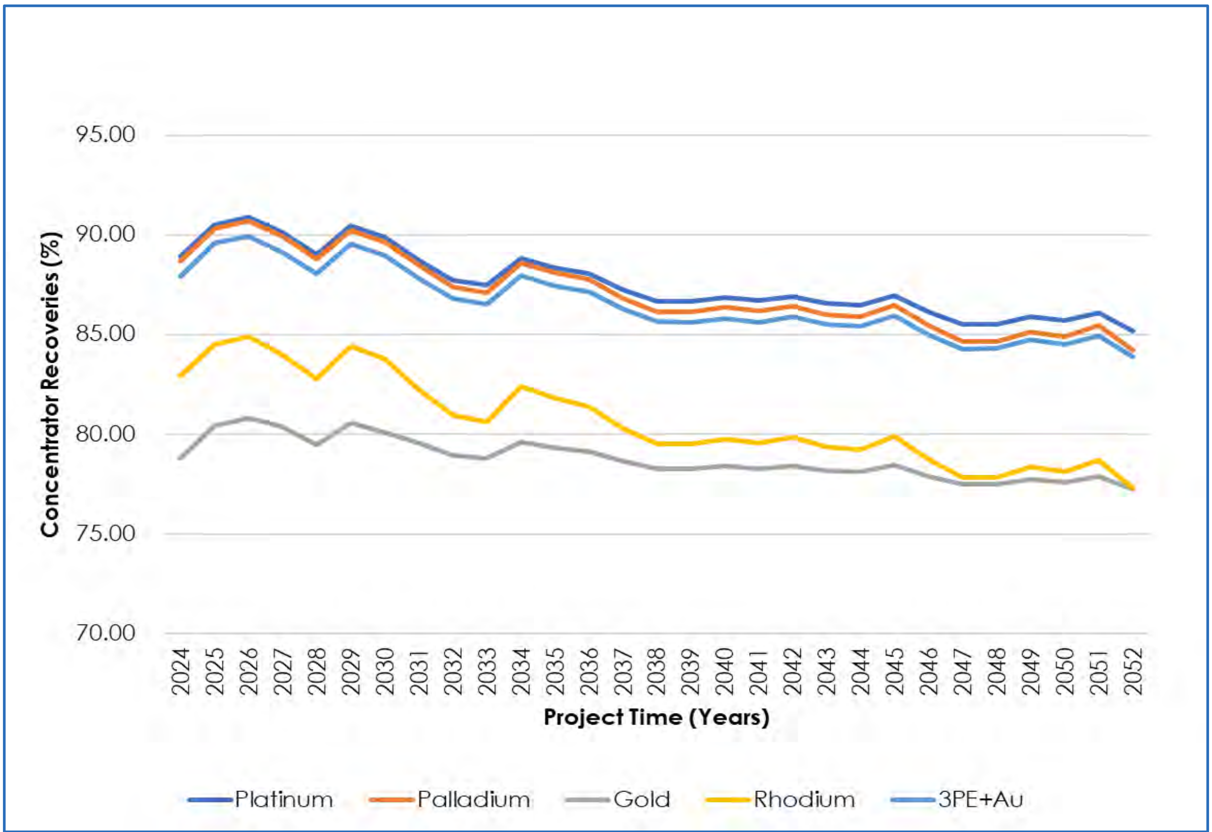


DRA, 2021

13.15.1.7 Life-of-Mine Recovery Estimates

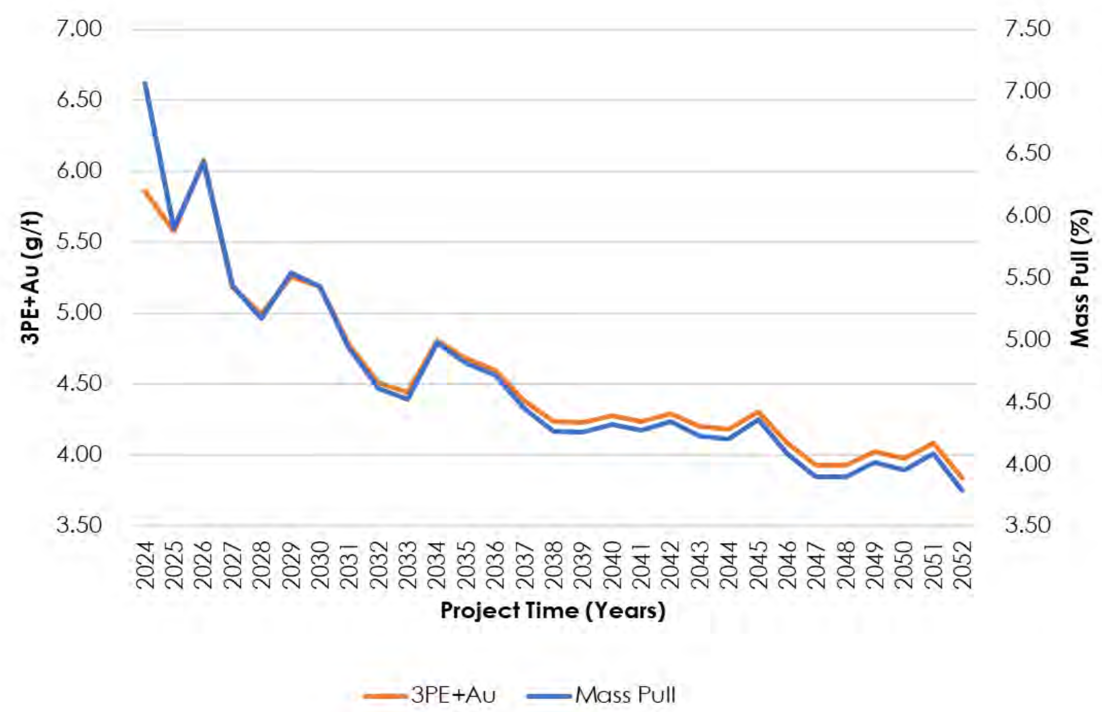
The recovery estimate for PGE's and expected concentrate mass pull over life of mine are presented in Figure 13.48 and Figure 13.49. The concentrate grade profile and calculated sulfur grade in concentrate are presented in Figure 13.50 and Figure 13.51. These recoveries and grade profiles have been based on the expected concentrator feed grades from the Platreef 2022 FS mine schedule. The recoveries have been derived for a minimum target grade of 85 g/t. It should be noted that the modelled recoveries are steady-state recoveries and do not consider any transient operations. The modelled recovery profile includes a 2% recovery discount in the first six months of operation to cater for start-up and commissioning.

Figure 13.48 Modelled Steady State Life-of-Mine 3PE+Au Recovery Profile



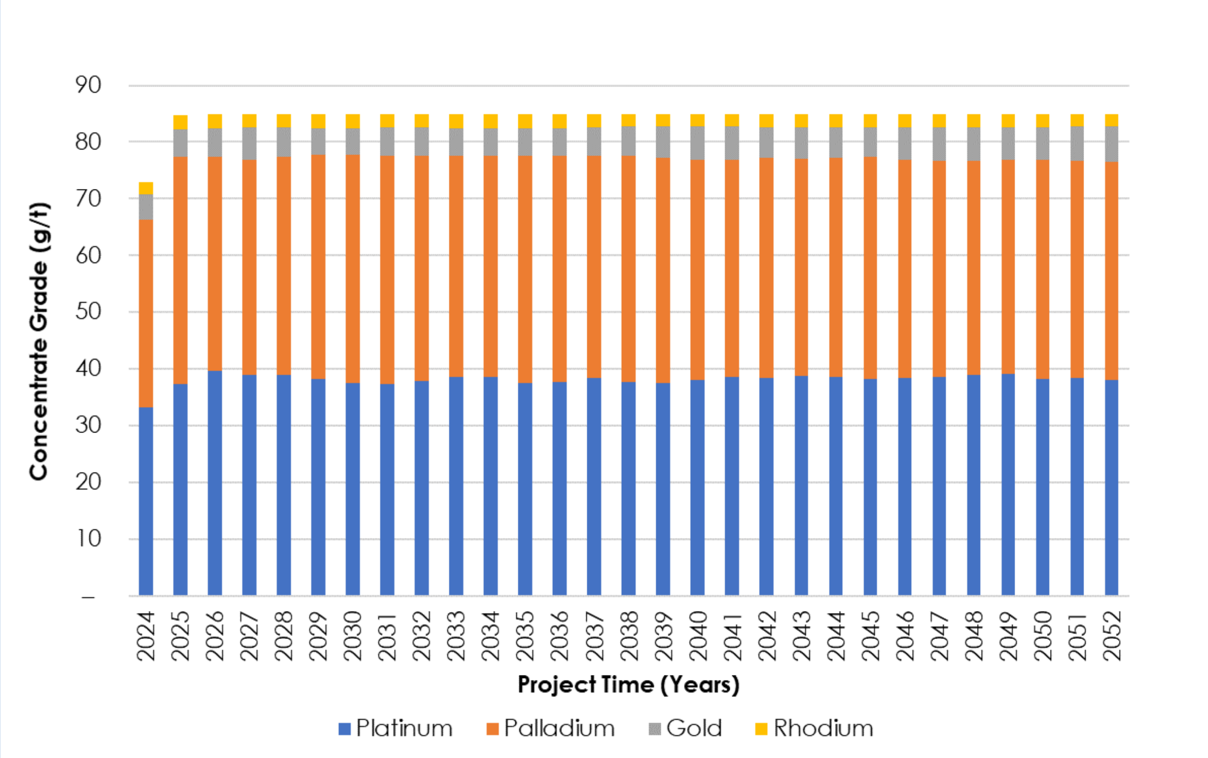
OreWin, 2021

Figure 13.49 Modelled Life-of-Mine Head Grade and Mass Pull Profile



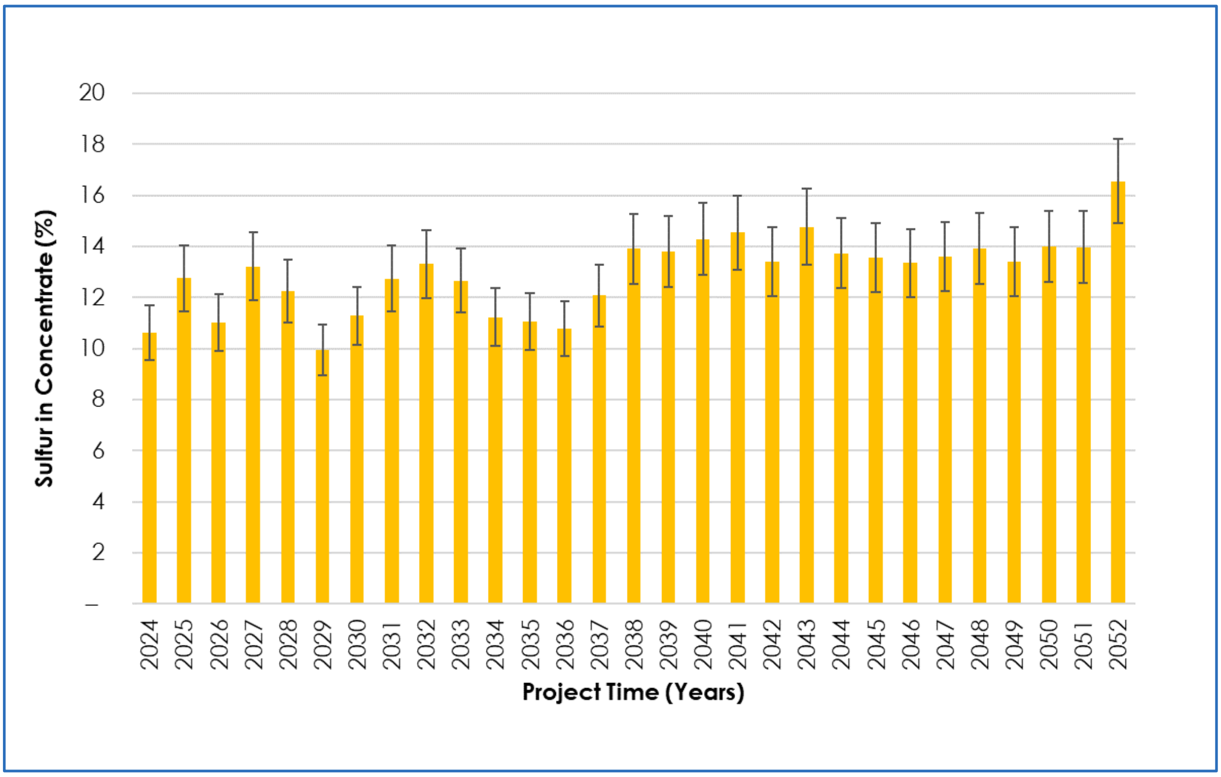
OreWin, 2021

Figure 13.50 Modelled Life-of-Mine Concentrate Grade Profile



OreWin, 2021

Figure 13.51 Modelled Life-of-Mine Sulfur in Concentrate



OreWin, 2021

The error bars as presented on the expected sulfur grades in concentrate (Refer Figure 13.51) represent the 10% variance as evidenced in the modelling of the sulfur recovery test data.

13.15.2 Comments on the Recovery Estimate

Locked cycle testwork on mineralised ore blends with the inclusion of footwall and hanging wall has been conducted. Further to this, detailed open circuit variability testing was conducted in order to quantify the expected recovery ranges and to highlight the degree of variability that can be expected.

The open circuit and locked cycle testwork conducted at SGS and Mintek during the Platreef 2022 FS testwork campaign were noted as being within the ranges evidenced in the 2017 FS variability testwork albeit on the lower end. The core composites that were used for these testwork campaigns reflect un-crushed ½ and ¼ drill core remainders from the 2017 FS drilling campaign. The age of these samples (4 – 5 years) may potentially have impacted on performance additionally sample variability, assay variances, procedural variances and non-optimized mass pull may also have contributed.

A mini pilot plant campaign was conducted, primarily, to produce bulk concentrate samples for downstream hydrometallurgical refining test work and concentrate de-watering test work. The added objective of deriving additional design data from the pilot runs was only partially achieved due to a number of operational challenges at Mintek. These runs are thus considered to reflect preliminary commissioning results. These commissioning runs, successfully, allowed for generation of concentrate samples for Kell test work and concentrate de-watering test work but did not provide comprehensive data to fully confirm the comprehensive design and metallurgical performance data. Additionally, the majority of the runs reflect commissioning runs on low grade samples with a 3PE+Au head grade of 2.9 to 3.8 g/t. To further evaluate optimisation opportunities and confirm additional detail design parameters, additional pilot plant test work on high grade samples aligned to the early years of mining (> 5g/t 3PE+Au) is proposed as part of the project implementation phase.

The locked cycle test results as derived during the 2017 FS are considered adequate for deriving metallurgical performance projections.

It is also noted that in addition to the recovery uncertainty demonstrated during variability testing, flotation recovery for full-scale operations could be lower than that achieved in a laboratory due to operational inefficiencies such as those listed below:

- Variation in ore types and blends: The mine plan includes geometallurgical units T1, T2U and T2L as well as fractions of hanging wall and footwall. The mine plan caters for mining in ten distinct areas.
- Power: The laboratory flotation cell power (and air) inputs are extremely high (typically in the order of 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 mill, as opposed to the actual plant which is often undertaken with ball milling. The difference in particle size distribution between these two types may have an effect on performance.
- Operating conditions: Laboratory operation is undertaken under controlled, 'ideal' conditions. Operational disturbances on full-scale operations such as starting and stopping of the plant undoubtedly cause loss of recovery.
- Operational skills: The bench scale laboratory tests are supervised by 'expert' operators. In the actual plant recovery losses may occur as a result of poor operational practices.

In order to address as many of these problems as possible the plant design will allow a high level of instrumentation and control within the flotation and milling circuit with the allowance for installation of Float Star and a Blue Cube online analyser to allow for improved flotation control. Process operators need to be trained and supervised to reduce the occurrence of losses due to poor operational practices.

It should also be noted that Mintek has reported that the Pt, Pd, Rh and Au detection limit is 0.02 ppm by ICP-OES analysis method used during the Platreef 2017 FS test campaign. All Rh recovery calculations as presented in this document have been based on this minimum detection limit in instances where values below this limit have been reported.

13.16 Future Testwork

The bench scale test work conducted to date is considered adequate to meet the requirements of the Platreef 2022 FS. Mini pilot test work as conducted at Mintek in June 2021 and November 2021 is considered to reflect preliminary commissioning runs.

To further evaluate optimization opportunities, confirm design parameters, evaluate scale-up factors and establish definitive criteria for performance guarantees, additional pilot plant test work should be considered. This testwork should be conducted on high grade samples aligned to the early years of mining (>5 g/t 3PE+Au). It is not believed that this will have a material impact on the overall capital and operating cost estimates derived in the current study phase and is furthermore expected to validate the recovery and concentrate specifications achieved during bench scale locked cycle test work on similar material once plant operation and performance has been fully optimized.

The mini-pilot testwork included trials of an SIBX reagent suite with preliminary data indicating this to be a viable alternative to the copper collector reagent suite. Additional testwork should be conducted to confirm this result and the inclusion of an SIBX make-up and dosing system should be undertaken during project implementation.

Repeat Jameson test work using the updated vendor procedure is scheduled to take place in the first quarter of 2022.

Pilot scale column test work is recommended to confirm the additional concentrate upgrade potential in a column cell as aligned to the Platreef design flowsheet.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Platreef Mineral Resource model includes three Mineral Resource estimates completed in 2015 and 2016 and validated as current in 2022:

- TCU Model (UMT-TCU) – The TCU Mineral Resources includes material within and adjacent to grade shells (3PE+Au) in the TCU. This Mineral Resource was updated using the revised geological interpretation. Additional drilling was completed in Zone 1, Zone 3 and Zone 5 (see Figure 7.1). The Mineral Resource amenable to selective underground mining methods is supported by the UMT-TCU model. Indicated and Inferred Mineral Resources were estimated for the UMT–TCU model.
- Bikkuri Model (UMT-BIK) – This consists of material within and adjacent to grade shells in the Bikkuri Reef. This Mineral Resource was estimated using a revised geological interpretation and incorporation of additional drilling in Zone 1. The Bikkuri reef has also been identified in Zone 2. The Mineral Resources amenable to selective underground mining methods in the Bikkuri Reef are supported by the UMT-BIK model. Indicated and Inferred Mineral Resources were estimated for the UMT-BIK model.
- Footwall Model (UMT-FW) – The UMT-FW model includes material that is footwall to the UMT-TCU model. This Mineral Resource was estimated using the revised geological interpretations and additional drilling in Zone 1. The UMT-FW Mineral Resources are potentially amenable to selective underground mining methods and possibly mass mining methods in local areas. Inferred Mineral Resources were estimated for the UMT–FW model.

The recognition of lithological controls (TCU stratigraphy) on grade has enabled declaration of Inferred Mineral Resources at wider drill spacings than would normally be possible. Infill drilling in Zone 1, Zone 2 and Zone 3 permitted the declaration of Indicated Mineral Resources in that portion of the Platreef Project area. Inferred Mineral Resources are declared in Zone 2, Zone 3 and the Madiba area (see Figure 14.1).

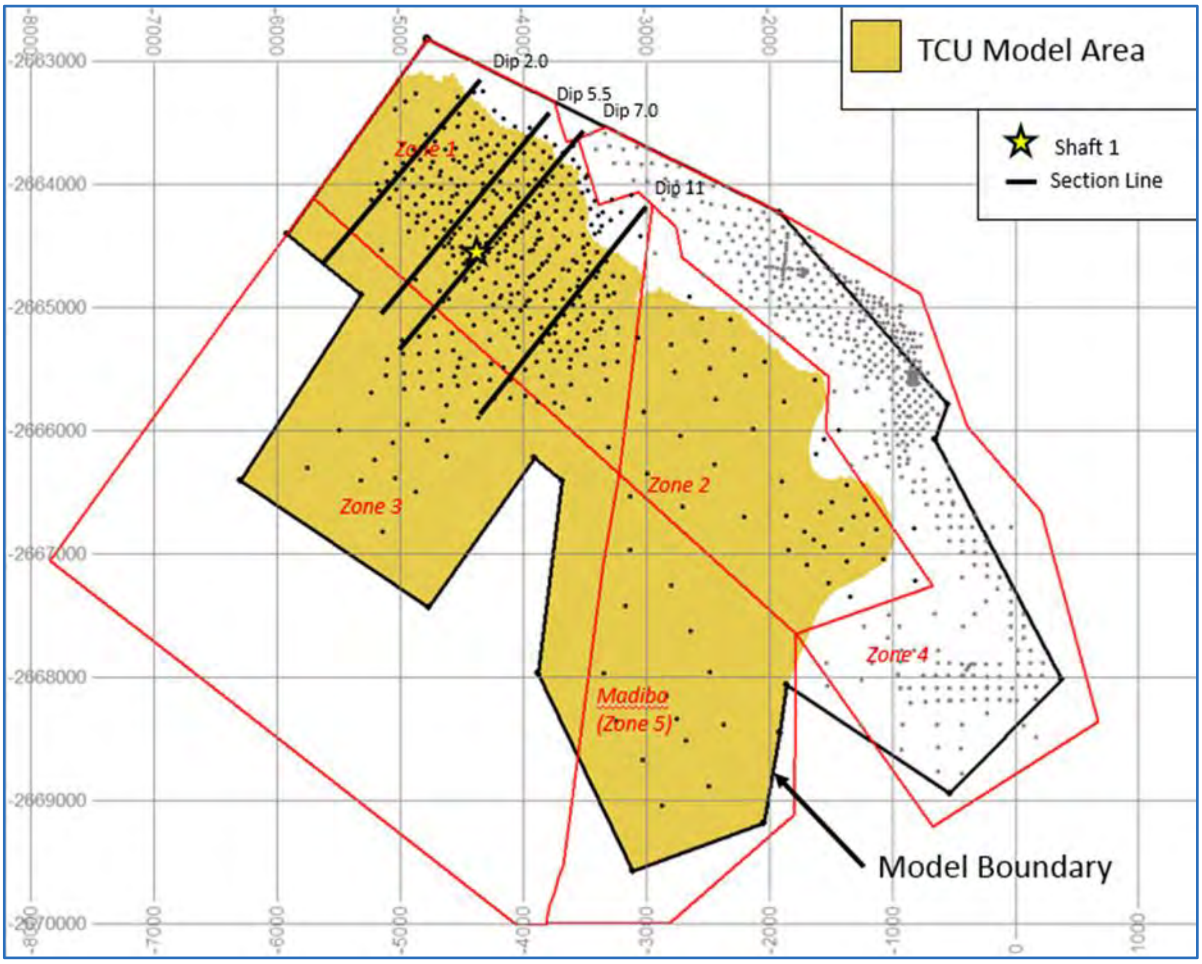
Additional drilling down-dip in Zone 3 and in the Madiba area permitted the expansion of the Inferred Mineral Resource in the UMT–TCU portion of the deposit. Additional down-dip/lateral potential could support estimation of additional Mineral Resources with further drilling. Revised geological interpretations decreased the extent of the TCU stratigraphy and decreased Inferred Mineral Resources in Zone 2.

The UMT–TCU deposit is the main focus of the Platreef Project underground mine development. The limits of the UMT-TCU model are shown in Figure 14.1. The UMT-TCU Mineral Resources are located in Zone 1, Zone 2, Zone 3 and the Madiba Area (See Section 14.2.11).

The UMT–BIK Mineral Resources are located in Zones 1 and 2. The UMT-BIK resource model is located stratigraphically above the UMT–TCU resource model (Figure 14.2).

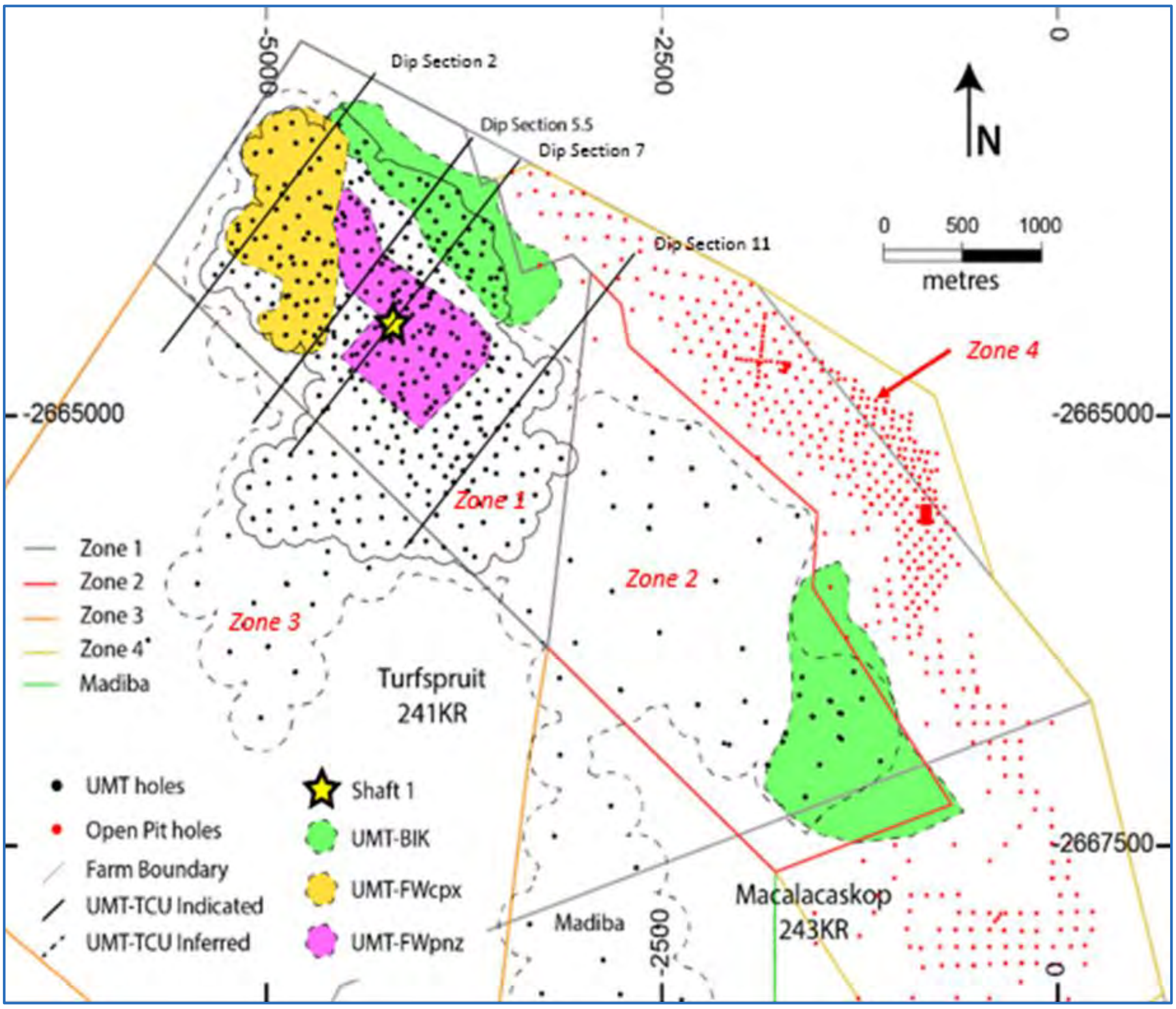
The UMT-FW model is located in Zone 1 and is situated stratigraphically below the UMT-TCU resource model (see Figure 14.2). Figure 14.3 shows the relative stratigraphic positions of the UMT-BIK, UMT-TCU and UMT-FW models.

Figure 14.1 Mineral Resource Area for the UMT-TCU



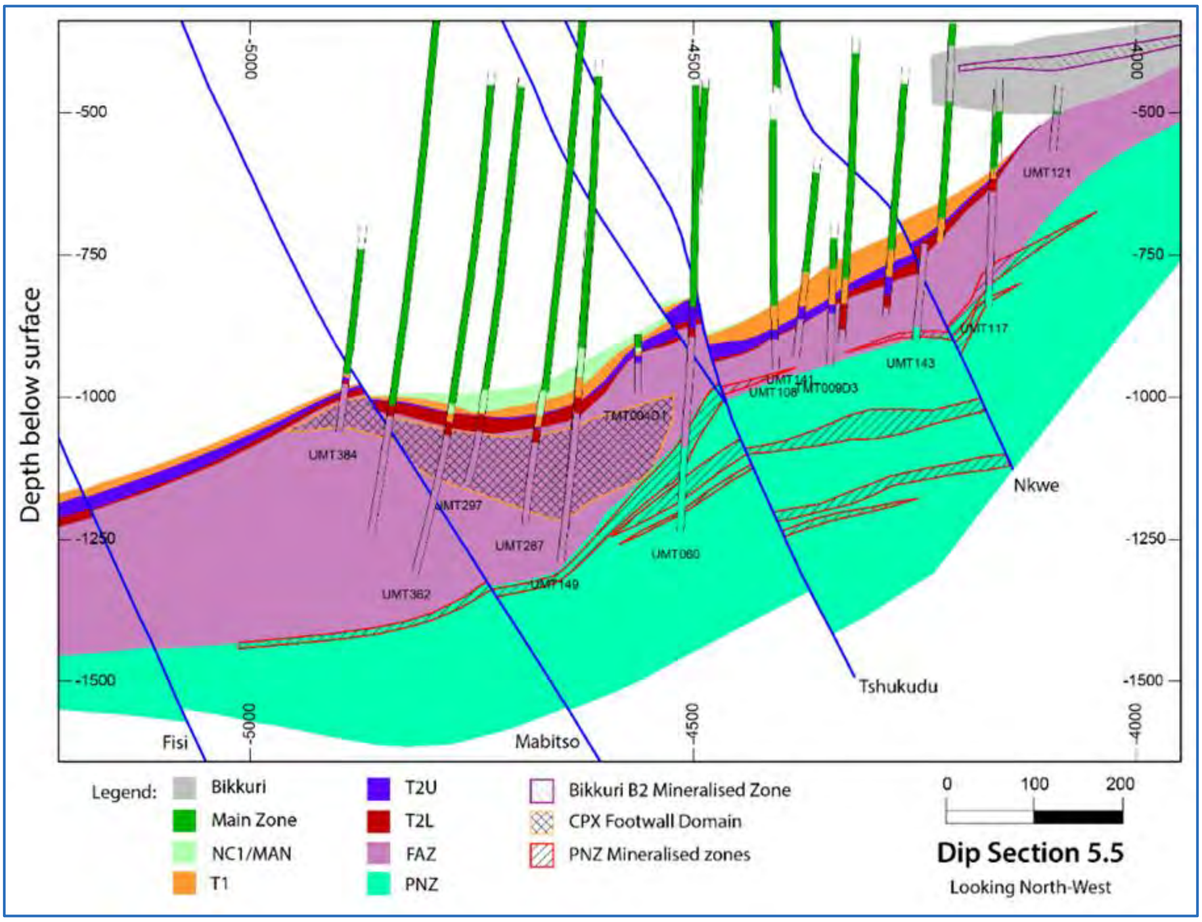
Wood, 2016

Figure 14.2 Mineral Resource Areas for the UMT-Bikkuri (BIK) and UMT-FW



Ivanhoe, 2016

Figure 14.3 Dip Section 5.5 Showing Relationship of Bikkuri, TCU and FW Models



Ivanhoe, 2016; Location of Dip 5.5 is shown on Figure 14.2.

14.1.1 Geology Model

The geology model provides a framework for Mineral Resource estimation.

The geology model for the three mineral resource models was created in Leapfrog using significant control from 2-D gridded seam models constructed in Datamine. The top surface of the T1 feldspathic pyroxenite was used as a reference surface (Figure 14.3 and Figure 14.4).

Stratigraphic units were assigned a unique integer code (MCODE). The MCODE was used to code drillhole composites and the block model. The MCODEs are summarised in Table 14.1.

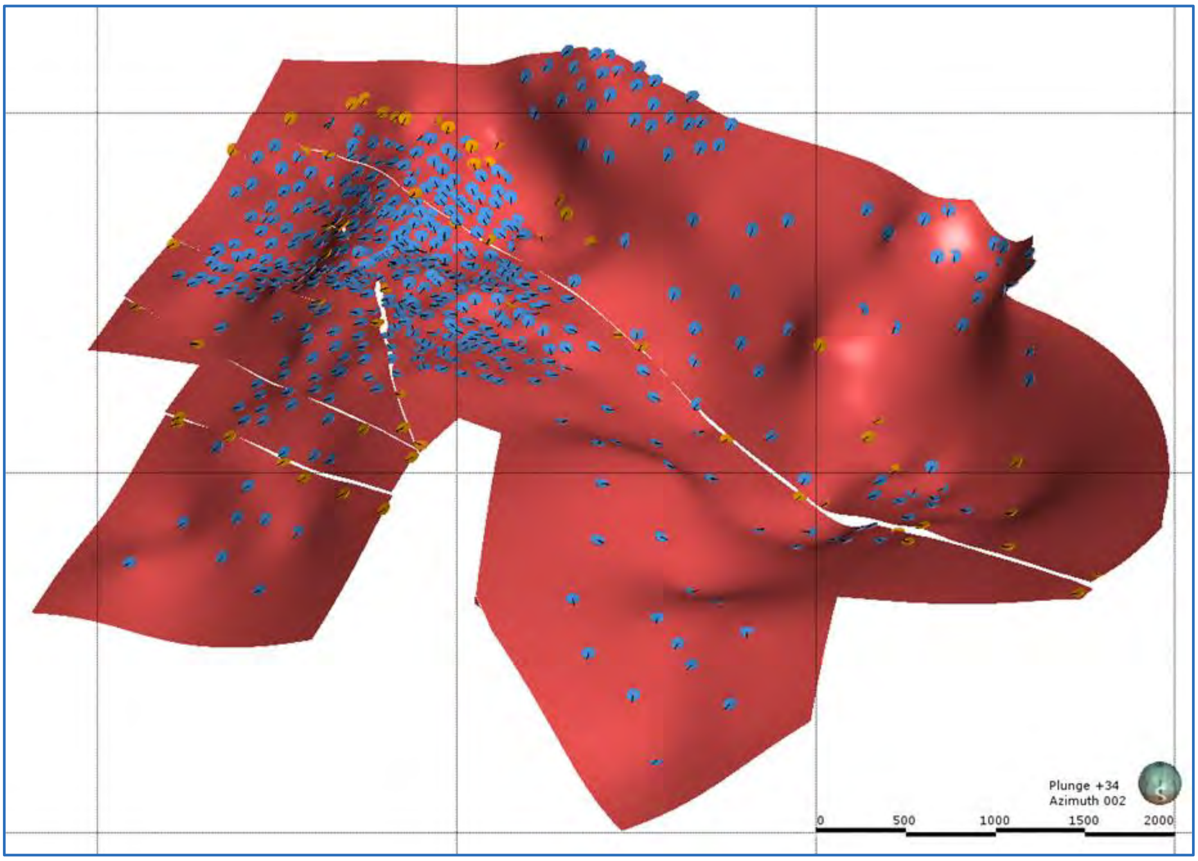
Relogging of drill core, infill drilling in Zones 1 and 2, expansion drilling in the Madiba area and a revised structural model since 2013 has resulted in modifications incorporated in the geology model.

The revised structural model identified or inferred numerous faults. Only faults with a high degree of confidence were used for the geology model. These include the Nkwe, Tshukudu, Tau, Mabitso, Fisi and Tlou faults (see Figure 7.15).

Mineralisation in the southern portions of Zone 2 that is stratigraphically above the TCU has been interpreted as similar to the Bikkuri Reef in Zone 1. Both areas are included in the UMT-BIK model (Figure 14.2).

The relogging of drill core in the footwall of the TCU identified the Footwall Assimilation Zone (FAZ) that includes the CPX (clinopyroxenite) domain and the underlying Pyroxenite-Norite Zone (PNZ). The CPX and PNZ domains were estimated separately and comprise the UMT-FW model (Figure 14.3).

Figure 14.4 T1 Reference Surface



Ivanhoe, 2016; Blue discs are drillhole intersections. Orange discs are control points. Black lines on discs show dip direction.

Table 14.1 Strat and MCODE Description

Strat Unit	Strat	MCODE
Main Zone in Bikkuri	MZBK	10
NC1 in Bikkuri area	NC1BK	11
Mottled Anorthosite in Bikkuri	MANBK	12
B1	B1	13
B2	B2	14
NC2 in Bikkuri	NC2BK	15
Lower Zone in Bikkuri	LZ1BK	16
Main Zone	MZ	20
Norite Cycles 1	NC1	21
Mottled Anorthosite	MAN	22
T1	T1	23
T2U	T2U	24
T2L	T2L	25
Norite Cycles 2	NC2	26
UG2 Hanging Wall	UG2HW	27
UG2	UG2	28
UG2 Footwall	UG2FW	29
Footwall Assimilated Zone	FAZ	30
Pyroxenite Norite Zone	PNZ	31
Lower Zone	LZ	32
FW	FW	33

14.1.2 Mineralised Zones

Nested grade shells were used to constrain the grade estimation in the UMT-TCU and UMT-BIK resource models. Nested grade shells were made for the TCU T1 mineralised zone (T1MZ) and T2 mineralised zone (T2MZ). Nested grade shells in the BIK model were identified for the B1 (B1MZ) and B2 (B2MZ) stratigraphic units. The nested grade shells were constructed using 1 g/t, 2 g/t and 3 g/t 3PE+Au. Mineral Zones (MZ) were identified in the FW-PNZ domain and were used to constrain higher grade mineralisation. The grade shells and mineral zones were validated on cross-sections to ensure consistency.

14.2 UMT-TCU Resource Model

The UMT-TCU model is the main focus of the Platreef Project and is considered amenable to selective underground mining methods. The UMT–TCU resource model update was limited to that portion of the UMT area that includes the TCU stratigraphic sequence. The limits of the UMT–TCU area Mineral Resource estimate are shown in Figure 14.1. The UMT–TCU model includes a densely-drilled area in Zone 1 and less densely drilled areas in Zones 2, and 3 and the Madiba area.

In the discussion which follows, some tables include information for the UMT-BIK and UMT-FW models so as to avoid the need for duplicate presentation of model parameters.

14.2.1 Drillhole Data

Only valid drillholes from the UMT drill programme were used for the grade estimation of the UMT-TCU mineral resource. The cut-off date for the drillhole database used for the Mineral Resource estimate was 24 July 2015.

Drillholes were considered as not valid when:

- Drillhole was abandoned prior to intersecting the mineralised zones.
- Drillhole Intersected a mineralised zone that was interpreted as not representative due to significant faulting.
- Wedge holes.

Wedges off the parent hole were commonly drilled to intersect the T1MZ and T2MZ, but locally drillholes targeted deeper footwall mineralisation. Wedges were primarily drilled for metallurgical purposes resulting in a cluster of wedges around the parent hole. An analysis was completed to determine the possibility of merging wedge holes (for wedges with available assay data) with their parent drillholes and thus to provide a single intercept per cluster of wedge holes. The analysis determined that differences in mineralised thicknesses between the wedges and parent hole caused unequal alignment of the mineralised zones and thus a smearing and smoothing of grades. The decision was taken to exclude wedge holes and only use the first intercept of the mineralised zones in any case where multiple intersections from a single parent hole occurred.

14.2.2 TCU Geology Model

The TCU Mineral Resource model occurs within the stratigraphic sequence referred to as the Turfspruit Cyclic Unit discussed in Section 7.2.7.

The geology model for the UMT-TCU was created in Leapfrog using two dimensional gridded seam models constructed in Datamine for control. The top surface of the T1 feldspathic pyroxenite was used as a reference surface (see Figure 14.4). The reference surface was used to construct the stratigraphic surfaces above and below the T1 reference surface. Control points were added near faults. Each stratigraphic unit was assigned a MCODE used in coding the drillhole composites and block model (see Table 14.1). The Nkwe, Tshukudu, Tau, Mabitso, Fisi and Tlou faults were used to divide the UMT-TCU model into seven structural blocks.

14.2.3 Model Envelope

The UMT-TCU model was constrained laterally within a polyline. The north and eastern limits were defined as the limit of recognisable TCU cyclical stratigraphy. Elsewhere the boundary was extended approximately 450 m beyond the drillhole data (see Figure 14.1). The UMT-TCU model was also constrained vertically by an envelope defined by surfaces controlled by the geological stratigraphy. The upper surface was defined as 20 m above the top of the T1MZ. The lower surface of the model envelope was defined as 75m below the base of the T2L. The UMT-TCU model does not extend above the 650 m elevation.

14.2.4 High-Grade Shells – UMT-TCU

Nested grade shells were constructed for the T1 mineralised zone (T1MZ) and the T2 mineralised zone (T2MZ) to constrain the grade estimation. The nested grade shells were identified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE (Pt+Pd+Rh) +Au cut-offs. The stratigraphic location of the mineralised intercepts was considered in the grade shell construction. The grade-shell intercepts were coded into the drillhole database. The grade-shell drillhole intercepts were validated on dip and strike sections to ensure consistency. The grade-shell drillhole intercepts were used to construct wireframes of the nested grade shells using Leapfrog and Datamine functions. Grade-shell codes (GCODES) were used to code blocks within and outside the grade shells. The GCODES are summarised in Table 14.2.

14.2.4.1 T1MZ

The T1MZ is interpreted to transgress the T1-NC1 stratigraphic boundary in the southern portions of Zone1 and into Zone 2 (see Figure 14.5). This transgression is localised and occurs in response to thickening of the T1 and NC1 units and the development of weak cyclicity within the T1. This relationship suggests the T1 is an undifferentiated portion of the cyclical units developed below the base of the Main Zone. Where the T1 and NC1 units are thinned, the T1MZ cannot be readily identified, and these areas have been excluded from the T1MZ model (see Figure 14.6).

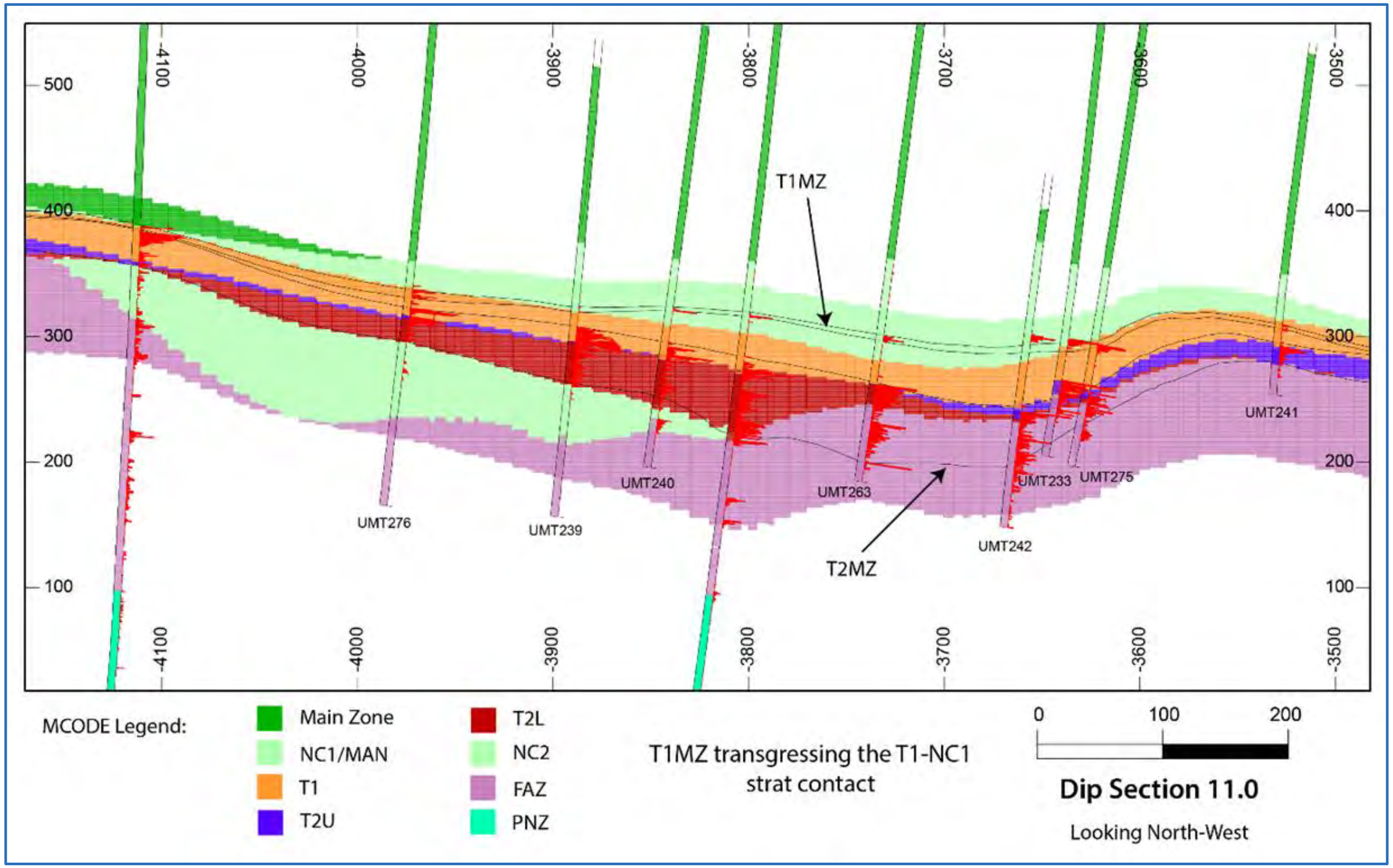
14.2.4.2 T2MZ

The T2MZ is defined by 3PE+Au assays and commonly occurs in the T2 stratigraphic unit of the TCU; however, the nested grade shells are not restricted to specific stratigraphic horizons and may transgress locally into the T1 or FW (see Figure 14.5).

14.2.5 Mineralisation Adjacent to the TCU Mineralised Zones

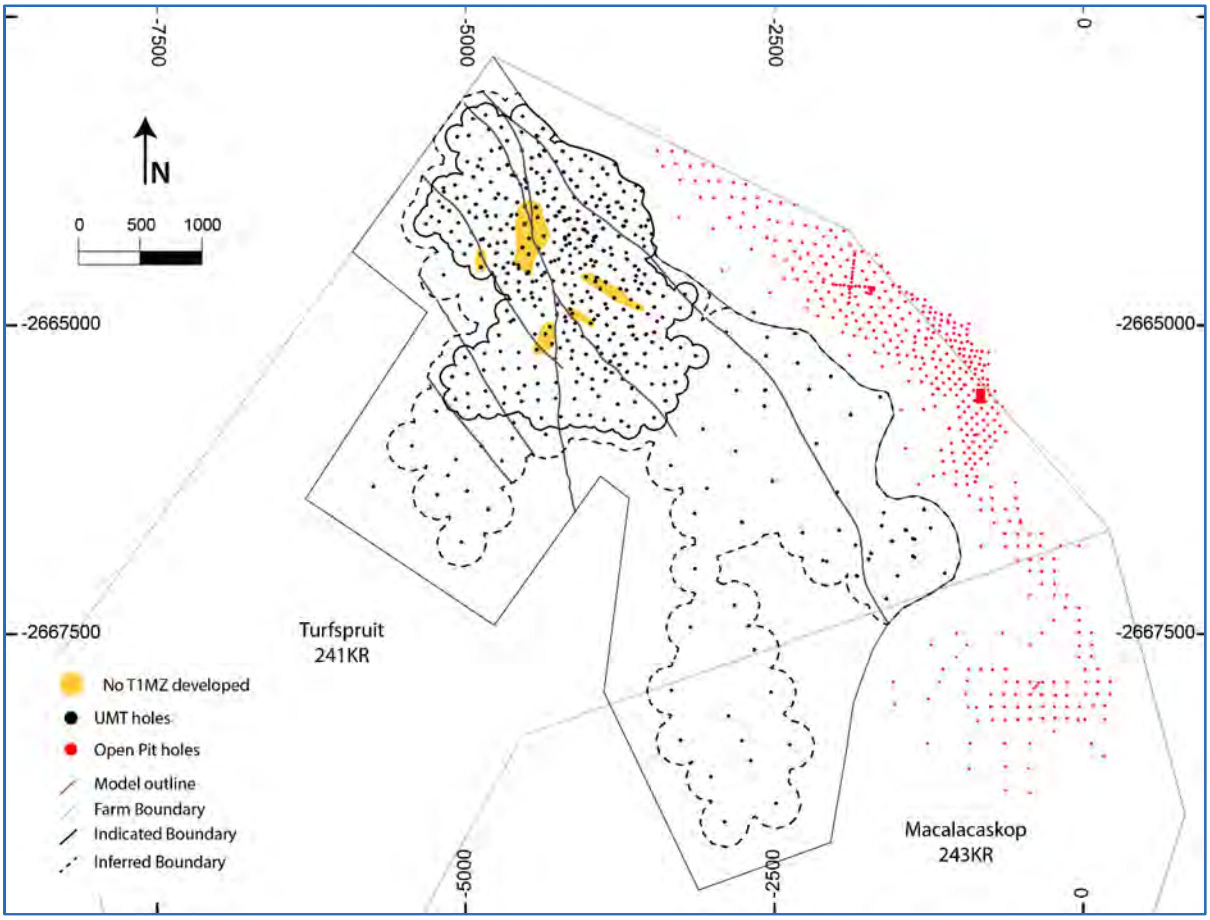
There is scattered mineralisation adjacent to the TCU mineralised zones that is locally continuous. Floating stope software is used in mining-related studies, and mineralisation adjacent to the TCU mineralised zones can be included in the resultant stopes; hence there is a need to estimate grades in blocks in an envelope around the T1MZ and T2MZ zones. Table 14.2 summarises the GCODES adjacent to the T1MZ and T2MZ mining zones within the TCU model envelope.

Figure 14.5 T1MZ Transgressing the Boundary Between the T1 and NC1 (Dip Section 11.0)



Ivanhoe, 2016; Location of Dip 11 is shown on Figure 14.2.

Figure 14.6 Areas Where the T1MZ is Not Developed



Ivanhoe, 2016

Table 14.2 Summary of GCODES For TCU and Bikkuri (All Elements)

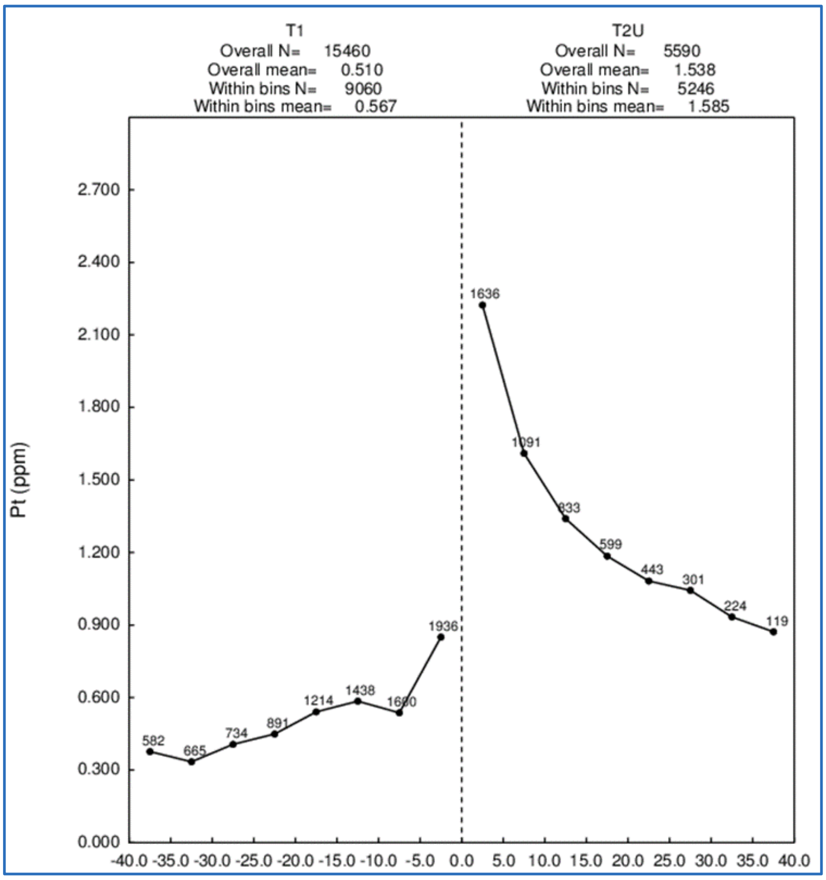
Model	Strat Unit Outside Grade Shells	Grade Shell	GCODE
TCU-BIK	NCBK, MANBK	(Outside Grade Shells)	0
		B1MZ 1g	301
		B1MZ 2g	302
		B1MZ 3g	303
	B1	(Outside Grade Shells)	1
		B2MZ 1g	401
		B2MZ 2g	402
		B2MZ 3g	403
	B2	(Outside Grade Shells)	2
	LZBK	(Outside Grade Shells)	0
UMT-TCU	MZ, NC1, MAN	(Outside Grade Shells)	0
		T1MZ 1g 3PE+Au	101
		T1MZ 2g 3PE+Au	102
		T1MZ 3g 3PE+Au	103
	T1	(Outside Grade Shells)	1
		T2MZ 1g 3PE+Au	201
		T2MZ 2g 3PE+Au	202
		T2MZ 3g 3PE+Au	203
	T2	(Outside Grade Shells)	2
	NC2, UG2HW, UG2, UG2FW, LZ1, LZ2	(Outside Grade Shells)	0

14.2.6 Compositing and Exploratory Data Analysis (EDA) for UMT–TCU Model

Valid drillholes were composited to 1 m length composites within the UMT–TCU model envelope. The compositing was controlled by the nested 3PE+Au grade shells and the TCU stratigraphic units.

EDA was completed using box plots, histograms, probability plots and contact profiles. EDA (observed discontinuities in grade profiles near contacts) suggested the grade shells and stratigraphic boundaries should be considered hard boundaries. Figure 14.7 displays the contact profile for Pt between the T1 and T2U.

Figure 14.7 Contact Profile for Platinum Between T1 and T2U



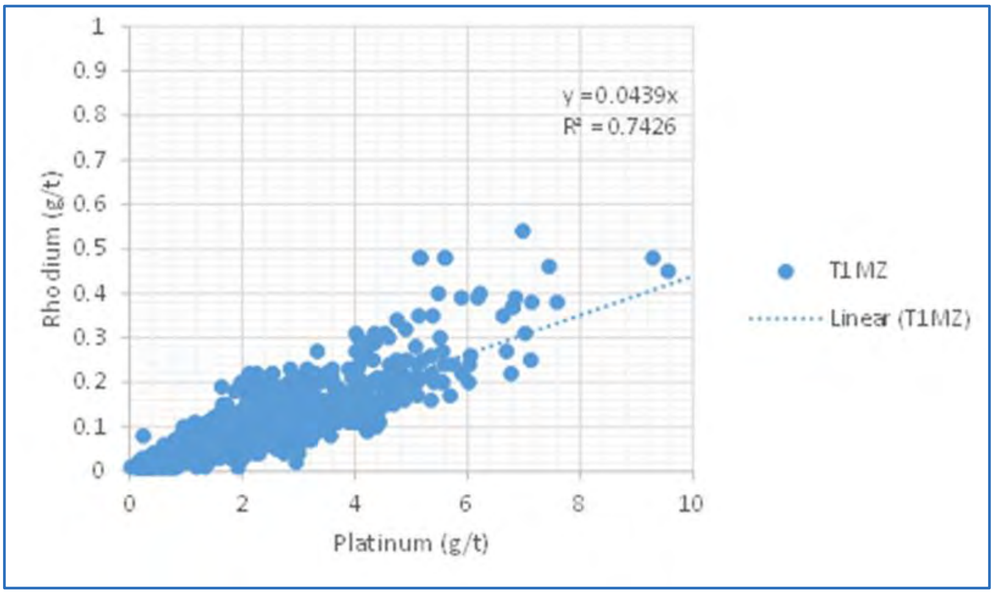
Wood, 2015

14.2.6.1 Rhodium Regressions

Rhodium analyses are available for most intercepts within the mineral zones. Rhodium to platinum regressions were constructed for samples missing rhodium analysis (Parker, 2015).

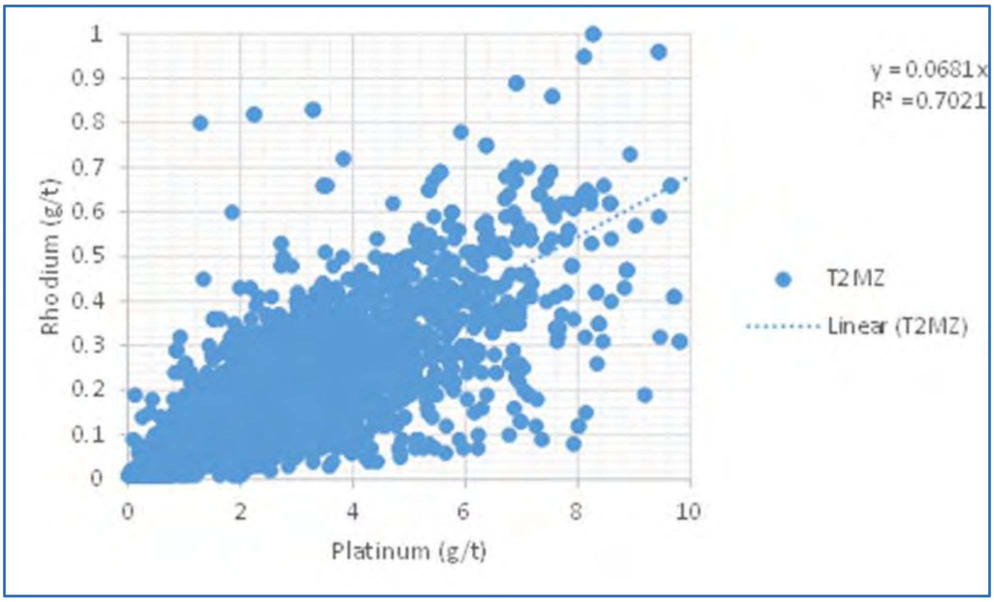
Figure 14.8 and Figure 14.9 show rhodium as a function of platinum regression for the T1 and T2 respectively. Table 14.3 summarises the proportions of assays with rhodium analysis within the grade shells and by stratigraphic unit. Table 14.4 summarises the lithology groups. The proportion of rhodium assays exceeds 50% within the 2 g/t 3PE+Au shell and exceeds 60% in the in T2U and T2L. Table 14.5 summarises the number of missing Rhodium analyses by grade shell.

Figure 14.8 Rhodium Regression for The T1



Wood, 2015

Figure 14.9 Rhodium Regression for The T2



Wood, 2015

Table 14.3 Proportions of Rhodium Assays by Strat Code and 3PE+Au Grade Shell

Inside Mineralised Zones									
Zone	Regression Equation	No. Data for Equations	Average Rh Value	Average Pt Value	Avg Rh/ Avg Pt Ratio	No. Data without Rh Values	Aver Rh Value After Eqn	Average Pt Value	Avg Rh/ Avg Pt Ratio
T1MZ	Rh=0.0439 Pt	2080	0.079	1.766	0.0445	1049	0.023	0.519	0.0439
T2MZ	Rh=0.0681 Pt	13526	0.133	1.874	0.0708	5387	0.043	0.634	0.0681
B1MZ	Rh=0.0437 Pt	47	0.052	1.163	0.0450	26	0.030	0.694	0.0437
B2MZ	Rh=0.0519 Pt	601	0.067	1.248	0.0533	281	0.028	0.534	0.0519
Outside Mineralised Zones									
Zone	Regression Equation	No. Data for Equations	Average Rh Value	Average Pt Value	Avg Rh/ Avg Pt Ratio	No. Data without Rh Values	Aver Rh Value After Eqn	Average Pt Value	Avg Rh/ Avg Pt Ratio
Group 1	Rh=0.0721 Pt	59	0.032	0.418	0.0766	4171	0.002	0.034	0.0720
Group 2	Rh=0.0585 Pt	1586	0.032	0.599	0.0534	18172	0.006	0.104	0.0585
Group 3	Rh=0.0630 Pt	582	0.047	0.619	0.0763	3440	0.015	0.232	0.0630
Group 4	Rh=0.0614	4045	0.061	1.081	0.0566	42932	0.061	0.242	0.2539
Group 5	Rh=0.1102 Pt	125	0.099	0.898	0.1102	179	0.029	0.260	0.1102
Group 6	Rh=0.0691	545	0.069	1.811	0.0382	22194	0.069	0.105	0.6571

Table 14.4 Group Definitions Table Caption

Group	Strat Unit
1	MZ, MZBK, MAN, MANBK
2	BAR, B1, T1, NCBK, NC1
3	B2 BBK, T2U, T2L, NC2
4	FAZ, FAZBK, UGHW
5	UG2, UG2FW
6	HFR, PNZ, TVL

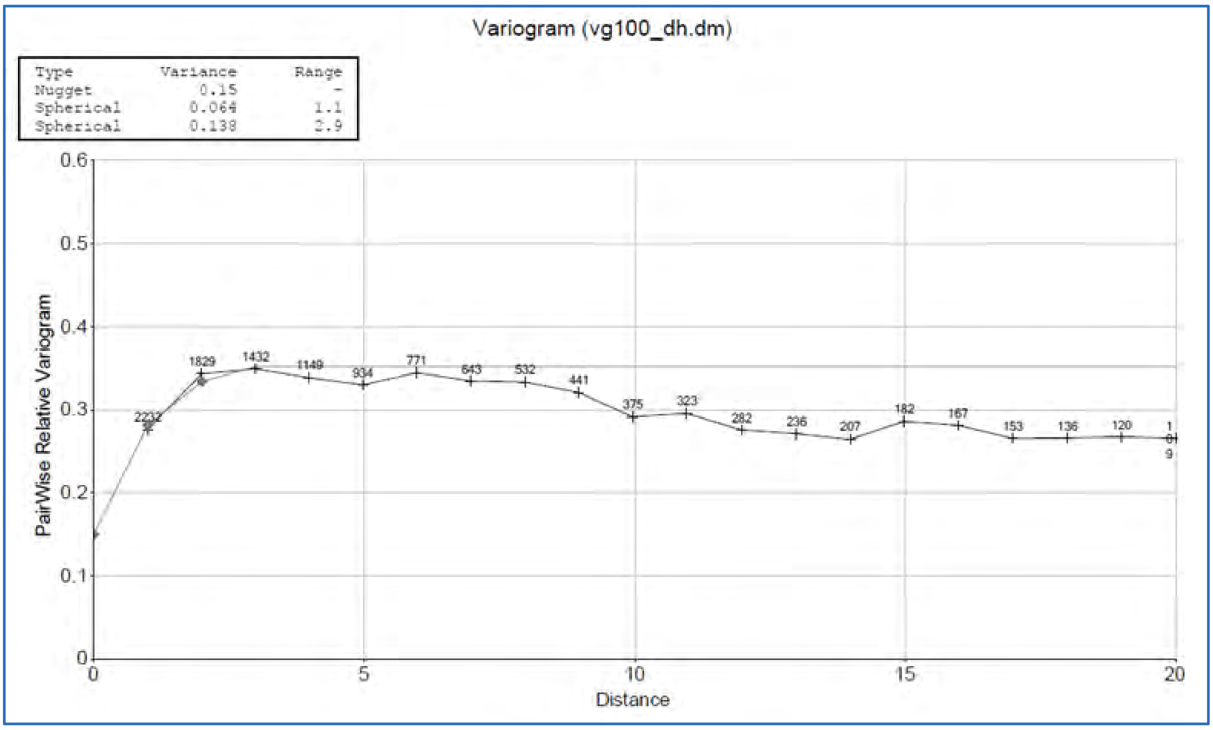
Table 14.5 Missing Rhodium Analysis by Grade Shell (All Intervals)

MinZone	Grade Shell (3PE+Au)	Number Sample Intervals	Samples Intervals with Rh Analysis	Samples Intervals Missing Rh Analysis	% Intervals Missing Rh Analysis
B1MZ	1 g/t	16	0	16	100
	2 g/t	8	0	8	100
	3 g/t	130	33	97	75
B2MZ	1 g/t	746	293	453	61
	2 g/t	231	162	69	30
	3 g/t	535	414	121	23
T1MZ	1 g/t	1,026	453	573	56
	2 g/t	303	239	64	21
	3 g/t	1,939	1,406	533	27
T2MZ	1 g/t	6,383	2,781	4,057	59
	2 g/t	5,006	3,515	1,491	30
	3 g/t	8,417	7,812	605	7

14.2.7 Variography

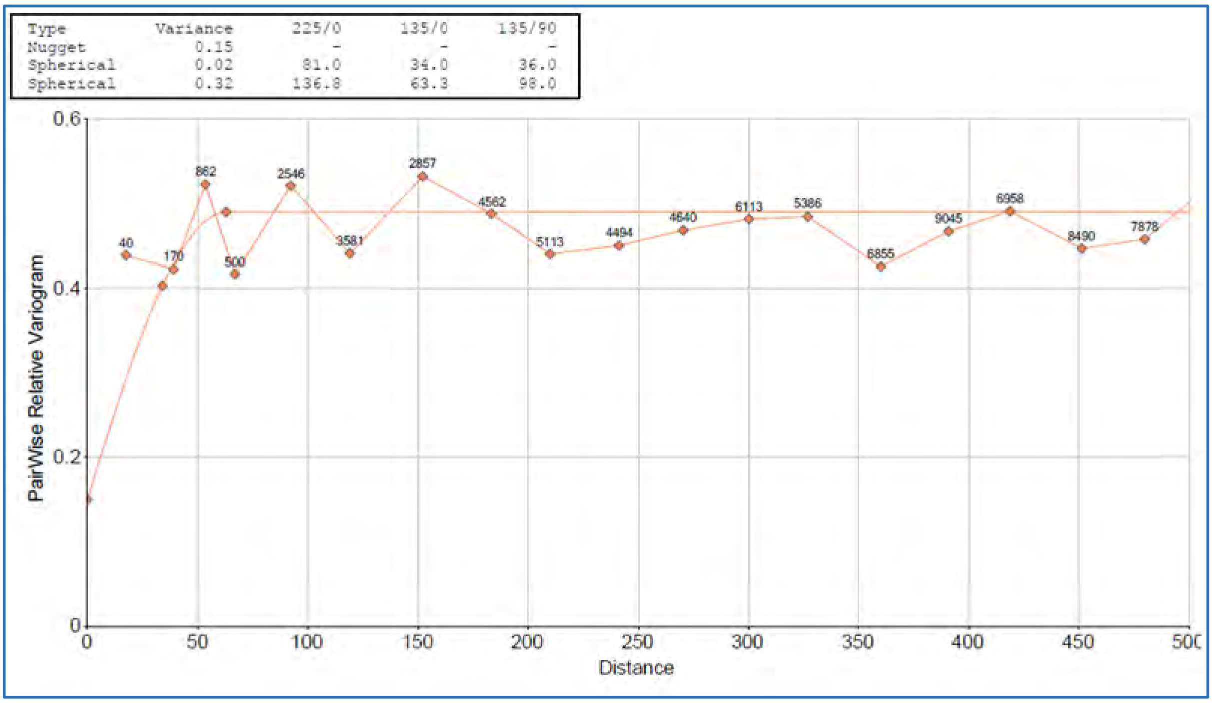
Pair wise relative variograms were completed by grade shell. Figure 14.10 shows a downhole pairwise relative variogram for Platinum. Figure 14.11 shows a directional pairwise relative variogram for Platinum (aziumuth 135, dip 0).

Figure 14.10 Downhole Pair Wise Relative Variogram for Platinum



Wood, 2015; Lag distances are in metres.

Figure 14.11 Directional Pair Wise Relative Variogram Model for Platinum (Az=135)



Wood, 2015; Lag distances are in metres.

14.2.8 Block Model

The UMT–TCU block model was constructed over the area of UMT drilling where the TCU has been recognised and correlated (Figure 14.1). Blocks were oriented parallel to the national coordinate system.

The block model used a parent block size of 10 m x 10 m x 2 m. Sub-celling was 5 m x 5 m x 0.5 m. The block model parameters are summarised in Table 14.6. The geological stratigraphic units and 3PE+Au grade shells were coded to the blocks and used to control the grade estimation.

After estimation, the final resource model blocks were regularised to 10 x 10 x 2 m to soften the hard boundaries used in the grade estimation. Because of the limited thickness of the T1MZ, blocks in the T1MZ were regularised to 10 x 10 x 1 m to avoid excessive dilution on the contacts.

Table 14.6 Block Model Parameters

Axis	Origin	Maximum	Block Size	No Blocks
Easting (X)	-6,400	400	10	680
Northing (Y)	-2,669,600	-2,662,800	10	680
Elevation (Z)	-850	-50	2	800

14.2.9 Block Grade Estimation

To eliminate the effects of the structural blocks and variability in elevation, the individual stratigraphic units and mineralised zones were transformed to the 1,000 m elevation.

The zones were hung at the centre of the stratigraphic units or mineralised zone with the exception of the T2MZ. The mineralisation in the T2MZ is commonly top-loaded and the 1 g/t, 2 g/t and 3 g/t 3PE+Au grade shells were individually transformed to hang from the top of the zone to preserve the grade profile. After grade estimation, all blocks and drillhole composites were back-transformed to the original elevation.

Grades were estimated for Pt, Pd, Rh, Au, Cu, Ni and S using inverse distance weighting to the third power (ID3) and ordinary kriging (OK). Nearest neighbour (NN) and OK grade estimates were completed for validation purposes.

Estimations were completed in Datamine using expanding search volumes summarised in Table 14.7.

Table 14.7 Estimation Parameters

Search Pass	Axis	Azimuth	Dip	Search Range	Min. Samples	Max. Samples	Max. per Drillhole
1	X	90	0	250	4	15	3
	Y	0	0	250	4	15	3
	Z	0	90	10	4	15	3
2	X	90	0	500	4	15	3
	Y	0	0	500	4	15	3
	Z	0	90	20	4	15	3
3	X	90	0	2,500	1	15	3
	Y	0	0	2,500	1	15	3
	Z	0	90	2,500	1	15	3

Samples are 1 m composites.

14.2.9.1 T2MZ

The grade estimation in the T2MZ included block and drillhole composite matching by a combination of MCODE and GCODE to ensure the stratigraphic components of the T2MZ were estimated separately.

14.2.9.2 Blocks Outside Grade Shells

Grade estimation for blocks not located within the nested grade shells were estimated by matching blocks and composites by MCODE.

14.2.9.3 Grade Capping and Outlier Restriction

An outlier restriction distance threshold of 15 m was applied to high-grade composites within each stratigraphic unit and mineralised zone. The grade thresholds for outliers were selected from the histograms and probability plots of 1 m drillhole composites; thresholds are summarised in Table 14.8 and Table 14.9. Composites with grades above the grade threshold and with distances from composite to block centre beyond the distance thresholds were not used in grade estimation.

Table 14.8 Outlier Restriction Thresholds for Stratigraphic Units (MCODE)Table Caption

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZ	0.40	0.50	0.10	0.05	0.22	0.30	2.0
NC1/MAN	0.60	1.00	0.25	0.05	0.40	0.25	1.0
T1	2.00	1.20	0.60	0.06	0.50	0.35	2.5
T2	1.50	2.20	0.35	0.10	0.50	0.35	2.0
NC2	1.50	1.10	0.30	0.10	0.30	0.20	-
UG2HW	2.00	1.20	0.40	0.15	0.35	-	-
UG2	-	-	-	-	-	-	-
UG2FW	-	-	-	-	-	-	-
FAZ	1.60	2.50	0.60	0.15	1.00	0.60	5.0
PNZ	0.25	0.50	0.06	0.15	1.00	0.65	10.0

Table 14.9 Outlier Restriction Thresholds for Mineralised Zones (GCODE)

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
T1MZ	7.5	10.0	-	-	-	-	5.0
T2MZ 1g/t	4.0	4.0	0.7	0.25	1.0	0.5	4.0
T2MZ 2g/t	5.0	10.0	-	-	-	-	-
T2MZ 3g/t	9.0	10.0	2.0	-	-	-	4.0
B1MZ	-	-	-	-	-	-	-
B2MZ	-	-	-	-	-	-	-

14.2.9.4 Unestimated Blocks

Blocks that were not estimated were assigned a default grade determined as the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.10. Unestimated blocks were generally located at the extremities of the block model. Unestimated blocks within the FW stratigraphy were found to be located in areas of wide-spaced drilling and were not assigned an average grade.

Table 14.10 TCU Mean Grades by Stratigraphic Unit

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZ	0.001	0.001	0.001	0.001	0.010	0.010	0.010
NC1	0.056	0.051	0.034	0.003	0.049	0.015	0.096
MAN	0.041	0.034	0.014	0.003	0.012	0.006	0.047
T1	0.212	0.160	0.073	0.012	0.103	0.041	0.244
T2	0.344	0.432	0.066	0.023	0.164	0.078	0.423
NC2	0.261	0.255	0.046	0.017	0.097	0.042	0.227
UG2HW	0.796	0.798	0.13	0.078	0.160	0.790	0.397
UG2	1.229	0.661	0.078	0.161	0.236	0.135	0.680
UG2FW	0.390	0.460	0.071	0.040	0.195	0.118	0.704
LZ1	0.341	0.419	0.067	0.061	0.146	0.080	0.533
LZ2	0.195	0.250	0.040	0.069	0.106	0.061	0.474
T1MZ 1g	0.660	0.571	0.202	0.039	0.255	0.156	0.837
T1MZ 2g	1.088	0.947	0.309	0.057	0.255	0.156	0.837
T1MZ 3g	1.709	1.387	0.420	0.092	0.358	0.166	0.887
T2MZ 1g	0.670	0.765	0.118	0.059	0.225	0.115	0.667
T2MZ 2g	1.059	1.187	0.177	0.083	0.278	0.141	0.786
T2MZ 3g	2.150	2.258	0.322	0.156	0.383	0.190	0.988

14.2.10 Bulk Density

Bulk density was assigned to stratigraphic units using the mean density for each unit (Table 14.11). Whilst some stratigraphic units are comprised of a number of different lithologies (the NC1 and NC2 cyclical units for example), in general, the variability in density values is considered low.

Table 14.11 Density Values by Stratigraphic Code

Zone	Number of Samples	Mean	CV
MZ	17,368	2.90	0.03
NC1/MAN	1,184	2.95	0.05
T1	2,387	3.18	0.03
T2U	787	3.19	0.04
T2L	793	3.04	0.05
NC2	292	3.05	0.06
UG2HW	43	3.10	0.08
UG2	3	3.50	0.01
UG2FW	33	3.17	0.04
FAZ	6,453	3.11	0.05
PNZ	4,001	3.09	0.06
LZ	534	3.14	0.07

14.2.11 Mineral Resource Classification

Mineral Resources have been classified using the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014):

"A Mineral Resource is a concentration or occurrence of solid material, of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify, geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail, to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve."

Classification is determined both laterally and vertically. A drillhole spacing study was completed in 2013 (Parker and Kuhl, 2013) to review the classification parameters used at Platreef. The study was based on the 2013 structural and geology models. The study concluded that the existing 100 m drill grid required for Indicated in the 2013 resource could be expanded to 150 m, but reduced locally in areas showing high variability in grade, uncertain geometry of the mineralisation or the position of the mineralisation. Drilling conducted in 2014 and 2015 was planned on an offset 200 m grid resulting in a 150 m spacing between holes.

Early drilling in the UMT programme extended well below the T2MZ. Later infill drill programmes were focused on the TCU stratigraphy and were completed 20 to 50 m into the FW. This results in a wider drillhole spacing below the TCU.

A triangulated surface was constructed to define the boundary from a drillhole spacing suitable for Indicated Mineral Resources to the wider drillhole spacing below the TCU suitable for Inferred Mineral Resources. This surface was used to define a boundary between Indicated and Inferred Mineral Resources. A similar methodology was applied in the Madiba area where the drillhole spacing below the T2MZ is too wide to support Inferred Mineral Resources and the block model is unclassified.

14.2.12 UMT–TCU Model Classification

The Mineral Resource Classification for the TCU model is shown in Figure 14.12. No Measured Mineral Resources are declared. Indicated Mineral Resources are declared where closer spaced drilling has been completed (Predominantly Zone 1). Inferred Mineral Resources are declared where the drillhole spacing is 400 m to 800 m (predominately Zone 2, Zone 3 and Madiba area). Inferred Mineral Resources are also declared in Zone 1 below the TCU where drillhole spacing increases. The Inferred Mineral Resources are permitted at a wider drillhole spacing than would normally apply because of the well defined geology of the TCU. Figure 14.13 displays the regions of Indicated and Inferred Mineral Resources on Dip Section 7.0.

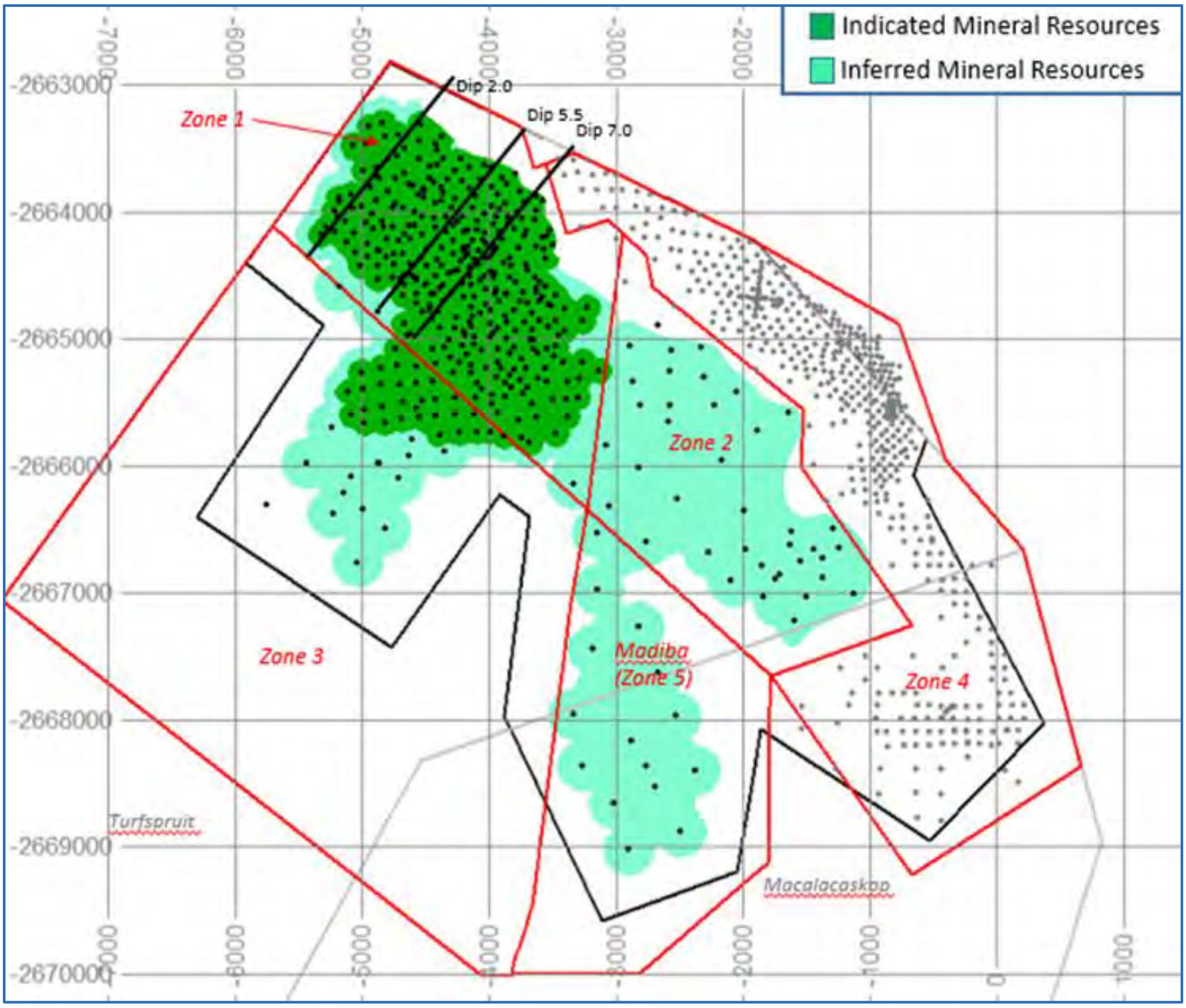
14.2.13 UMT–TCU Model Validation

Model validation included blocks classified as Indicated Mineral Resources and included visual inspection of block grades relative to composite grades on cross-sections and level plans. Statistical comparisons consisting of box plots and grade profiles tabulated in different directions (swaths) for each metal by stratigraphic unit and 3PE+Au grade shell were constructed to compare the ID3 grade estimates, OK estimate, NN estimates, and 1 m composites.

14.2.13.1 Visual Inspection

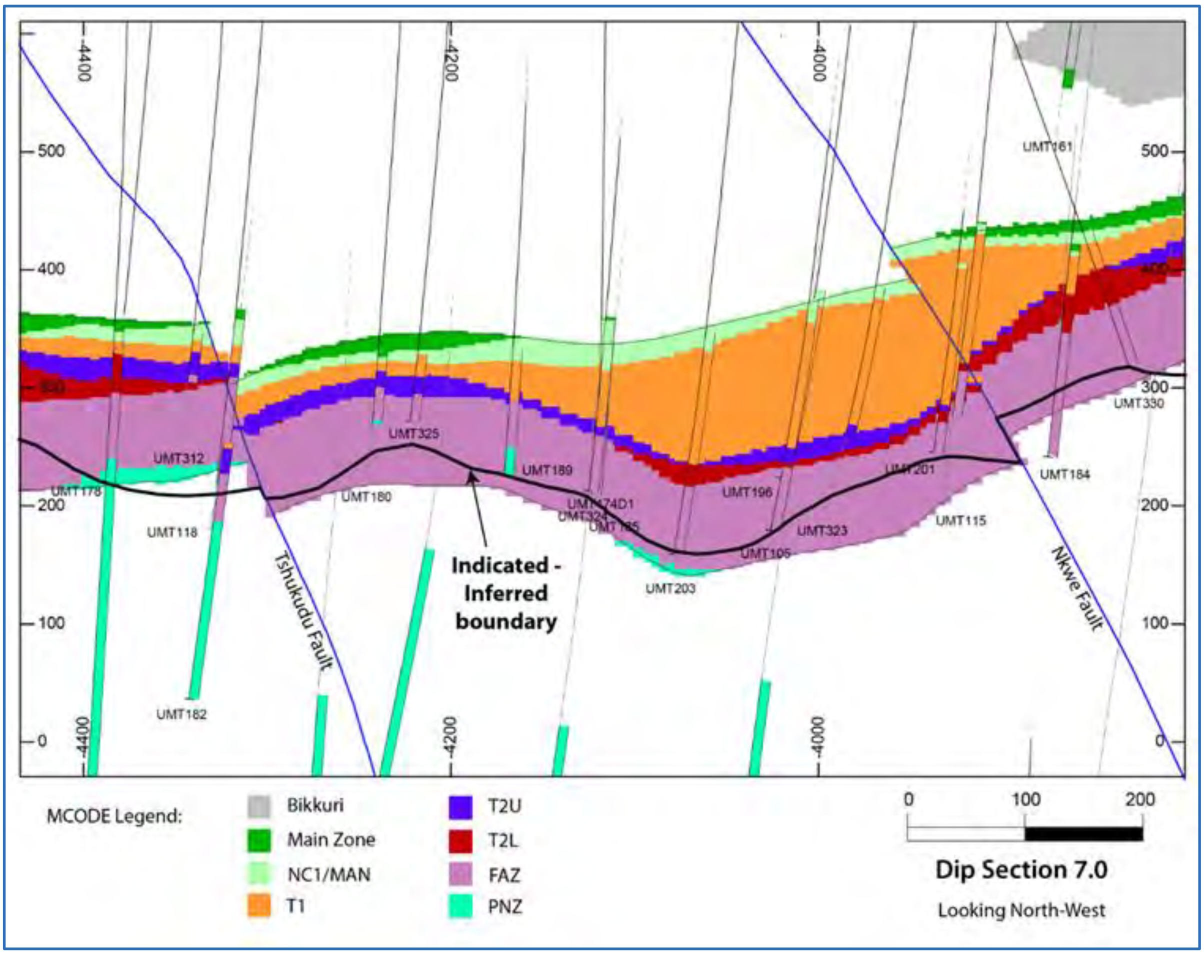
Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long-sections and level plans. In general, the block grades honoured the composite grades. Representative cross-sections for 3PE+Au are presented in Figure 14.14 (Dip Section 7.0) and Figure 14.16 (Dip Section 2.0). Representative cross-sections for Ni are presented in Figure 14.15 (Dip Section 7.0) and Figure 14.17 (Dip Section 2.0).

Figure 14.12 Mineral Resource Classification for The TCU Model



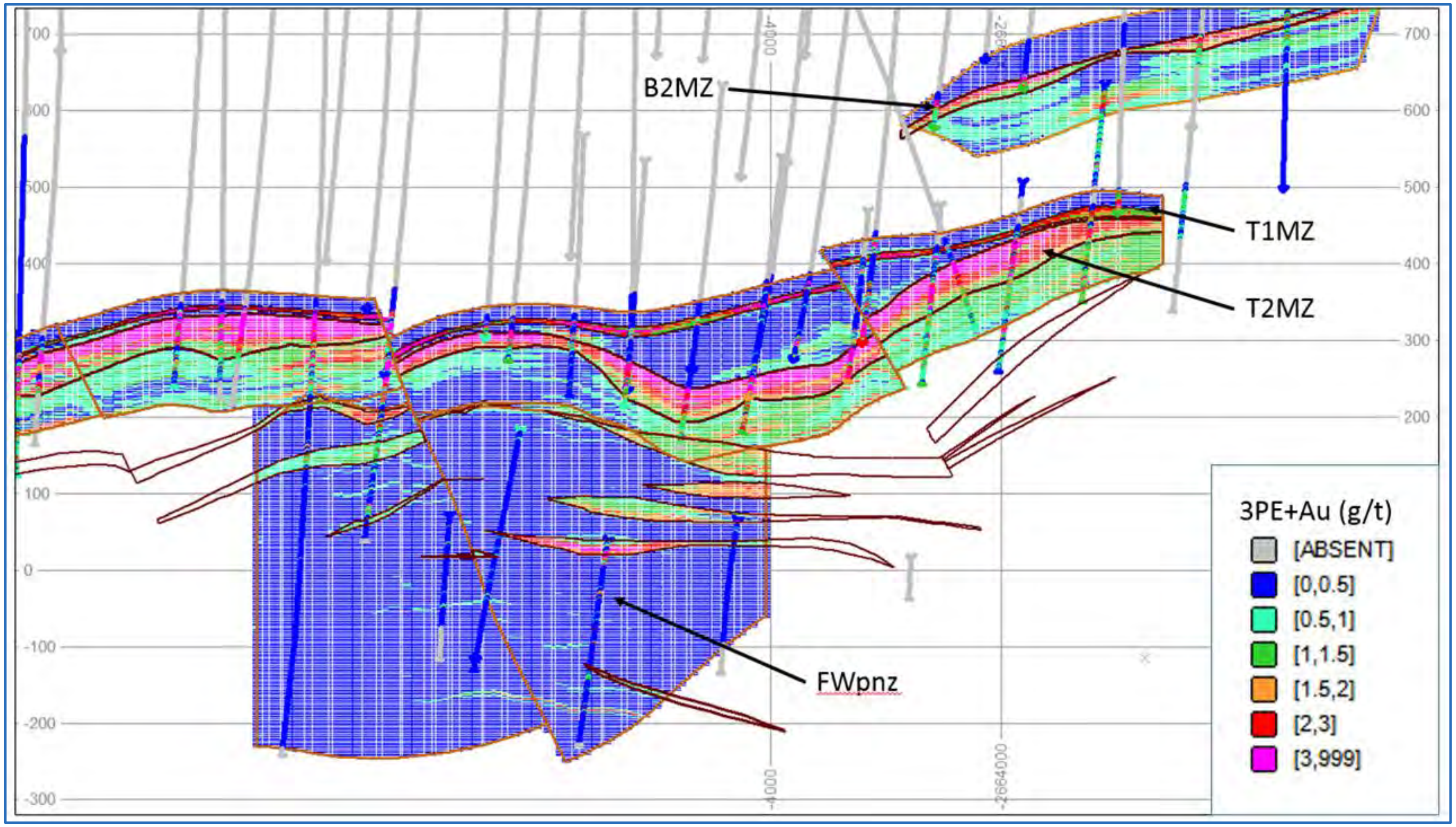
Wood, 2016

Figure 14.13 Dip Section 7.0; Constraints on Mineral Resource Classification



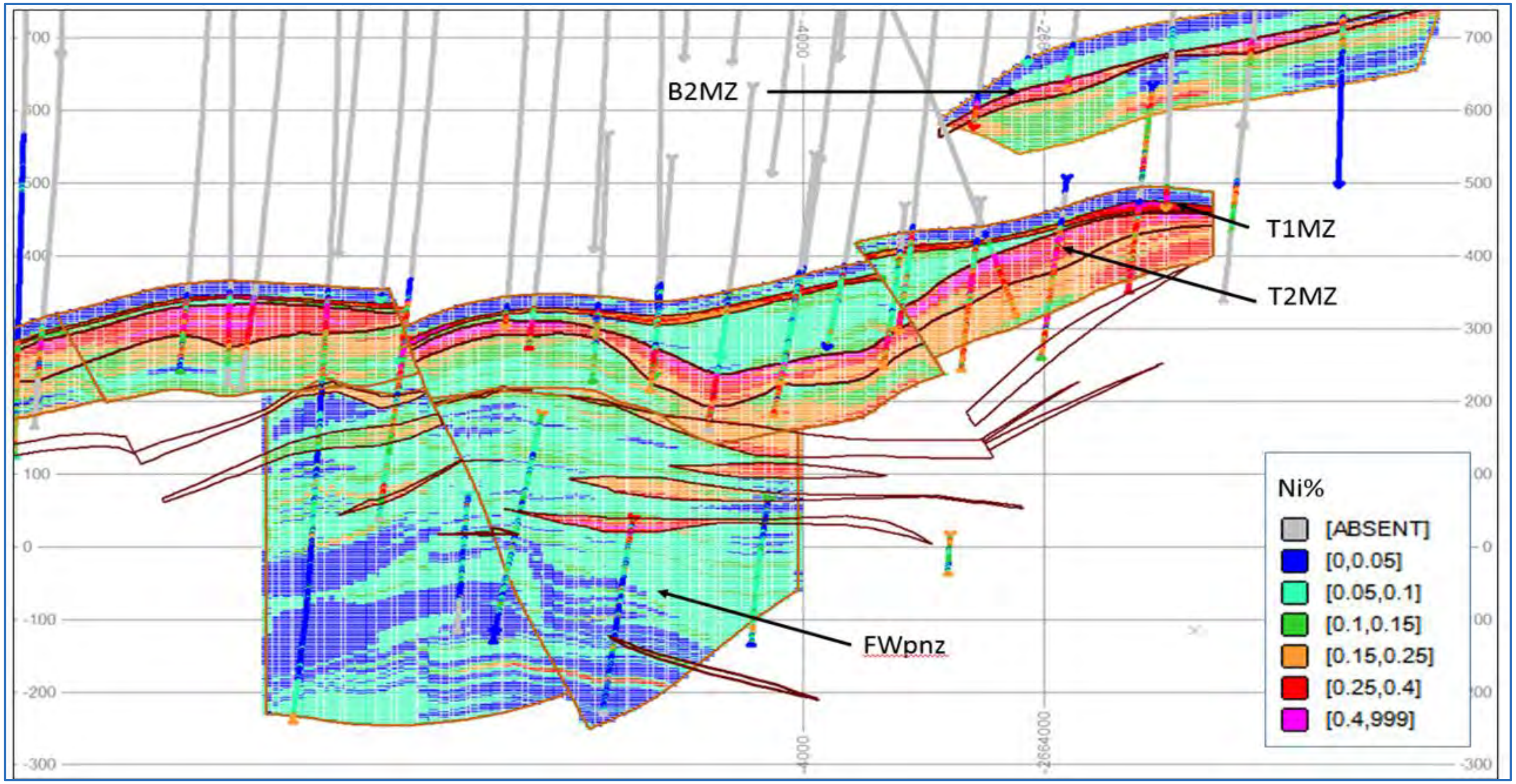
Ivanhoe, 2016; Location of Dip 7 is shown on Figure 14.2.

Figure 14.14 Dip Section 7.0 Showing 3PE+Au (50 m Drillhole Projection) Looking North-West



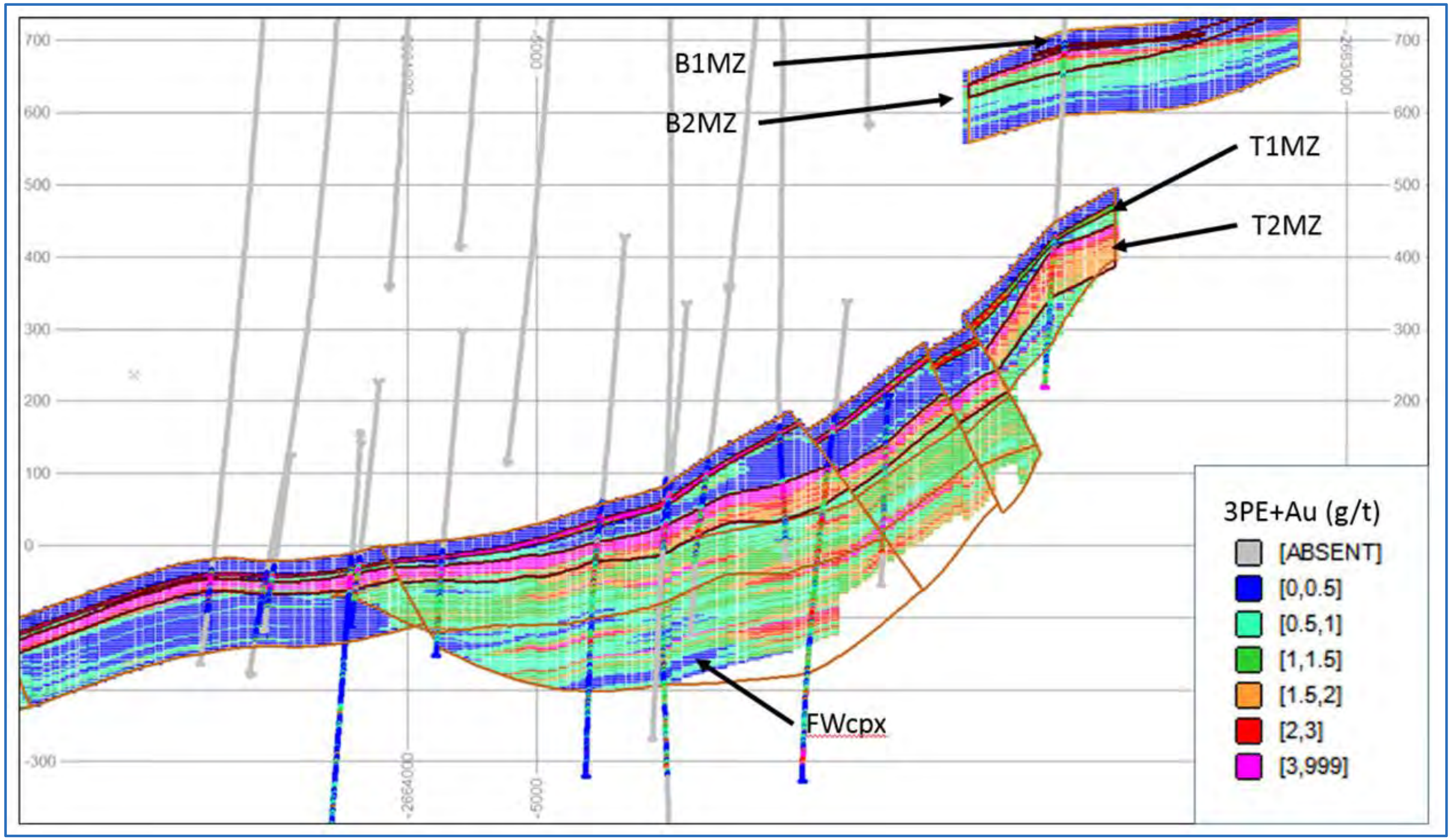
Wood, 2016; Location of Dip 7 is shown on Figure 14.2.

Figure 14.15 Dip Section 7.0 Showing Ni% (50 m Drillhole Projection) Looking North-West



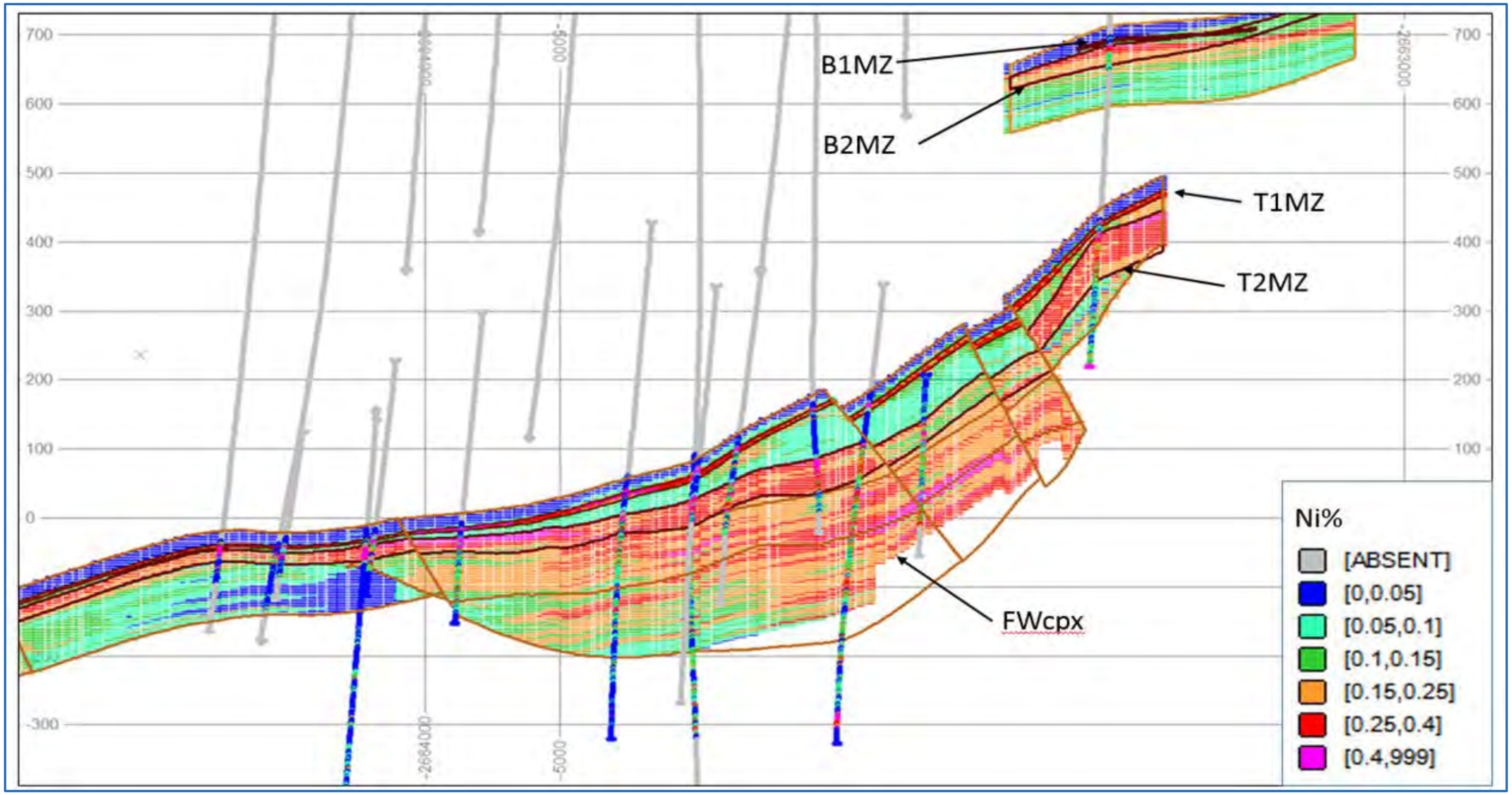
Wood, 2016; Location of Dip 7 is shown on Figure 14.2.

Figure 14.16 Dip Section 2.0 Showing 3PE+Au (50 m Drillhole Projection) Looking North-West



Wood, 2016; Location of Dip 2 is shown on Figure 14.2.

Figure 14.17 Dip Section 2.0 Showing Ni% (50 m Drillhole Projection) Looking North-West



Wood, 2016; Location of Dip 2 is shown on Figure 14.2.

14.2.13.2 Global Bias Check

A global bias check compared the Nearest Neighbour (NN) model to the ID3 model. A NN model represents the declustered composite distribution as, when correctly implemented, it is statistically unbiased to aid in the validation of grade estimates. Mr Kuhl checked the Pt, Pd, Au, Ni, Cu and S resource models for global and local bias.

The checks for global bias were performed by comparing the ID3 average grade (with no cut-off) with NN estimates by mineralised unit. Blocks reviewed were restricted to those classified as Indicated Resources. Domains with a global bias outside guidelines of ±5% (relative) are highlighted (see Table 14.12).

The grade shells for the T1MZ and T2MZ are within the guideline for all elements.

14.2.13.3 Box Plots – Sub-Cell Model

Box plots were completed for the sub-celled model for each element comparing the 1 m composites, NN, ID3 and OK estimates by mineralised unit. Block selection was restricted to those classified as Indicated.

Figure 14.18 displays the box plots for Pt within the T2MZ 2 g/t 3PE+Au.

The sub-celled and regularised models were also compared (Figure 14.19).

The box plots for the UMT-TCU model show good agreement between 1 m composites, ID3, NN and OK grade.

14.2.13.4 Swath Plots

Swath plots (width of 200 m) of the ID3 model, NN model and 1 m composites were completed for Pt, Pd, Au, Rh, Cu and Ni.

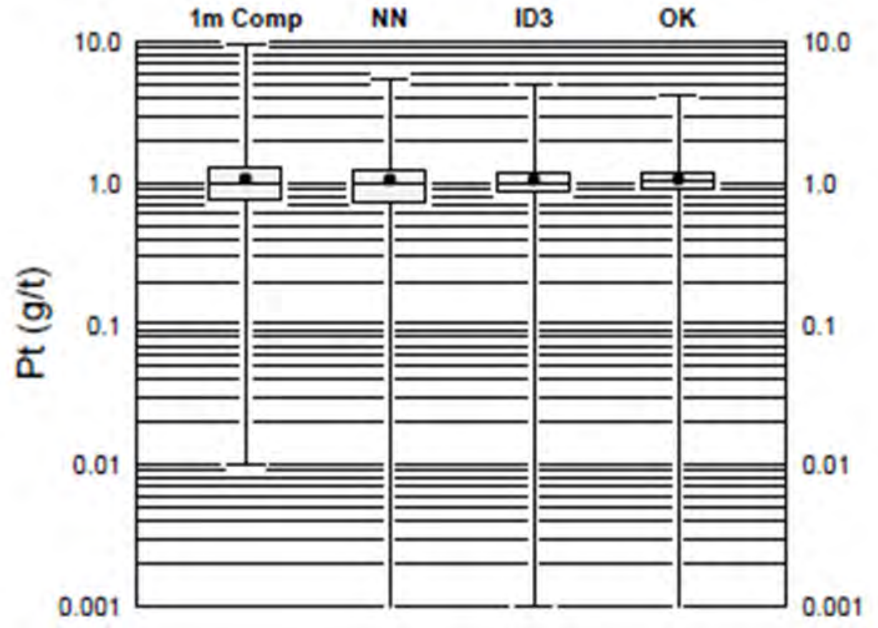
Overall, swath plots display reasonable comparisons between the ID3 estimates and their respective NN estimates; however, locally there are some differences, particularly in areas with limited drilling.

The platinum swath plot for the T1MZ 2 g/t 3PE+Au shell is presented in Figure 14.20.

Table 14.12 TCU Model Global Bias Check for Pt

GCODE	Element	NN	ID3	Relative Difference
101	Pt	0.618	0.604	-2.2%
	Pd	0.489	0.477	-2.4%
	Au	0.210	0.209	-0.7%
	Rh	0.028	0.027	-1.8%
	Ni	0.205	0.201	-1.5%
	Cu	0.107	0.105	-2.3%
102	Pt	1.139	1.155	1.4%
	Pd	0.897	0.901	0.4%
	Au	0.311	0.318	2.2%
	Rh	0.048	0.049	1.0%
	Ni	0.267	0.265	-0.7%
	Cu	0.142	0.141	-0.7%
103	Pt	2.389	2.428	1.6%
	Pd	1.981	2.025	2.2%
	Au	0.558	0.561	0.4%
	Rh	0.109	0.112	2.5%
	Ni	0.366	0.368	0.6%
	Cu	0.189	0.190	0.6%
201	Pt	0.663	0.657	-0.9%
	Pd	0.772	0.765	-0.9%
	Au	0.116	0.114	-1.6%
	Rh	0.046	0.046	-0.8%
	Ni	0.221	0.220	-0.5%
	Cu	0.110	0.110	-0.4%
202	Pt	1.045	1.045	0.0%
	Pd	1.168	1.166	-0.2%
	Au	0.177	0.175	-1.4%
	Rh	0.073	0.073	0.2%
	Ni	0.272	0.274	0.6%
	Cu	0.134	0.136	1.0%
203	Pt	2.295	2.295	0.0%
	Pd	2.326	2.336	0.4%
	Au	0.341	0.343	0.4%
	Rh	0.158	0.159	0.5%
	Ni	0.376	0.375	-0.3%
	Cu	0.187	0.187	-0.3%

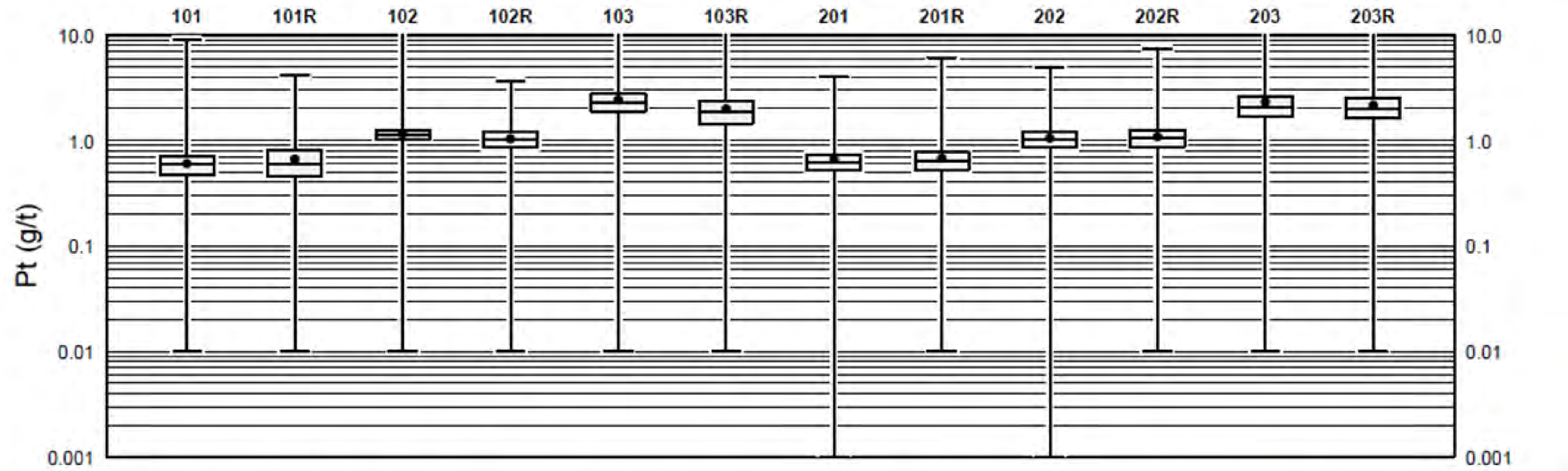
Figure 14.18 Box Plot of Pt for T2MZ 3PE+Au



	1m Comp	NN	ID3	OK	
No. of data	4029	1007591	1007591	1007591	No. of data
Mean	1.068	1.052	1.052	1.056	Mean
Maximum	9.670	5.379	4.870	4.254	Maximum
U.Quartile	1.260	1.246	1.190	1.176	U.Quartile
Median	0.990	0.990	1.006	1.017	Median
L.Quartile	0.747	0.740	0.858	0.884	L.Quartile
Minimum	0.010	0.000	0.001	0.000	Minimum
Variance	0.346	0.280	0.117	0.078	Variance
CV	0.550	0.503	0.325	0.265	CV
Skewness	2.867	1.605	1.266	0.978	Skewness

Wood, 2015

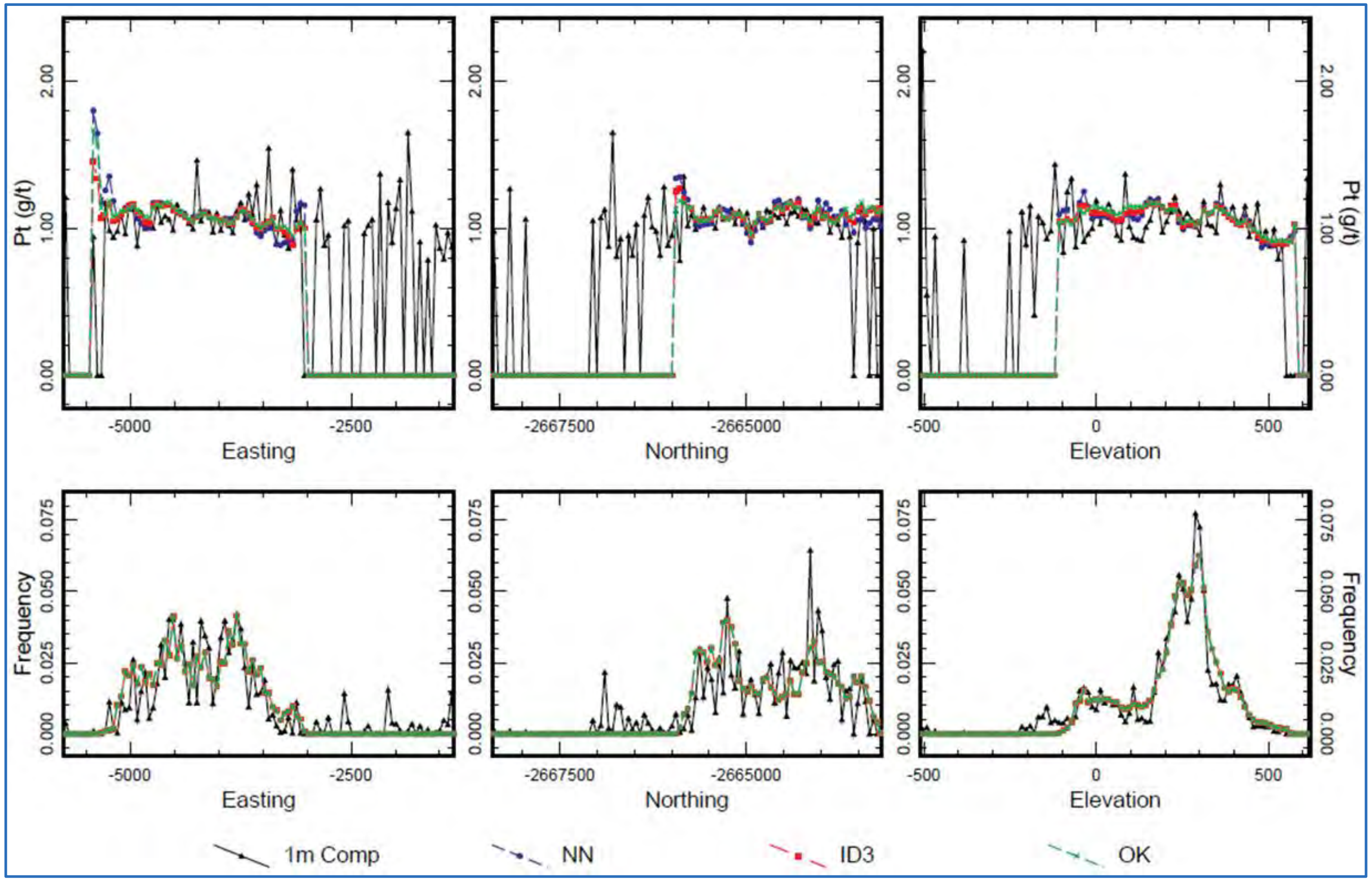
Figure 14.19 Box Plot of Pt Comparing Sub-Celled and Regularised Blocks for T1MZ and T2MZ Indicated Plots By 3PE+Au Grade Shell



	101	101R	102	102R	103	103R	201	201R	202	202R	203	203R	
No. of data	417421	76103	228155	40007	277452	85766	1171379	251086	1007591	183611	1180868	245494	No. of data
Mean	0.605	0.663	1.155	1.039	2.428	1.997	0.662	0.679	1.052	1.084	2.314	2.159	Mean
Maximum	8.994	4.210	15.770	3.630	15.849	15.830	3.970	6.130	4.870	7.400	15.191	13.520	Maximum
U.Quartile	0.716	0.810	1.269	1.190	2.754	2.370	0.742	0.770	1.190	1.250	2.635	2.520	U.Quartile
Median	0.593	0.600	1.144	1.010	2.284	1.860	0.628	0.640	1.006	1.040	2.077	1.990	Median
L.Quartile	0.477	0.450	1.006	0.850	1.893	1.450	0.522	0.530	0.858	0.870	1.680	1.610	L.Quartile
Minimum	0.010	0.010	0.010	0.010	0.010	0.010	0.001	0.010	0.001	0.010	0.010	0.010	Minimum
Variance	0.050	0.106	0.068	0.087	0.694	0.645	0.058	0.064	0.117	0.125	0.868	0.647	Variance
CV	0.369	0.492	0.225	0.285	0.343	0.402	0.363	0.373	0.325	0.327	0.403	0.372	CV
Skewness	1.342	1.701	2.648	0.751	2.269	2.069	2.275	2.288	1.266	1.261	1.753	1.570	Skewness

Wood, 2015; "R" implies regularised 10 x 10 x 2 m blocks.

Figure 14.20 Platinum Swath Plot for T2MZ – 2 g/t 3PE+Au Shell; Regularised Model



Wood, 2016

Comments on the UMT–TCU Model

The UMT–TCU model covers the TCU stratigraphic units. The UMT–TCU model also includes estimation of grades in blocks adjacent to the TCU, up to 20 m into the barren Main Zone gabbro norite, and 75 m into the footwall stratigraphy below the TCU.

Additional drilling is required in the areas classified as Inferred to better define the stratigraphic units and the fault domains.

Local bias is expected in the area classified as Inferred Mineral Resources because of the wide-spaced drilling and large search distances required for grade estimation. Additional drilling should permit better grade estimations.

14.3 UMT–BIK Bikkuri Reef Resource Estimate

The Mineral Resource for the Bikkuri Reef is located in Zones 1 and 2 (Figure 14.2) and is situated stratigraphically above the TCU (Figure 14.3). The Bikkuri Reef includes mineralisation that is amenable to underground selective mining methods and consists of material within and adjacent to grade shells for the Bikkuri Reef. The Mineral Resource for the Bikkuri Reef has been constructed using a geological interpretation that is similar to the TCU. The current interpretation is the Bikkuri Reef is a slump block of the main Platreef (see Section 7 and Figure 7.20). The Mineral Resource estimate is based on the UMT–BIK model. Controls for mineralisation on the Bikkuri Reef are similar to those recognised in the UMT-TCU model; however, mineralisation is typically lower in grade.

14.3.1 Drillhole Data

The drillhole data for the UMT–BIK resource model are a subset of the valid drillholes of the Platreef database (See Section 0) and include 58 drillholes (66,865 m). All UMT drillholes have been re-logged for consideration of the TCU and the Bikkuri Reef. Three ATS drillholes (ATS123, ATS173, ATS176) were included in the Bikkuri model in Zone 2 for constructing the geological model.

14.3.2 Geological Model

The geology model for the UMT–BIK resource model was constructed in Leapfrog. A numeric model code (MCODE) was assigned to each lithology interpreted to be part of the Bikkuri Reef (Table 14.1). Stratigraphic surfacing functions in Leapfrog were used to construct the Bikkuri Reef geological model wireframes.

14.3.3 Model Envelope

The UMT–BIK model envelopes are constructed to include only the Bikkuri stratigraphic sequences.

14.3.4 Mineralised Zones

For the B1 mineralised zone (B1MZ), only a 1 g/t grade shell was modelled. Nested grade shells were constructed for the B2 mineralised zone (B2MZ) to constrain the grade estimation. The nested grade shells were identified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE +Au cut-offs. The grade-shell intercepts were coded into the drillhole database. The grade-shell drillhole intercepts were validated on dip and strike sections to ensure consistency. The grade shell drillhole intercepts were used to construct wireframes of the nested grade shells using stratigraphic surfacing functions in Leapfrog. Grade shell codes (GCODES) were used to code blocks within and outside the grade shells. The GCODES are summarised in Table 14.2.

14.3.5 Mineralisation Adjacent to the Bikkuri Mineralised Zones

There is scattered mineralisation locally adjacent to the B1MZ and B2MZ. Mineralisation adjacent to the Bikkuri mineralised zones may be included in future mine development, and a grade estimate is required for blocks within the Bikkuri model envelope.

14.3.6 Compositing and Exploratory Data Analysis (EDA) For UMT–BIK Model

The subset of drillholes used for the UMT-BIK resource was composited to 1 m length composites within the UMT-BIK model envelope. The compositing was controlled by the nested grade shells and the Bikkuri stratigraphic units.

EDA was completed using box plots, histograms, probability plots and contact profiles. Discontinuities in grade profiles near contacts suggested the grade shells and stratigraphic boundaries should be considered hard boundaries.

14.3.6.1 Rhodium Regressions

Rhodium analyses are only partially complete on the Bikkuri drillholes. Rhodium regressions for the Bikkuri drill data were used to address the missing rhodium data (See Section 14.2.6.1 and Table 14.3 and Table 14.4).

14.3.7 Block Model

The UMT-BIK block model includes two areas where the Bikkuri Reef has been interpreted (Figure 14.2). Blocks were oriented parallel to the national coordinate system. The block model used a parent block size of 10 m x 10 m x 2 m. Sub-celling was 10 m x 10 m x 0.5 m. The geological stratigraphic units and grade shells were coded to the blocks. After estimation, the final resource model blocks were regularised to 10 m x 10 m x 2 m block sizes.

14.3.8 Grade Estimation — UMT–BIK

14.3.8.1 B1MZ and B2MZ

Grade estimation in the B1MZ and B2MZ included block and composite matching by GCODE. To eliminate the effects of the structural blocks and variability in elevation, the centre of individual stratigraphic units and mineralised zones were transformed to hang from the 1,000 m elevation.

The grade estimation in the B1MZ and B2MZ included block and drillhole composite matching by a combination of MCODE and GCODE to ensure the stratigraphic components of the B1MZ and B2MZ were estimated separately. Estimation was completed by ID3. A NN and OK estimations were completed for model validation.

After grade estimation, the blocks and composites were back transformed to the original elevation.

14.3.8.2 Blocks Adjacent Grade Shells

Grade estimation in the blocks not included in the B1MZ and B2MZ mineral zones were estimated by matching blocks and composites by MCODE. The individual stratigraphic units were hung from the 1,000 m elevation. Estimation was completed by ID3. Alternate NN and OK grade estimation was completed for model validation. After grade estimation, the blocks and composites were back transformed to the original elevation.

Grade estimations were completed in Datamine using expanding search volumes. Search volumes are summarised in (see Table 14.13).

Table 14.13 Search Strategy for Bikkuri Grade Estimation (All Elements)

Search Volume	Search Distances			Min. Samples	Max. Samples	Max. per Drillhole
	X	Y	Z			
1	300	300	100	4	15	3
2	600	600	200	4	15	3
3	120	1200	400	1	15	3

14.3.8.3 Grade Capping and Outlier Restriction

No grade capping or outlier restriction was implemented.

14.3.8.4 Unestimated Blocks

Blocks that were not estimated were assigned a default grade of the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.14.

Table 14.14 Bikkuri Mean Grades by Stratigraphic Unit

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZBK	0.001	0.001	0.001	0.001	0.010	0.010	0.010
NC1BK	0.078	0.075	0.034	0.004	0.069	0.029	0.213
B1	0.125	0.090	0.049	0.007	0.084	0.030	0.195
B2	0.219	0.194	0.083	0.013	0.137	0.065	0.425
NC2BK	0.156	0.220	0.040	0.010	0.108	0.073	0.432
LZ1BK	0.267	0.313	0.060	0.060	0.134	0.086	0.945
B1MZ 1g	0.497	0.364	0.155	0.025	0.172	0.081	0.451
B2MZ 1g	0.632	0.661	0.145	0.034	0.244	0.145	0.867
B2MZ 2g	1.047	1.029	0.229	0.056	0.303	0.168	0.926
B2MZ 3g	1.445	1.323	0.311	0.077	0.389	0.211	1.099

14.3.9 Regularisation

Upon completion of the estimation, the UMT-BIK block model was regularised to 10 m x 10 m x 2 m (no sub-cells) model blocks. The 10 m x 10 m x 2 m regularised model permitted better resolution along the faulted boundaries and softened the hard boundaries used in the grade estimation.

Densities were coded to the blocks by stratigraphic unit using the mean density values for each stratigraphic unit (see Table 14.15).

Table 14.15 Bulk Density Values for The Bikkuri Model

Zone	Mean Density	CV	Max. SG	Min. SG
HW	2.91	0.04	4.47	2.04
NCMANBK	2.86	0.02	2.95	2.63
B1	3.11	0.06	3.90	2.60
B2	3.13	0.04	3.30	2.62
FW	2.91	0.04	4.47	2.04

14.3.10 Mineral Resource Classification

The Bikkuri Mineral Resources has been classified using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014), as discussed in Section 14.2.11.

The boundaries of Indicated and Inferred Mineral Resources for the TCU–BIK resource model are shown in Figure 14.2. The drill spacing in the Indicated Mineral Resource is nominally 100 m. Drill spacing in the Inferred Mineral Resource ranges up to 400 m.

14.3.11 UMT–BIK Model Validation

Model validation included visual inspection of block grades relative to composite grades on cross-sections and level plans. Statistical comparisons consisting of box plots and grade profiles tabulated in different directions (swaths) for each metal by stratigraphic unit and 3PE+Au grade shell were constructed to compare the kriged (where present), ID3 grade estimates, NN estimates and 1 m composites.

14.3.11.1 Global Bias Check

The checks for global bias for the UMT-BIK grade estimate were performed by comparing the ID3 average grade (with no cut-off) from the NN estimates by mineralised unit. Blocks reviewed were restricted to those classified as Indicated Resources. Domains with a global bias outside guidelines of ±5% (relative) are highlighted in Table 14.16.

The grade estimates for the B1MZ grade shells are outside of the stated guideline. The Au grade estimate for the B2MZ 2g/t 3PE+Au is outside of the stated guideline.

Table 14.16 UMT-BIK Global Relative Bias Check for Pt

GCODE	Element	NN	ID3	Relative Difference
301	Pt	0.390	0.438	10.9%
	Pd	0.272	0.299	9.0%
	Au	0.130	0.157	17.1%
	Rh	0.018	0.020	7.1%
	Ni	0.146	0.157	6.6%
	Cu	0.062	0.068	8.6%
401	Pt	0.597	0.597	0.1%
	Pd	0.593	0.596	0.5%
	Au	0.150	0.149	-1.3%
	Rh	0.033	0.033	0.1%
	Ni	0.217	0.220	1.0%
	Cu	0.130	0.130	0.2%
402	Pt	1.045	1.083	3.5%
	Pd	0.946	0.968	2.3%
	Au	0.225	0.239	5.6%
	Rh	0.052	0.053	2.9%
	Ni	0.315	0.323	2.5%
	Cu	0.163	0.171	4.9%
403	Pt	0.618	0.604	-0.5%
	Pd	1.452	1.453	0.1%
	Au	0.332	0.338	2.0%
	Rh	0.087	0.086	-1.1%
	Ni	0.399	0.403	0.9%
	Cu	0.223	0.225	1.0%

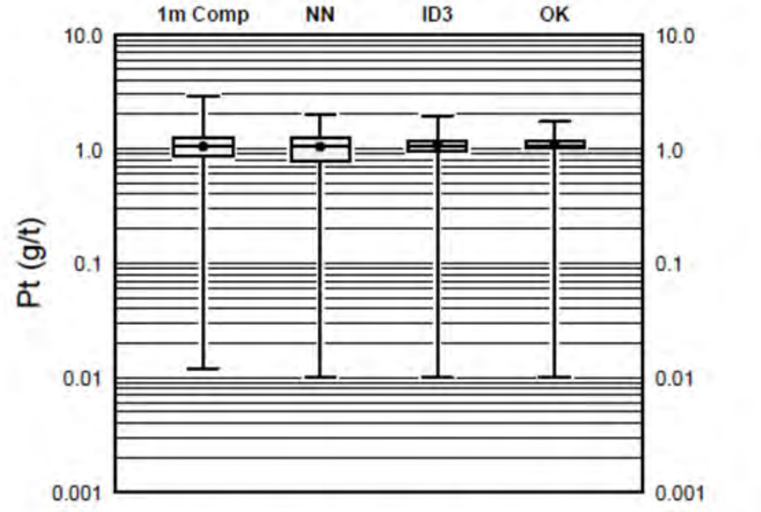
Note: Rows are shaded where the relative bias is outside (±5%).

14.3.11.2 Box Plots – UMT-BIK Model

Box plots were completed for the sub-celled model for each element comparing the 1 m composites, NN, ID3 and OK estimates by mineralised unit. Block selection was restricted to those classified as Indicated. Figure 14.21displays the box plots for Pt within the B2MZ 2 g/t 3PE+Au. The box plots for the UMT-BIK sub-celled model show good agreement for the B2MZ. A bias is observed for the B1MZ.

The sub-celled and regularised models were also compared (Figure 14.22). The grades for the regularised model is commonly low due to the inclusion of low-grade blocks in the regularisation process.

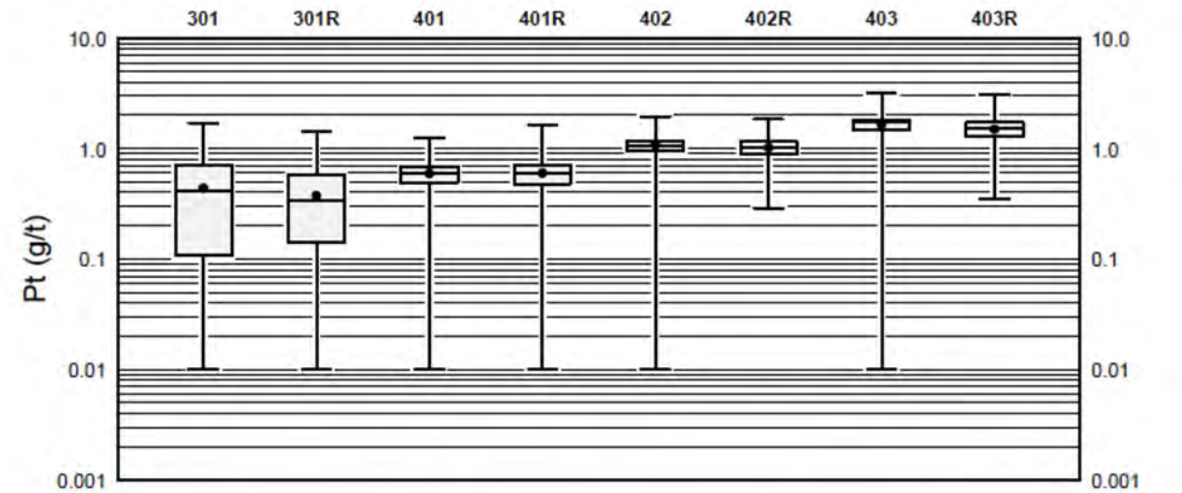
Figure 14.21 Box Plot Pt For B2MZ 2g/t 3PE+Au Figure Caption



	1m Comp	NN	ID3	OK	
No. of data	172	59505	59505	59505	No. of data
Mean	1.050	1.045	1.083	1.087	Mean
Maximum	2.891	1.980	1.959	1.725	Maximum
U.Quartile	1.257	1.252	1.187	1.164	U.Quartile
Median	1.052	1.040	1.070	1.059	Median
L.Quartile	0.857	0.780	0.968	1.010	L.Quartile
Minimum	0.012	0.010	0.010	0.010	Minimum
Variance	0.143	0.117	0.029	0.016	Variance
CV	0.360	0.327	0.156	0.118	CV
Skewness	0.568	0.211	0.151	-0.470	Skewness

Ivanhoe, 2015

Figure 14.22 Regularised Model Check for UMT-BIK



	301	301R	401	401R	402	402R	403	403R	
No. of data	12939	1871	103815	18409	59505	6083	36245	5410	No. of data
Mean	0.438	0.374	0.597	0.602	1.083	1.020	1.651	1.497	Mean
Maximum	1.707	1.440	1.257	1.620	1.959	1.840	3.203	3.070	Maximum
U.Quartile	0.705	0.570	0.687	0.700	1.187	1.150	1.835	1.730	U.Quartile
Median	0.420	0.340	0.590	0.590	1.070	1.020	1.728	1.520	Median
L.Quartile	0.107	0.140	0.492	0.480	0.968	0.880	1.485	1.290	L.Quartile
Minimum	0.010	0.010	0.010	0.010	0.010	0.290	0.010	0.350	Minimum
Variance	0.099	0.068	0.022	0.025	0.029	0.044	0.167	0.126	Variance
CV	0.718	0.698	0.247	0.264	0.156	0.206	0.248	0.238	CV
Skewness	0.306	0.460	0.275	0.510	0.151	0.091	-0.843	-0.186	Skewness

Ivanhoe, 2015; "R" indicates regularised blocks.

14.3.11.3 Visual Validation

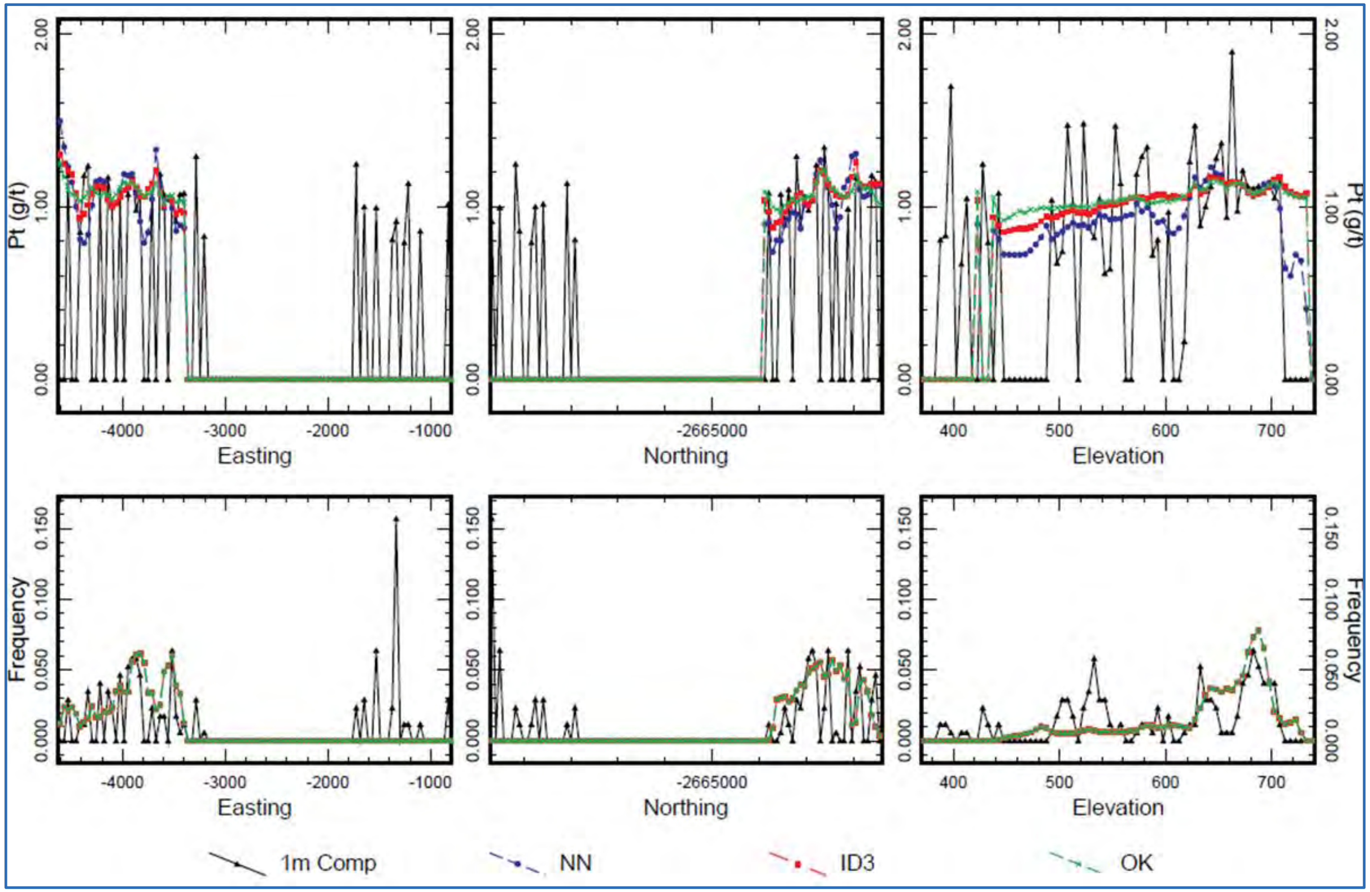
Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long-sections and level plans. In general, the composite grades were honoured in the block distributions. Representative cross-sections for 3PE+Au are shown in Figure 14.14 and Figure 14.16. Representative cross-sections showing Ni grades are presented in Figure 14.15 and Figure 14.17.

14.3.11.4 Swath Plots

Swath plots (width of 200 m) of the ID3 model, NN model and 1 m composites were completed for Pt, Pd, Au, Rh, Cu, and Ni. Overall, swath plots display reasonable comparisons between the ID3 estimates to their respective NN estimates; however, locally there are some differences.

Swath plot analysis are commonly focussed on blocks classified as Measured and Indicated. However, because of the limited extent of the UMT-BIK resource model, swath plots were completed for the entire UMT–BIK resource model. The Pt swath plot (sub-celled model) for the B2MZ 2 g/t 3PE+Au grade shell is presented in Figure 14.23.

Figure 14.23 Swath Plot for Pt; B2MZ 2g/T 3PE+Au; Sub-Celled Model; Indicated + Inferred Blocks



Wood, 2016

14.3.12 Comments on the UMT–BIK Model

As currently configured the UMT–BIK model covers the stratigraphic units that are interpreted to be the Bikkuri Reef.

The UMT–BIK model locally includes estimation of grades in blocks adjacent to the Bikkuri mineral zones (B1MZ and B2MZ). The UMT-BIK model is limited to the BIK model envelope. Additional drilling is required to better define the lateral extents of the Bikkuri mineralisation and the boundary between the Bikkuri stratigraphic units and the TCU stratigraphic units, interpreted to be within the Footwall Assimilation Zone (FAZ) and the Pyroxenite-Norite Zone (PNZ) (See Section 7.7). The mineralisation found within the FAZ is generally less continuous and disrupted by: 1) rafts of metasedimentary rocks, and 2) rock types that have been heavily assimilated. There are distinct assimilation products associated with dolomite assimilated rocks (calc-silicates and para lithologies) and hornfels assimilated rocks (a variety of norite products).

Numerous areas of footwall mineralisation have been identified. Two footwall domains were identified to have sufficient drillhole density and grade continuity to warrant the construction of Mineral Resource models. These include the Clinopyroxenite domain (FWcpx) and the Pyroxentie-Norite Zone (FWpnz). Figure 14.2 shows the location of the FWcpx and FWpnz domains. Additional footwall mineralisation is recognised, but the insufficient drillhole and sample data does not support resource modelling. These areas represent future exploration potential.

14.4 Drillhole Data — UMT–FW

The drillhole data for the UMT–FW resource model are a subset of the Platreef valid drillhole database and include 102 drillholes (121,879 m). Only drillholes from the UMT drill programme were used for the estimation of the UMT-FW Mineral Resource.

14.4.1 Geology Model (UMT–FW)

The geological interpretations for the UMT–FW Mineral Resource model are based on revised geological interpretation. This interpretation is based primarily on the drill core re-logging campaign. The FW Stratigraphic coding is summarised in Table 14.16. Two mineralised domains are included in the UMT-FW mineral resource model. The upper domain is the FWcpx domain within the FAZ. Below the FAZ is mineralisation associated with the FAZ – PNZ contact and mineralisation within the PNZ associated with hornfels units (Figure 14.2).

14.4.2 CPX Domain

The FWcpx domain is confined to the NW area of Zone 1 (Figure 14.2), located within the FAZ (Figure 14.3). The domain is constrained by a very distinct, homogeneous clinopyroxene-rich pyroxenite where metasedimentary xenoliths have been completely assimilated. There are three main lithologies that make up the FWcpx domain. The main lithology is a clinopyroxenite (CPX) that locally includes added feldspar (FCPX) or added olivine (OLCPX).

14.4.3 PNZ Domain

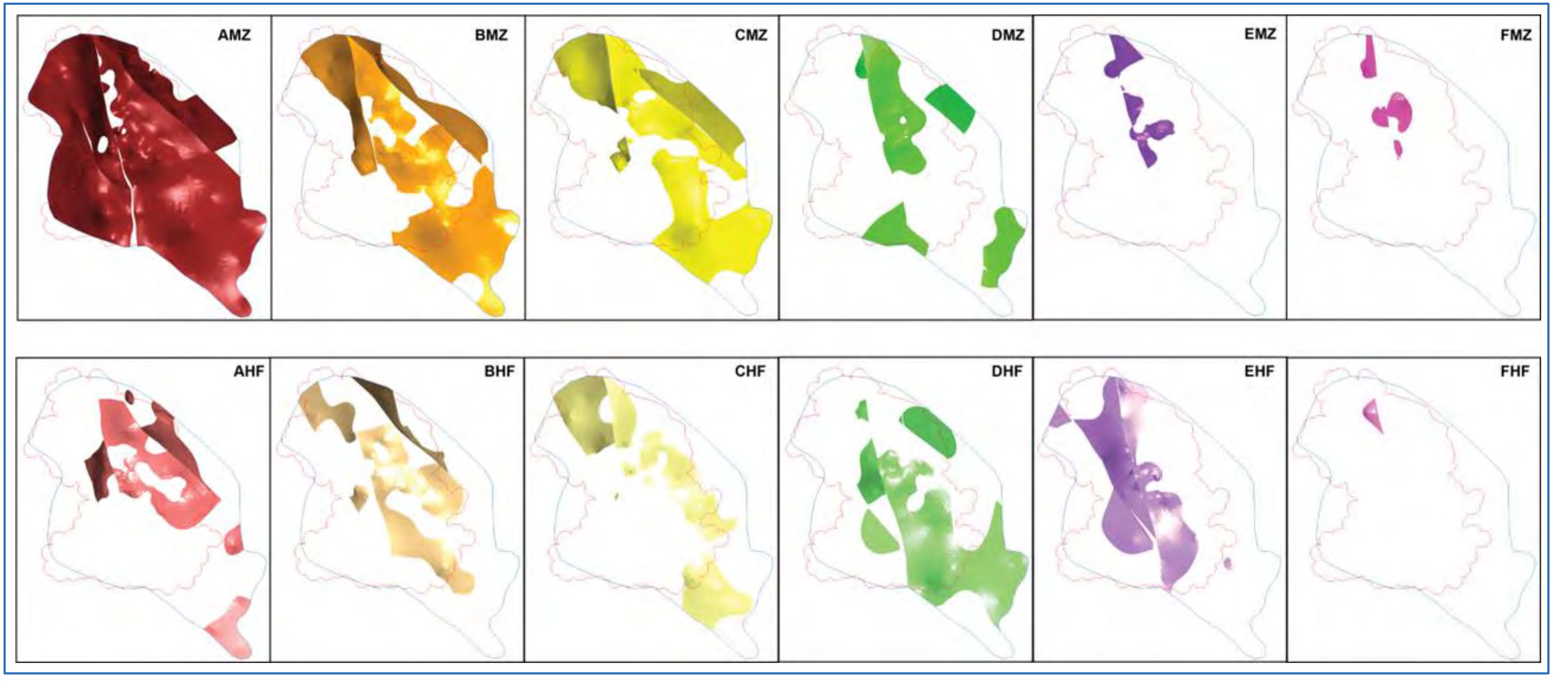
The PNZ is a unit of pyroxenite – norite composition that includes local unassimilated hornfels xenoliths. The mineralisation occurs predominantly as disseminated sulfides. Local massive sulfides are recognised at the contacts with the hornfels xenoliths. The 3PE+Au grades are commonly in the 1 g/t range, but locally can be 2 – 5 g/t 3PE+Au.

The contact between the FAZ and the PNZ is typically sharp (see Figure 14.3). A marked increase in mineralisation occurs locally at or near the contact FAZ-PNZ contact and is designated Mineralised Zone A (AMZ). The AMZ is distributed across the entire UMT-FW model area (Figure 14.24). Below the FAZ-PNZ contact, two distinct styles of mineralisation are observed within the PNZ pyroxenite. Discontinuous mineralisation, commonly as massive sulfides, is observed locally at the contacts of the hornfels xenoliths. More consistent mineralisation forms mineralised zones between hornfels units and it is these zones that were included in the UMT-FW model.

Six hornfels xenoliths (AHF, BHF, CHF, DHF, EHF and FHF in descending order) have been identified in the FWpnz domain. Five minzones (BMZ, CMX, DMZ EMZ and FMZ in descending order) are identified between the hornfels xenoliths. The lateral extent of the correlated mineralisation is less with each deeper minzone (see Figure 14.24). The units and mineralised zones were modelled across a wide portion of the project area (the blue perimeter in (Figure 14.24), the grade estimate was restricted to a narrower zone where tighter drillhole spacing allowed for continuity to be assumed.

The hornfels xenoliths and minzones were correlated in cross-section by Ivanhoe geology staff and coded to the drillhole database. Wireframes for the hornfels xenoliths and the FWpnz mine zones were constructed in Leapfrog using the vein modelling functions. Model and composite coding is summarised in Table 14.17.

Figure 14.24 Extent of Mineralised Zones AMZ To FMZ and Hornfels Units AHF to FHF



Ivanhoe, 2016; All of FWpnz is Inferred Mineral Resources. Blue boundary is FWpnz model boundary. Red boundary is TCU Indicated Mineral Resource boundary.

Table 14.17 MCODE and GCODE for FW Model

Modelled Unit	MCODE	Modelled Unit	GCODE
CPX	301	CPX	0
FAZ	30	FAZ	0
PNZ	31	PNZ	0
AHF	311	AMZ	311
BHF	312	BMZ	312
CHF	313	CMZ	313
DHF	314	DMZ	314
EHF	315	EMZ	315
FHF	316	FMZ	316

14.4.4 Density — UMT–FW

Density was assigned based on the average density value for the Strat units (see Table 14.18). This is considered appropriate, as the distribution of density values per unit have low coefficients of variation (CV). Mean density values were applied for the CPX and FAZ zones. For the PNZ, separate density values were assigned to the magmatic and sedimentary rock portions of this zone.

Table 14.18 Density Values for the UMT–FW Model

Zone	Unit	No of Samples	Mean	CV	10th Percentile	90th Percentile
HF	MZ	196	2.84	0.02	2.77	2.91
	TCU	10	2.90	0.06	2.71	3.15
	FAZ	207	2.94	0.06	2.77	3.16
	HFR	75	2.87	0.05	2.74	3.13
	PNZ	673	2.85	0.03	2.77	2.95
	LZ	1	2.87	-	-	-
	TVL	27	2.82	0.05	2.69	3.00
	Total	1189	2.87	0.04	2.76	3.01
PNZ	SED	940	2.87	0.06	2.71	3.12
	MAGMA	3031	3.15	0.03	3.04	3.25
	Total	3971	3.08	0.06	2.81	3.24
FAZ	Total	6389	3.11	0.05	2.91	3.28
CPX	Total	545	3.24	0.03	3.12	3.35

14.4.5 Outlier Restriction — UMT–FW

An outlier restriction distance threshold of 15 m was applied to high grade samples within each stratigraphic unit and mineralised zone. The grade thresholds for outliers were selected from inspection of the histograms and probability plots of 1m drillhole composites and are summarised in Table 14.19. Composites with grades above the grade threshold and with distances from composite to block centre beyond the distance thresholds were not used in grade estimation.

Table 14.19 Outlier Restriction Thresholds for Stratigraphic Units (MCODE)

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
CPX	4.50	5.00	1.20	1.50	0.75		6.00
FAZ	7.50	7.50	1.00	1.00	0.80	0.40	8.00
PNZ	2.00	2.00	0.60	1.00	0.90	-	8.00
AMZ	4.00	4.00	-	1.00	0.60	-	
BMZ	4.00	-	-	1.20	0.60	-	6.50
CMZ	4.50	-	0.50	1.20	-	-	7.00
DMZ	5.00	-	-	1.20	-	0.30	7.00
EMZ	5.00	4.00	0.60	0.80	0.60	-	4.50
FMZ	-	-	-	-	-	-	-
AHF	0.30	0.4	-	0.30	0.17	-	-
BHF	0.32	0.4	0.10	0.24	0.30	-	3.20
CHF	0.35	0.35	0.07	0.21	0.18	-	3.80
DHF	0.60	0.6	0.07	0.22	0.26	-	3.50
EHF	0.35	0.45	0.12	0.30	0.29	-	3.50
FHF*	0.35	0.45	0.12	0.30	0.29	-	3.50

14.4.6 Grade Estimation - UMT–FW

14.4.6.1 Grade Estimation - CPX Domain

The CPX model was constrained within a CPX model envelope that defined the limits of the FWcpx geological domain.

The block model used parent blocks of 10 m x 10 m x 2 m with no subcelling (Table 14.6). Fault blocks were not used to sub-domain the grade estimation for the FWcpx domain. The FWcpx model included portions of the TCU model but was not permitted to include blocks within the T2MZ.

Composites and model blocks were transformed so that the centre of the CPX domain was hung from the 1,000 m elevation for grade estimation. The composites and blocks were back-transformed to the original elevation after the grade estimation was completed.

Grade estimation was completed for Pt, Pd, Rh, Au, Cu, Ni and S using ID3. NN and OK estimations were completed for validation purposes. Variograms for the T2MZ g/t model were utilised for the OK estimation.

Grade estimations were completed in Datamine using only the first search volume (see Table 14.20). Blocks not estimated in the first search volume were excluded from the CPX model. The blocks not estimated are commonly located at the base of the CPX domain where drillholes were either not deep enough or the CPX intercept is not sampled.

14.4.6.2 Grade Estimation - PNZ Domain

The FWpnz model was constrained within the FAZ and PNZ envelopes. Fault blocks were not used to sub-domain the grade estimation for the FWpnz domain because only an isolated portion of the PNZ domain occurs east of the Tshukudu Fault.

The block model used parent blocks of 10 m x 10 m x 2 m with subcelling to 5 m x 5 m x 0.5 m (see Table 14.6) for better geological resolution. The PNZ model was permitted to overwrite the TCU model below the lower boundary of the T2MZ.

Composites and model blocks were transformed to hang from the centre of each lithology or MZ from the 1,000 m elevation for grade estimation. The composites and blocks were back-transformed to the original elevation after the grade estimation was completed.

Grade estimation was completed for Pt, Pd, Rh, Au, Cu, Ni and S using ID3. NN and OK estimations were completed for validation purposes. Variograms for the T2MZ 1 g/t model were utilised for the OK estimation. Grade estimations were completed in Datamine in three estimation passes using expanding search volumes (see Table 14.20). Blocks not estimated were excluded from the FWpnz model.

An outlier restriction was applied to the grade estimation using a distance threshold of 15 m and grade thresholds summarised in Table 14.19.

Table 14.20 Estimation Parameters for FW Model

Search Pass	Axis	Azimuth	Dip	Search Range	Min. Samples	Max. Sample	Max. per Drillhole
1	X	90	0	250	4	15	3
	Y	0	0	250	4	15	3
	Z	0	90	10	4	15	3
2	X	90	0	500	4	15	3
	Y	0	0	500	4	15	3
	Z	0	90	20	4	15	3
3	X	90	0	2,000	1	15	3
	Y	0	0	2,000	1	15	3
	Z	0	90	2,000	1	15	3

14.4.6.3 Unestimated Blocks

Blocks that were not estimated were assigned a default grade; the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.10. Unestimated blocks were generally located along fault-block boundaries. Unestimated blocks within the FW stratigraphy were found to be located in areas of wide-spaced drilling.

14.4.7 TCU–FW Model Validation Global Bias Check

The checks for global bias for the UMT-FW grade estimate were performed by comparing the ID3 average grade (with no cut-off) from the NN estimates by GCODE. Domains with a global bias outside guidelines of ±5% (relative) are highlighted in Table 14.21 and Table 14.22.

Table 14.21 summarises the global bias check for the FWcpx domain. The grade estimations are within the recommended ±5% tolerance.

Table 14.22 summarises the global bias check for the FWpnz domain. Generally, the grade estimations are within the recommended ±5% tolerance. Exceptions are the Au and Ni estimations for the CMZ zone (313).

Table 14.21 UMT–FWCPX Global Bias Check

Element	NN	ID3	Relative Difference
Pt	0.47	0.46	-1.51%
Pd	0.60	0.59	-0.98%
Au	0.08	0.08	-0.77%
Rh	0.06	0.06	0.48%
Ni	0.20	0.20	-0.70%
Cu	0.09	0.08	-1.82%

Table 14.22 UMT-FWCPX Global Bias Check

MZ	Element	NN	ID3	Relative Difference
AMZ (311)	Pt	0.50	0.50	-0.25%
	Pd	0.55	0.55	0.49%
	Au	0.08	0.08	1.34%
	Rh	0.07	0.07	-0.49%
	Ni	0.19	0.19	1.68%
	Cu	0.11	0.11	0.71%
BMZ (312)	Pt	0.44	0.44	-0.12%
	Pd	0.58	0.56	-2.50%
	Au	0.09	0.09	-1.29%
	Rh	0.07	0.07	0.99%
	Ni	0.21	0.21	0.41%
	Cu	0.12	0.13	0.63%
CMZ (313)	Pt	0.45	0.47	4.81%
	Pd	0.65	0.66	2.47%
	Au	0.09	0.09	5.39%
	Rh	0.07	0.07	1.25%
	Ni	0.25	0.27	5.42%
	Cu	0.14	0.14	2.62%
DMZ (314)	Pt	0.60	0.59	-0.56%
	Pd	0.71	0.71	-0.02%
	Au	0.11	0.11	0.62%
	Rh	0.07	0.07	0.69%
	Ni	0.20	0.21	3.85%
	Cu	0.12	0.13	2.32%
EMZ (315)	Pt	0.18	0.19	2.41%
	Pd	0.11	0.11	2.01%
	Au	0.61	0.62	2.39%
	Rh	0.70	0.71	1.45%
	Ni	0.09	0.09	0.05%
	Cu	0.07	0.07	1.09%
FMZ (316)	Pt	0.18	0.19	3.34%
	Pd	0.12	0.13	2.27%
	Au	0.46	0.48	4.00%
	Rh	0.60	0.61	1.53%
	Ni	0.09	0.09	-0.01%
	Cu	0.07	0.07	-0.14%

14.4.7.1 Visual Inspection

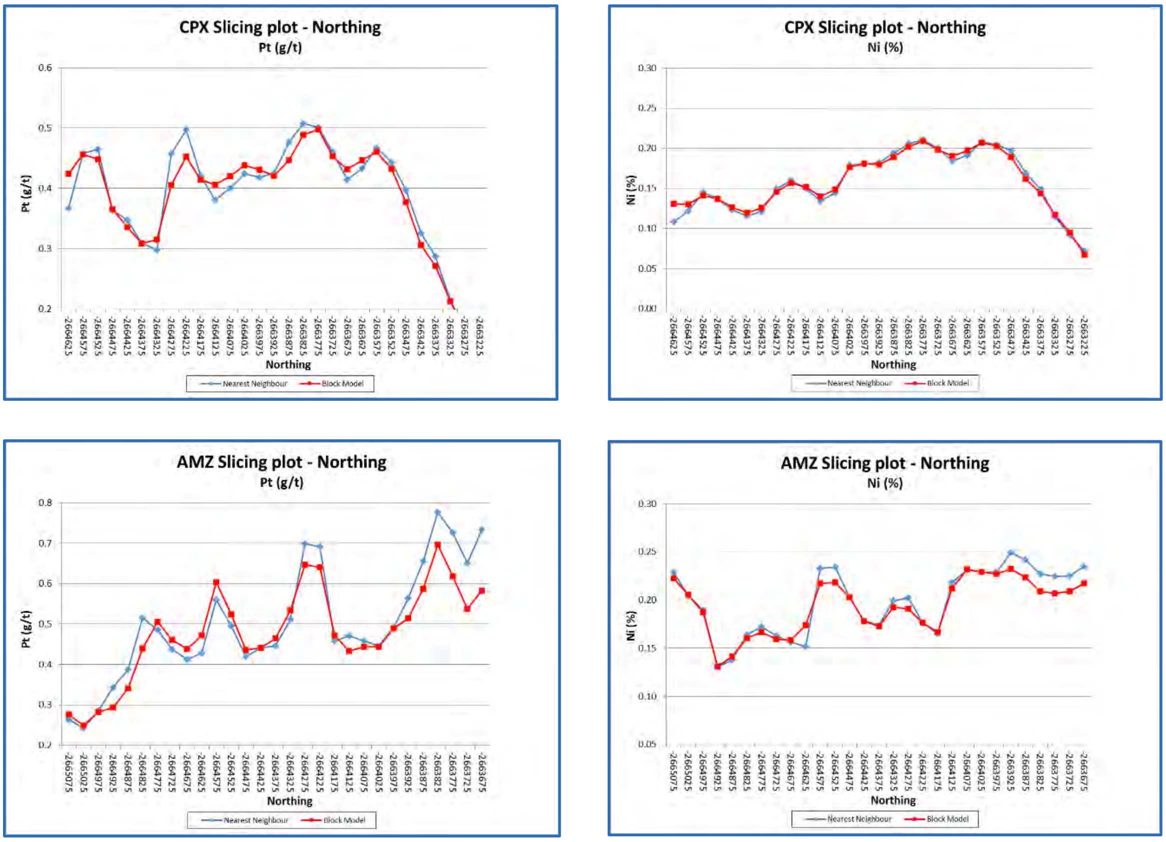
Visual validation of cross-sections for the FWcpx and FWpnz domains compared grade estimations and the 1 m composites. Figure 14.14 and Figure 14.15 shows the grade estimation for the FWcpx domain. Figure 14.16 and Figure 14.17 shows the grade estimation for the FWpnz domain.

The visual inspections indicate that grade continuity is best observed at a 1.0 to 1.5 g/t 3PE+Au cut-off). At a 2 g/t 3PE+Au cut-off, mineralisation is more restricted, and additional drilling is required to fully define the mineralisation.

14.4.7.2 Swath Plots

Swath plots were completed to compare grade estimation to the NN estimation and also to composites (see Figure 14.25).

Figure 14.25 Platinum Swath Plot for B2MZ – 2 g/t 2PE+Au Shell



Ivanhoe, 2016

14.4.8 Classification

The Mineral Resource Classification for the FWcpx and FWpnz domains are Inferred due to the limited drilling.

14.4.9 Regularisation

Upon completion of the estimation, the FWpnz and FWcpx model were regularised to 10 m x 10 m x 2 m blocks and combined into a single UMT-FW Mineral Resource model. The 10 m x 10 m x 2 m softened the hard boundaries used in the grade estimation.

14.4.10 Comments on the UMT–FW Model

The UMT–FW model includes geological domains stratigraphically beneath the TCU that are observed to have a degree of geological continuity and homogeneity.

The UMT-FW model is limited to the FWcpx and FWpnz model envelopes that define unique geological domains within footwall stratigraphy. Additional drilling is required to better define the lateral extents of the FWcpx and FWpnz domains.

The continuity of FW mineralisation has been modelled based on limited drill data, as not all of the UMT drillholes extended into the FW. For this reason, estimation of Mineral Resources has been restricted to the north-western area of the Platreef Project where drill spacing is in the order of 100 m to 200 m. Similar mineralisation has been seen in drillholes across the entire Platreef Project, but the current drill spacing is insufficient to define Mineral Resources amenable to selective mining methods in these areas. This represents exploration upside for the Platreef Project.

Drill intercepts ≥ 2.0 g/t 3PE+Au in the FW domains are narrow, and suggest selective mining would be required. Grade continuity is best observed at a 1.0 to 1.5 g/t 3PE+Au cut-off. Discontinuous pods of mineralisation at a 2.0 g/t 3PE+Au cut-off are present, but are not well defined at the current drill spacing, and additional drilling is required. The FWcpx domain includes thicker zones of low-grade mineralisation that may permit mass mining methods at a lower cut-off (1 g/t 3PE+Au).

14.5 Final Platreef Mineral Resource Model

The three Mineral Resource Models (UMT-TCU, UMT-BIK and UMT-FW) were combined into a final Platreef Mineral Resource Model.

14.6 Assumptions Made to Assess Reasonable Prospects of Eventual Economic Extraction

MTS undertook a conceptual analysis to assess reasonable prospects for eventual economic extraction for declaration of Mineral Resources. Underground mining methods considered are conventional, mechanised mining methods that have a reasonable safety factor. Assumptions made have been based on Base Data Template 20, received from Ivanhoe on 15 September 2015. These economic inputs were rerun on 28 January 2022 to confirm the estimates as current.

14.6.1 Commodity Prices

MTS considers that forecast long-term commodity prices should be used in declaration of Mineral Resources. For the Mineral Resource estimates, the following prices were used: $1,600/troy ounce for Pt, $815/troy ounce for Pd, $1,300/troy ounce for Au, $1,500/troy ounce for Rh, $3.00/lb for Cu and $8.90/lb for Ni. The Mineral Resources were evaluated using 2022 forecast prices and there was no material change to the Mineral Resources.

14.6.2 Onsite Operating Costs

For the selectively-mineable higher-grade scenario, a production rate of 4 Mtpa was assumed. Mining costs for some form of selective mining were estimated at $34.27/t. Process, concentrate transport and general and administrative (G&A) costs for this case were estimated at an average of $15.83/t of mill feed.

14.6.3 Process Recoveries

For the selective high-grade option, typical process recoveries are shown in Table 14.23. These recoveries were available from Base Data Template 20, provided by Ivanhoe on 15 September 2015.

Table 14.23 Typical Metallurgical Recoveries (15 September 2015, BDT20)

	Metallurgical Domain		
	Case 1	Case 2	Case 3
Mass Pull (%)	3.39	3.82	4.14
Typical Head Grade			
Pt (g/t)	1.13	1.68	2.11
Pd (g/t)	1.18	1.71	2.12
Au (g/t)	0.20	0.28	0.34
Rh (g/t)	0.08	0.11	0.14
4PGE (g/t)	2.59	3.78	4.65
Cu (%)	0.13	0.16	0.18
Ni (%)	0.26	0.32	0.35
Recoveries (%)			
Pt	79.9	84.6	87.7
Pd	80.6	84.3	85.8
Au	71.5	86.4	90.9
Rh	78.4	85.5	89.7
4PGE			
Cu	84.1	88.6	88.6
Ni	65.2	68.1	69.7

Mass Pull = percentage weight recovery to concentrates.

14.6.4 Smelter Payables

MTS assumed that a smelter would pay for 82% of the metals contained in the concentrates. This assumption is based on a survey made by Kramer (2012). It is likely to cost an average of $39.77/t of concentrates (approximately $1.21/t of mineralised material) for road-freight to transport concentrates to a smelter, which for the purposes of assessing reasonable prospects, was assumed to be Rustenburg, in RSA.

There is some risk that if PGE concentrate grades are low, smelters would also levy treatment charges; on the other hand, it is envisioned that Platreef concentrates would be low in chromium, which might make them attractive to smelters whose feedstock primarily comes from Merensky and UG2 reef concentrates. MTS's conceptual analysis does not include treatment charges.

Platreef concentrates could also be marketed to smelters outside RSA.

14.6.5 Royalty

The royalty has been assumed as 5% of smelter payables.

14.7 Mineral Resource Statements

Mineral Resource statements for Mineral Resources amenable to underground mining methods are tabulated in this Section. The term base case has been used to indicate the tonnage and grade estimate that are considered by MTS to provide a starting point for feasibility studies. Other rows in the resource statements have been provided to show sensitivity of the estimated tonnages and grade to changes in cut-off criteria.

Mineral Resources are reported on a 100% basis. Attributable ownership is discussed in detail in Section 4.

14.7.1 UMT-TCU Mineral Resources Amenable to Underground Mining Methods

A selective mining scenario is considered the base case that could exploit mineralisation at depth within the Platreef. The selectively-mineable option is considered the base case Mineral Resource estimate for the purposes of this Report.

Mining costs have been considered in setting the cut-off ($34.27/t) for the selective mining case. Other considerations include process, concentrate transport and site G&A costs that must be covered for reporting Mineral Resources.

14.7.2 UMT-TCU Mineralisation Within and Adjacent to TCU Amenable to Underground Mining Methods (Estimate Assuming Underground Selective Mining Methods)

The TCU and adjacent blocks above T1, between T1 and T2 and below T2 contain higher-grade mineralisation that could be mined using underground selective methods such as longhole open-stoping, drift/cut and bench, bench-and-fill or Drift-and-Fill.

Table 14.24 shows Mineral Resources lying within and adjacent to the TCU mineralised zones.

In August 2017, the Qualified Person tested the Mineral Resources for reasonable prospects for eventual economic extraction (Parker, et al, 2017). The metals prices reviewed were: $1250/oz for platinum, $850/oz for palladium, $1300/oz for gold, $1000/oz for rhodium, $7.60/lb for nickel, and $3.00/lb for copper. At a 2 g/t 3PE+Au cut-off grade, >99% of the blocks will generate an NSR/t of $50 or higher, meaning they will pay mining, process, concentrate transport, and G&A costs. In 2016, an NSR/t of $50 was being considered by Ivanhoe, with longhole open stoping being the primary mining method. Approximately 90% of the blocks will generate an NSR/t of $70 or higher, which would be applicable to a Drift-and-Fill mining method. All of the blocks above a 1 g/t 3PE+Au cut-off generate an NSR of $15/t, meaning they will cover process, concentrate transport, and G&A costs. A check of the effect of implementing the metallurgical recoveries shown in Table 13.29 was also made, and given those recoveries were higher for Pt, Au and Ni and lower for Pd, Rh and Cu, the new metallurgical recovery equations do not meaningfully affect block valuation compared to equations BDT20 (Table 14.23).

Mr. Kuhl completed a similar evaluation in January 2022 using the 2022 prices and confirmed the statements above are still valid.

14.7.3 Bikkuri Reef Resource Estimate

Table 14.25 provides the total Mineral Resource estimate for mineralisation lying within and adjacent to 3PE+Au grade shells for the Bikkuri Reef.

Table 14.24 Mineral Resources Within and Adjacent to TCU (Base Case is Highlighted)

Indicated Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	202	2.11	2.12	0.34	0.14	4.71	0.18	0.35
2 g/t	339	1.68	1.71	0.28	0.11	3.79	0.16	0.32
1 g/t	685	1.13	1.18	0.20	0.08	2.59	0.13	0.26
Indicated Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t		13.7	13.7	2.2	0.9	30.6	788	1,576
2 g/t		18.4	18.7	3.1	1.2	41.3	1,197	2,386
1 g/t		24.9	26.1	4.3	1.8	57.19	1,977	3,938
Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	212	1.93	1.95	0.32	0.13	4.33	0.17	0.35
2 g/t	459	1.45	1.48	0.27	0.10	3.29	0.16	0.31
1 g/t	1,213	0.91	0.96	0.18	0.07	2.12	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t	–	13.1	13.3	2.2	0.9	29.4	802	1,625
2 g/t	–	21.3	21.9	3.9	1.4	48.6	1,591	3,103
1 g/t	–	35.4	37.5	6.9	2.7	82.5	3,472	6,579

1. Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.

3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.

Table 14.25 Mineral Resources Within and Adjacent to Bikkuri (Base Case is Highlighted)

Indicated Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	2	1.67	1.45	0.34	0.09	3.55	0.22	0.40
2 g/t	7	1.30	1.16	0.28	0.07	2.81	0.19	0.35
1 g/t	31	0.73	0.71	0.16	0.05	1.65	0.14	0.24
Indicated Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t		0.13	0.12	0.03	0.01	0.28	12	22
2 g/t		0.29	0.26	0.06	0.01	0.62	29	52
1 g/t		0.74	0.71	0.17	0.05	1.67	99	170
Inferred Mineral Resources -Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	8	1.59	1.52	0.36	0.09	3.55	0.20	0.37
2 g/t	27	1.23	1.17	0.25	0.07	2.72	0.16	0.30
1 g/t	112	0.75	0.76	0.15	0.05	1.72	0.14	0.24
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t		0.41	0.39	0.09	0.02	0.92	35	65
2 g/t		1.09	1.03	0.22	0.06	2.40	100	184
1 g/t		2.70	2.74	0.56	0.19	6.19	353	593

1. The Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.

14.7.4 UMT-FW

Table 14.26 provides the total Mineral Resource estimate for mineralisation within the TCU-FW Model. The tabulation includes the FWcpx and the FWpnz resources.

Table 14.26 Mineral Resource Estimates for the TCU-FW Assuming Selective Underground Mining Methods (Base Case is Highlighted)

Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
2.5 g/t	9	1.44	1.66	0.24	0.09	3.43	0.22	0.39
2.0 g/t	20	1.15	1.34	0.19	0.08	2.76	0.19	0.34
1.5 g/t	49	0.88	1.04	0.15	0.07	2.14	0.16	0.29
1.0 g/t	105	0.66	0.81	0.11	0.07	1.65	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
2.5 g/t		0.43	0.49	0.07	0.03	1.02	45	80
2.0 g/t		0.75	0.87	0.12	0.05	1.79	84	153
1.5 g/t		1.39	1.65	0.23	0.11	3.38	169	318
1.0 g/t		2.23	2.73	0.39	0.23	5.58	304	587

1. The Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.

4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.

14.7.4.1 UMT-FWCPX

Table 14.27 provides the portion of the TCU-FW Mineral Resource estimate for mineralisation within the CPX domain.

Table 14.27 Mineral Resource Estimates for the FW-CPX Assuming Underground Mining Methods (Base Case is Highlighted)

Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
2.5 g/t	4	1.34	1.60	0.21	0.09	3.24	0.19	0.40
2.0 g/t	10	1.08	1.29	0.17	0.08	2.63	0.16	0.34
1.5 g/t	27	0.82	1.00	0.14	0.07	2.03	0.14	0.29
1.0 g/t	58	0.64	0.78	0.11	0.06	1.60	0.11	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
2.5 g/t		0.17	0.20	0.03	0.01	0.41	16	35
2.0 g/t		0.33	0.40	0.05	0.02	0.81	34	73
1.5 g/t		0.71	0.86	0.12	0.06	1.74	80	170
1.0 g/t		1.19	1.47	0.21	0.12	2.99	144	317

1. The Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.

5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.
7. The FWcpx domain includes zones of low-grade mineralisation that may permit mass mining methods at a lower cut-off.
8. Mineral Resources in Table 14.27 are included in the tabulations in Table 14.26 and are not additive to that table.

14.7.4.2 UMT-FWPNZ

Table 14.28 provides the portion of the TCU-FW Mineral Resource estimate for mineralisation within the PNZ Domain.

Table 14.28 Mineral Resource Estimates for the FW-PNZ Assuming Selective Underground Mining Methods (Base Case is Highlighted)

Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
2.5 g/t	6	1.51	1.68	0.25	0.08	3.52	0.24	0.39
2.0 g/t	11	1.21	1.39	0.21	0.08	2.88	0.21	0.34
1.5 g/t	22	0.94	1.09	0.16	0.07	2.27	0.18	0.30
1.0 g/t	47	0.69	0.83	0.12	0.07	1.71	0.15	0.26
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
2.5 g/t		0.27	0.30	0.04	0.01	0.63	29	45
2.0 g/t		0.41	0.47	0.07	0.03	0.99	50	81
1.5 g/t		0.68	0.79	0.12	0.05	1.64	89	148
1.0 g/t		1.04	1.26	0.18	0.11	2.59	160	270

1. The Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.

7. Drill intercepts ≥2.0 g/t 3PE+Au suggest selective mining is required. Grade continuity best observed at a 1.0 to 1.5 g/t 3PE+Au cut-off. Discontinuous pods of mineralisation at a 2.0 g/t 3PE+Au are not well defined and additional drilling is required.
8. Mineral Resources in Table 14.28 are included in the tabulations in Table 14.26 and are not additive to that table.

14.7.4.3 Combined Mineral Resources

Table 14.29 provides a summary of the combined Platreef Mineral Resources for the UMT-TCU, UMT-BIK and UMT-FW models.

Table 14.29 Mineral Resources for All Platreef Mineralised Zones (Base Case is Highlighted)

Indicated Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3.0 g/t	204	2.11	2.11	0.34	0.14	4.70	0.18	0.35
2.0 g/t	346	1.68	1.70	0.28	0.11	3.77	0.16	0.32
1.0 g/t	716	1.11	1.16	0.19	0.08	2.55	0.13	0.26
Indicated Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3.0 g/t		13.86	13.86	2.23	0.92	30.86	800	1,597
2.0 g/t		18.66	18.94	3.12	1.23	41.95	1 226	2,438
1.0 g/t		25.63	26.81	4.49	1.82	58.75	2 076	4,108
Inferred Mineral Resources -Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3.0 g/t	225	1.91	1.93	0.32	0.13	4.29	0.17	0.35
2.0 g/t	506	1.42	1.46	0.26	0.10	3.24	0.16	0.31
1.0 g/t	1,431	0.88	0.94	0.17	0.07	2.05	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3.0 g/t		13.78	13.96	2.33	0.94	31.01	865	1,736
2.0 g/t		23.17	23.78	4.26	1.56	52.77	1,775	3,440
1.0 g/t		40.38	43.01	7.81	3.06	94.27	4,129	7,759

1. The Mineral Resources were estimated as of 22 April 2016. The economic inputs used in assessing reasonable prospects of eventual economic extraction and the resource tabulation were rerun on 28 January 2022 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 28 January 2022. The Qualified Person for the estimate is Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The table shows sensitivity to cut-off and the rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to +650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.

6. Totals may not sum due to rounding.
7. Mineral Resources reported in Table 14.29 are included in Table 14.24, 14.25, 14.26 (note that Table 14.27 and 14.28 are included in Table 14.26) and are not additive to those tables.

14.8 Targets for Further Exploration

Beyond the current Mineral Resources, mineralisation is open to expansion to the south and west. Targets for further exploration (exploration targets) have been identified. MTS cautions that the potential quantity and grade of these exploration targets is conceptual in nature. There has been insufficient exploration and/or study to define these exploration targets as a Mineral Resource. It is uncertain if additional exploration will result in these exploration targets being delineated as a Mineral Resource.

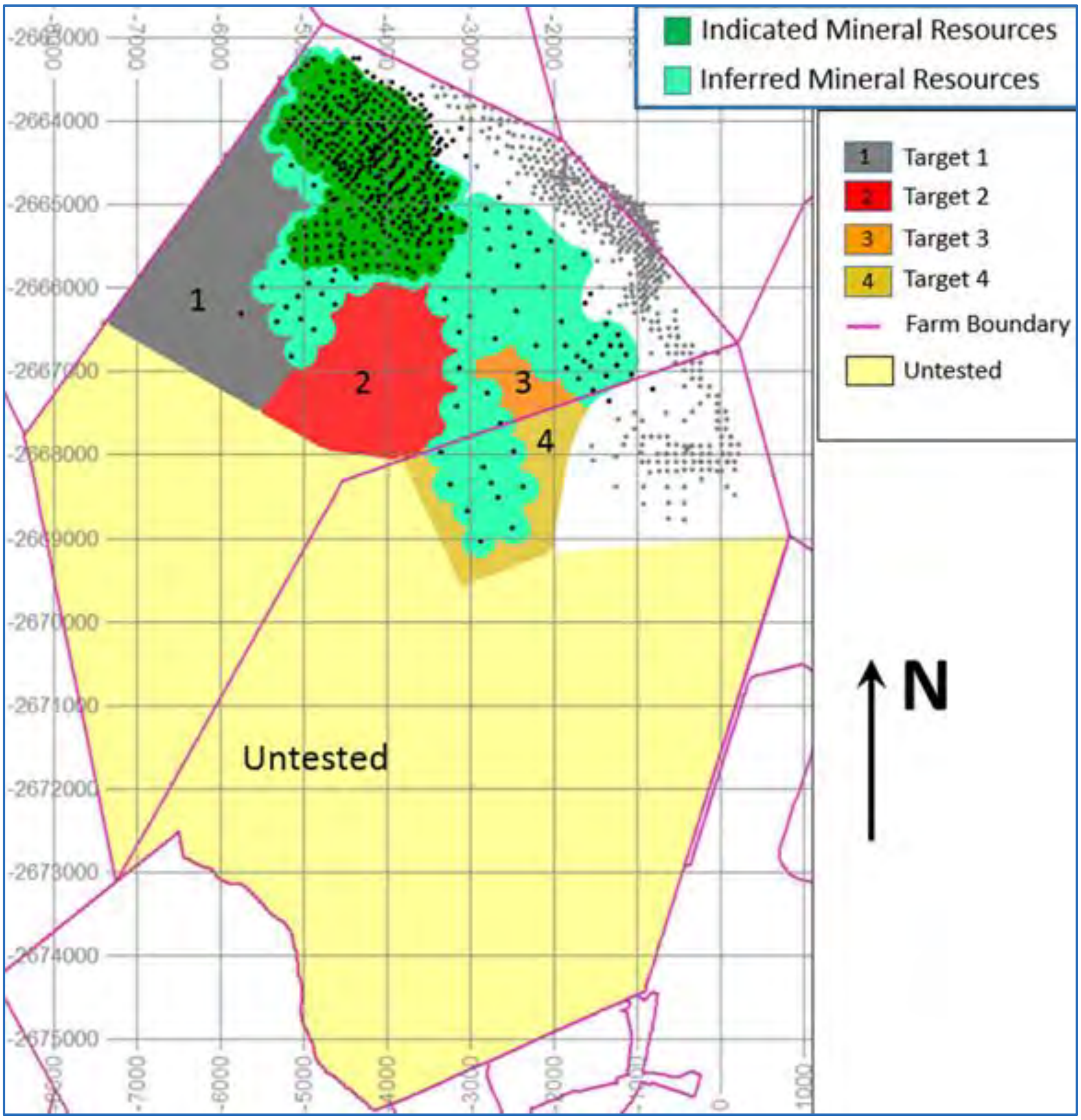
The Bushveld Igneous Complex (BIC) PGE-Ni-Cu deposits have characteristics of lateral continuity over several thousands of metres. Based on this, four exploration targets have been identified (Figure 14.26). Target areas are defined based on the 2016 Mineral Resource Model, and represent currently undrilled extension areas from the model.

- Target 1 could contain 100 to 165 Mt grading 3.1 to 5.2 g/t 3PE+Au (1.3 to 2.2 g/t Pt, 1.5 to 2.5 g/t Pd, 0.18 to 0.30 g/t Au, 0.12 to 0.21 g/t Rh), 0.10 to 0.17% Cu, and 0.22 to 0.36% Ni over an area of 4.1 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 2 could contain 50 to 90 Mt grading 2.9 to 4.9 g/t 3PE+Au (1.3 to 2.1 g/t Pt, 1.4 to 2.3 g/t Pd, 0.19 to 0.31 g/t Au, 0.11 to 0.18 g/t Rh), 0.11 to 0.19% Cu, and 0.23 to 0.39% Ni over an area of 3.3 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 3 could contain 20 to 30 Mt grading 2.6 to 4.4 g/t 3PE+Au (1.2 to 1.9 g/t Pt, 1.2 to 2.0 g/t Pd, 0.19 to 0.32 g/t Au, 0.10 to 0.16 g/t Rh), 0.12 to 0.20% Cu, and 0.23 to 0.39% Ni over an area of 0.5 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.
- Target 4 could contain 10 to 20 Mt grading 2.1 to 3.4 g/t 3PE+Au (1.0 to 1.6 g/t Pt, 0.9 to 1.4 g/t Pd, 0.13 to 0.22 g/t Au, 0.10 to 0.17 g/t Rh), 0.09 to 0.15% Cu, and 0.19 to 0.32% Ni over an area of 1.5 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE+Au mineralisation in drillholes located adjacent to the target.

Beyond these exploration target areas is approximately 48 km^2 of unexplored ground on the property under which prospective stratigraphy is projected to lie. It is not possible to estimate a range of tonnages and grades for this ground without additional drilling.

There is excellent potential for the extent of known mineralisation to significantly increase with further step-out drilling to the southwest.

Figure 14.26 Exploration Target Areas



Wood, 2016

14.9 Comments on Section 14

Mr Kuhl is of the opinion that the Mineral Resources for the Platreef Project, which have been estimated using core-drill data, have been performed to industry best practices (CIM, 2019), and conform to the requirements of the 2014 CIM Definition Standards. NSR values were calculated using the 2022 prices. The results indicated there was no material change to the Mineral Resource. This evaluation confirms the Mineral Resource remains current.

Since the commencement of exploration in the UMT area, iterative Mineral Resource estimates between 2010 and 2015 have led to a progressive increase in the tonnage of Inferred Mineral Resources. With the inclusion of results from the 2014-2015 drill programme in the block model reported herein, higher confidence category upgrades in the classification are supported.

As noted in Section 7, drill data have allowed recognition of the structural regime and interpretation of faults that explain offsets in the subunits on cross-sections. These faults tie in with three sets that have been established in the region.

Mr Kuhl reviewed twin hole drill data, and notes that there are typically large differences in the positions of the top and bottom of the T1 and T2 in twin holes spaced less than 10 m apart. There is good correlation between the position of chromite stringers at the top of the T2. Other chromite stringers, granite veins, pegmatite veins that are actually formed from irregular bodies of intercumulus melt, and massive sulfides show weak or no correlation. There is fair correlation for length of the 1 g/t 3PE intercept, nickel grade and 3PE grade within the T1MZ. The T2MZ is thicker, and there are more assay intervals. The correlation is good to excellent for length of the 1 g/t 3PE intercept, nickel grade and 3PE+Au grade within the T2MZ. The implication for modelling is that the position of grade shell boundaries will be variable, and it will be difficult for mining to follow them.

14.9.1 Considerations for Next Model Update

MTS notes the following considerations for the next model update:

- The T2U and T2L domains will be preserved, in case there are differences in metallurgical responses for these units.
- The presence of chrome stringers is known to enhance grade. Possibly distance from chrome stringers should be used in local domaining.
- The data on positions of grade shell boundaries should be examined to the extent possible to estimate their short-scale variability; the likely accuracies of down-hole surveys should be taken into account, and it is recognised a definitive answer may have to await exposures in underground workings.

MTS recommends the following:

- Re-logging ATS and AMK holes consistent with the new geological interpretation.
- Re-modelling ATS and AMK using the UMT litho-stratigraphic units and interpolation using total nickel and copper.

This will put all models on the same litho-stratigraphic and assay (total) basis.

14.9.2 Uncertainties Implicit in the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Permitting, environmental, legal and socio-economic assumptions.
- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction including:
  - Long-term commodity price assumptions.
  - Long-term exchange rate assumptions.
  - Assumed mining method.
  - Availability of water and power.
  - Operating and capital cost assumptions.
  - Metal recovery assumptions.
  - Concentrate grade and smelting/refining terms.
- Additional metallurgical sampling from specific mineralisation layers may result in changes to the metallurgical recovery and smelter payables assumptions used to evaluate reasonable prospects of eventual economic extraction.
- Unmineralised GV dykes are not included in the geology model. These dykes may result in local over estimations of the volume of the mineralised material, however, any quantification of the dykes would require close-spaced drill data from underground drill stations.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. Dilution and mining recoveries will vary with the geometry (dip, thickness, faulting and or irregularities in contacts) of the mineralisation and the eventual mining method used. These factors can only be estimated after life of mine plans are prepared. Typically, dilution (low-grade or waste materials) ranges from 10–30%, and mining recoveries range from 70% to 100% using the mining methods considered for evaluation of reasonable prospects of eventual economic extraction.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

High-grade platinum group element (PGE) and copper-nickel mineralisation at Platreef occurs within a large area of layered mafic intrusion on a northern limb of the Bushveld Igneous Complex. Economic mineralisation occurs in two main zones within the Turfspruit cyclic unit: The Upper T1 pyroxenite unit and the Lower T2 pegmatoid unit. T1, with thicknesses of 10–80 m, is located directly above T2; however, mineralisation in T1 is, on average thinner and less continuous than in T2. The T2 unit ranges from less than 5 m to over 50 m in thickness. T2 is further broken into upper and lower units. The Upper T2 unit generally has stronger mineralisation than the lower unit, although metal mineralisation often persists through the entire thickness of T2; thus, T2 is well suited for large-scale mechanised mining. The T1 and two T2 units are often separated vertically by a variable thickness of weaker mineralised Lower T1 rock.

The evaluation concentrates on mining the thicker T2 resource, as mineralisation in T1 is thinner and less continuous than in T2: more than 70% of the targeted reserves are contained in T2, while 22% are contained in T1. (The remaining 8% is dilution and other rock types.) While most the targeted reserves are contained within T2, T1 is occasionally mined where the grade of the T1 mineralisation justifies mining it along with T2. T1 is occasionally mined along with T2, where the grade of the T1 mineralisation justifies mining it along with the T2.

15.2 Resource Block Model

The probable mineral reserves available for mining at the Platreef were based on the resource block model (file name: prf2021b.dm) developed by Ivanhoe with supervision by Wood and provided to OreWin.

Model No. prf2021b.dm contains the following:

- 10 m x 10 m x 2 m parent blocks and 10 m x 10 m x 1 m sub-blocks,
- An updated base data template 20 (BDT20) was used to calculate the NSR to use for the mine planning,
- Grades-inverse distance (ID^3),
- Structural domains,
- Lithologic units,
- Classification, and
- Model units.

In addition, as contained in the models provided for the previous studies, the block model includes a variable named 3PE+Au, which is calculated by adding platinum (Pt), palladium (Pd), rhodium (Rh), and gold (Au), resulting in the equation shown below.

$$3PE+Au = [(Pt + Pd + Rh) + Au)]\ g/t$$

The 1 g/t 3PE+Au grade shell was used as a base case for the Mineral Reserve to assess the available Mineral Resource.

For the Platreef 2022 FS work, only the Indicated Resource is used. Inferred metal values are zeroed out in the block model for all Mineral Reserve calculations.

The NSR calculation does not consider costs for mining, milling, or general and administrative tasks. For mining of a resource block to be economic, the NSR value must be high enough to cover these additional costs. The NSR values were calculated and inserted into the model by Ivanplats and OreWin. Metal prices and charges used in the NSR calculation are shown in Table 15.1. Other charges and assumptions used in the NSR calculation are shown in Table 15.2.

Validation of Block Model No. prf2021b.dm was compared to the interim model from the studies prior to feasibility. NSR17 and NSR20 used all the same metal prices and costs. The only difference between NSR17 to NSR20 is the concentrator recoveries. The comparison is illustrated below in Table 15.3 and Figure 15.1.

Table 15.1 Metal Prices Used in NSR Calculations (NSR17 and NSR20)

Metal	Selling Price ($)
Au	$1,300.00/oz
Pd	$815.00/oz
Pt	$1,600.00/oz
Rh	$1,500.00/oz
Cu	$3.00/lb
Ni	$8.90/lb

Table 15.2 Charges and Other Assumptions Used in NSR Calculations

Item	Assumption
Concentrator Recovery*	Varies by metal, rock type, and feed grade
Concentrate Transportation Charge	$35.00/t
Payable Metal (smelting and refining)	82%
Royalty	5% of payable

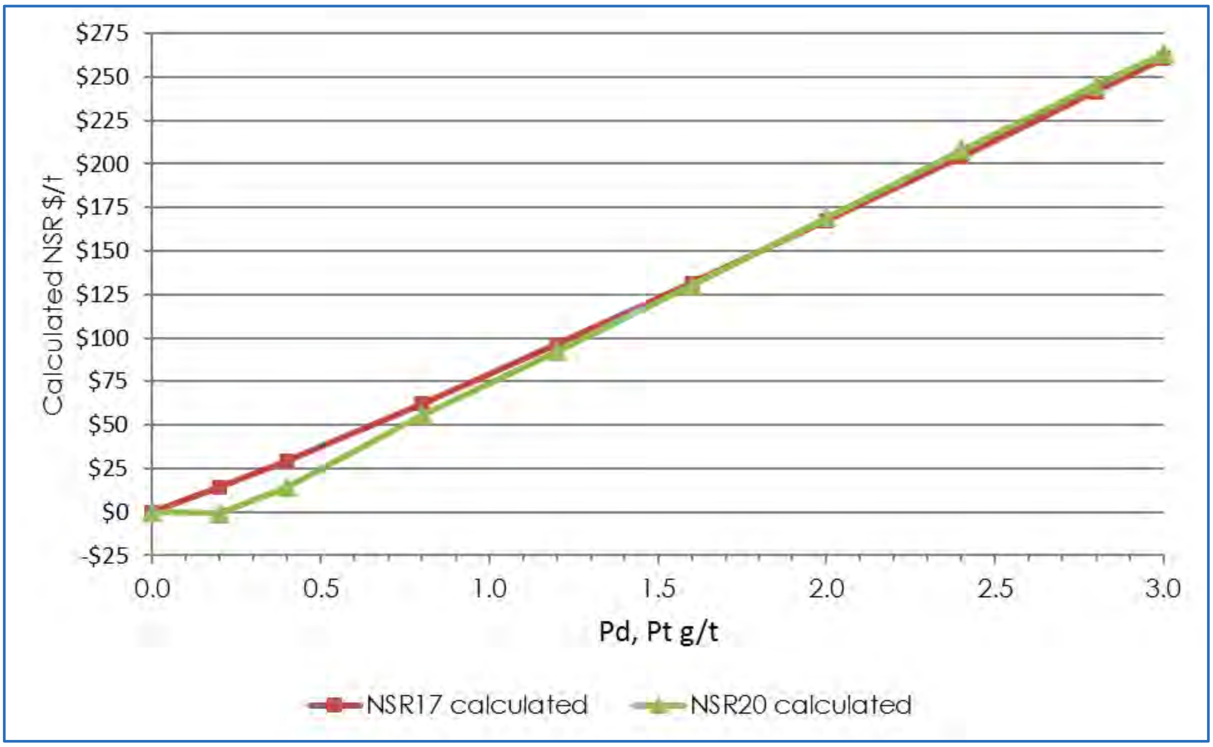
*The only change between NSR17 and NSR20 is in the concentrator recoveries.

Table 15.3 NSR20 / NSR17 Comparison

Input	Grade Ratio	Set 1	Set 2	Set 3	Set 4	Set 5	Set 6	Set 7	Set 8	Set 9	Set 10
NSR20 Calculated	–	$0.00	–$0.87	$14.20	$55.57	$92.34	$130.16	$169.38	$207.91	$245.22	$263.49
NSR17 Calculated	–	$0.00	$14.22	$29.52	$61.91	$96.06	$131.58	$167.59	$204.32	$241.63	$260.47
Input Metal Grades	–	–	–	–	–	–	–	–	–	–	–
Cu (%)	0.10	0.00	0.02	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.30
Ni (%)	0.20	0.00	0.04	0.08	0.16	0.24	0.32	0.40	0.48	0.56	0.60
Pt (g/t)	1.00	0.00	0.20	0.40	0.80	1.20	1.60	2.00	2.40	2.80	3.00
Pd (g/t)	1.00	0.00	0.20	0.40	0.80	1.20	1.60	2.00	2.40	2.80	3.00
Au (g/t)	0.15	0.00	0.03	0.06	0.12	0.18	0.24	0.30	0.36	0.42	0.45
Rh (g/t)	0.070	0.000	0.014	0.028	0.056	0.084	0.112	0.140	0.168	0.196	0.210

Note: Typical metal ratios with respect to Pt were used to generate a test data set for an NSR20 & NSR17 calculation comparison.

Figure 15.1 NSR20 and NSR17 Comparison Graph



OreWin, 2021

15.3 Cut-off Grades

Global Net Smelter Return Evaluation

In the Platreef 2015 PFS, a marginal cut-off grade analysis was completed by mining method, the resulting cut-off grade ranged from $47–$58/t. The marginal NSR cut-off must be high enough to pay for mining, processing, and on-site general and administration costs. Ivanplats' operational philosophy required additional allowances for capital and profit margin to be included in the NSR cut-off evaluation. As a starting point, and based on the Platreef 2015 PFS operating costs, NSR values of $80/t and $100/t were selected for evaluation. As part of the NSR evaluation further optimisation on a declining NSR cut-off approach was evaluated using NSR values from previous optimisation studies. These studies focused on the thickness of the deposit and the tonnage based on different cut-offs.

Based on a 30-year mine life and an achievable 4 Mtpa for the Platreef 2017 FS, the analysis was performed to determine the most economic tonnage to support the mine plan. The output from that analysis was used for the Platreef 2022 FS. The resource evaluation was performed on the grade shells and tonnages by height for each cut-off were determined. After this analysis, stope shapes were created for the high NSR cut-offs ($155/t and $130/t). The outliers that could not be economically accessed were removed, and to get the required tonnage for the 30-year mine life, an additional $100/t NSR cut-off resource was brought in, with stope shapes developed for those. The highest-grade stopes near the current mining areas were selected to achieve the tonnage required for the mine plan.

Based on the differential between the 2017 marginal cut-off grades and the applied mining cut-off grades, the 2017 Reserve cut-off grades were still deemed to be valid.

Table 15.4 summarises the global Indicated Resource tonnes, NSR, and 3PE+Au for the various NSR cut-offs.

Table 15.4 Resource Evaluation at Various Cut-offs

NSR Cut-off	Indicated Resource (kt)	NSR20_2P ($/t)	3PE+Au (g/t)
$80/t	327,790	138.76	3.81
$100/t	239,450	158.36	4.36
$130/t	155,935	183.08	5.06
$155/t	101,848	204.42	5.69

15.4 Mineral Reserve

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in the Platreef 2017 FS. Only Indicated Mineral Resources have been used for determination of the Probable Mineral Reserve.

The Mineral Resource block model also includes the NSR variable. NSR calculation formulas and metal prices used in the block model were provided by Ivanplats. NSR is the dollar value of the metals recovered from a tonne of rock minus the cost for transportation of concentrate to the smelter, royalties, smelting and refining charges, and other smelter deductions. These parameters were used to calculate the NSR in units of $/t for each cell in the Platreef 2017 FS Mineral Resource block model.

Mineral Reserves were calculated from the block model using the combination of stope optimiser and generated grade based on the economic NSR cut-off values. Two stoping methods (longhole, and Drift-and-Fill) were selected for the project as they satisfy the following design criteria:

- Maintain maximum productivities by incorporating bulk-mining methods and operational flexibility, which will result in lower operating costs.
- Maintain high overall recovery rates.
- Minimise overall dilution.
- Prevent surface subsidence from underground mining.

Calculated marginal cut-off grades for each mining method, exclude capital recovery and profit margin.

In order to increase initial mined grades and provide increased revenue early in the mine life, areas designated as early mine production an NSR cut-off value of $130 was used for the identification and design of the Longhole Stopes. Areas within the early production, that could not support a $130 NSR cut-off, were in-filled by including $100/t NSR cut-off stopes. The cut-off value was lowered to $100/t for areas mined later in the mine life. Lowering the cut-off grade ensures that adequate reserves are available to satisfy Ivanhoe's requirement of a 30-year mine life after mill start-up. Stope End Slash cuts and Drift-and-Fill shapes were generated using a $155/t cut-off.

In Phase 1 (700 ktpa) of the Platreef 2022 FS, stopes with the 3PE+Au grade greater than 4.5 g/t were targeted. Also, development with the 3PE+Au grade greater than 4.0 g/t was counted as ore. This provided increased revenue in early years of the mine.

A definitive mine plan based on detailed stope layouts supports the mineral reserve. Due to irregularities in the geometry of the mineralised zones, not all material meeting cut-off grade can be mined without incurring some dilution. Due to inefficiencies in final mining recovery from the stopes, small amounts of mineralised material are lost during final stope cleanout, and additional losses may occur in transit from the stopes to the mill. Hence, a mining recovery factor is applied to the diluted resources to account for these losses.

The design parameters for the mining areas are based on geotechnical recommendations provided by SRK. The stope orientation and dimensions are based on a recommended maximum hydraulic radius of 8 m. SRK divides the deposit into five major geotechnical zones, with recommendations for the best stope orientation within these zones.

A series of well-defined stope shapes was generated for the entire mining area. After completion of initial stope designs, the deposit was segregated into 17 mining zones. These stope shapes were then used to query the block model and report tonnes and grades within the shapes.

The variability of factors related to mining, metallurgy, infrastructure, permitting, and other areas relevant to the mining reserve calculation, the cost-per-tonne differential between the calculated marginal NSR cut-off grade ($47.71/t–$58.53/t) and the production schedule NSR cut-offs ($80/t and $155/t), provides a buffer from potential future negative impacts of these factors. The differential between the 2017 marginal cut-off grades and the applied production schedule cut-off grades, the applied 2017 Reserve cut-off grades were still deemed to be valid for Reserve purposes.

The Platreef 2022 FS cost estimates have been done to a feasibility study level of accuracy. For further detail on cost estimates, refer to Section 21

Table 15.5 and Table 15.6 show the total diluted and recovered Probable Mineral Reserve for Platreef.

Table 15.5 Platreef Probable Mineral Reserve – Tonnage and Grades as at 26 January 2022

Method	(Mt)	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Ore Development	11.0	142.4	1.79	1.85	0.27	0.12	4.03	0.15	0.31
Longhole	93.9	152.2	1.88	1.95	0.29	0.13	4.25	0.16	0.33
Drift-and-Fill	20.3	183.6	2.30	2.25	0.37	0.15	5.07	0.18	0.37
Total	125.2	156.4	1.94	1.99	0.30	0.13	4.37	0.16	0.34

1. Mineral Reserves have an effective date of 26 January 2022. The Qualified Person for the estimate is Curtis Smith (OreWin), B. Eng., MAusIMM (CP).
2. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
3. Metal prices used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

Table 15.6 Platreef Probable Mineral Reserve – Contained Metal as at 26 January 2022

Method	(Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Ore Development	11.0	0.6	0.7	0.1	0.04	1.4	37	76
Longhole	93.9	5.7	5.9	0.9	0.40	12.8	336	687
Drift-and-Fill	20.3	1.5	1.5	0.2	0.10	3.3	83	166
Total	125.2	7.8	8.0	1.2	0.54	17.6	455	929

1. Mineral Reserves have an effective date of 26 January 2022. The Qualified Person for the estimate is Curtis Smith (OreWin), B. Eng., MAusIMM (CP).
2. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
3. Metal prices used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

16 MINING METHODS

16.1 Mine Geotechnical

Much of the work done for the Platreef 2017 FS was used in the Platreef 2022 FS. The primary aim of the 2017 investigation was to increase the level of confidence in the current geotechnical database and to undertake various analyses, based on data from the mine site and laboratory testing, to provide geotechnical design parameters and optimise the mine design going forward. Following the completion of the 2017 FS, a detailed geotechnical investigation for the updated mine design was carried out. This work focused on the initial production period, with specific reference to the Drift-and-Fill mining (DF). No additional geotechnical data was provided, except for mapping conducted by Ivanplats during the development of vertical shaft No. 1 and respective station developments.

The geotechnical investigation was based on all available geotechnical and structural data, and included data specifically derived from geotechnically logged boreholes. Laboratory rock strength testing and stress measurement testing was also conducted to better understand the rock properties and the local stress regime. Local and regional seismicity have also been assessed. From the study, geotechnical design parameters have been derived to manage potential geotechnical risks that the mine may face. These parameters govern stope and mine access design and include the backfill and support requirements. The mine design has been reviewed and is generally in line with the geotechnical parameters provided.

Overall, the Tshukudu fault remains a major geotechnical hazard as it is often characterised by very poor-quality rock. Development through the fault should thus be planned carefully to avoid delays and costs. As the Tshukudu fault strikes from north to south and traverses the entire lease area, some development through the Tshukudu fault will be essential to provide access to ore to the west of the fault. Specialised support comprising resin injection, arch sets and void filling to be carried out by a specialist contractor is recommended for this case. An indication of the slow rate of this development is also provided.

Following the identification of the Tshukudu fault it has been established that the type of alteration within the fault is variable, indicating that improved characteristics of the fault zone in some areas may exist. It is therefore possible to develop through the fault in these cases with fewer delays and less intensive support, provided that there is no water ingress. Geotechnical drilling will be required to delineate and characterise the Tshukudu fault during implementation.

16.1.1 Study Methodology

The geotechnical investigation is based on an assessment of all the available geotechnical data, laboratory test data, the projects geological setting, major faulting, the proposed mine design and proposed stope orientations.

Laboratory rock strength testing and stress measurement testing was conducted for the following purposes:

- Rock mass characterisation.

- Empirical stope design.
- Determination of Hoek-Brown parameters for numerical modelling.

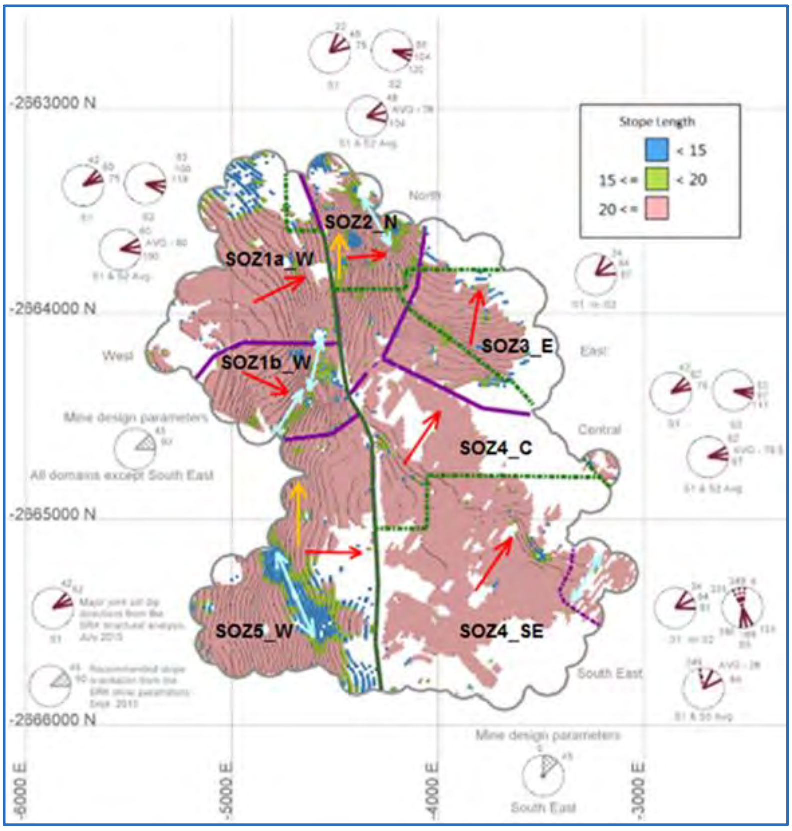
Significant input data were derived from 80 geologically and geotechnically logged boreholes, including 15 new geotechnical holes drilled to support the Platreef 2017 FS. Rock mass classification was conducted on all available geotechnical data and was assigned to a computer-generated geotechnical block model. The geotechnical block model was validated against borehole data.

A structural analysis that focused on 34 boreholes was carried out to determine the major joint sets and their relationship to major faults in the project area. This was done to optimise mining directions by establishing structural domains and areas of potential instability based on major faults and the orientation of major joint sets.

The stability of the longhole mining stopes was assessed using Mathews and Potvin's stability graph method, using the lower bound 20 percentile rock mass quality values. The stope height and width were determined in consultation with Ivanplats and OreWin. Following the analysis, maximum unfilled stope lengths were recommended for the different areas of the mine.

A risk assessment of the block failure potential of the back area and side walls of proposed transverse and longitudinal mining stopes was conducted using computer software JBlock. The failure potential of blocks was simulated in JBlock for the back, left wall and right wall of mining stopes of various proposed stope orientation zones (SOZ) considering Platreef's structural domains, for a vertically dipping rectangular excavation (Figure 16.1).

Figure 16.1 Proposed Stope Orientations



A backfill strength analysis was performed and backfill strength requirements outlined. Bulkhead specifications were also provided.

A preliminary stress analysis was carried out using three-dimensional elastic modelling of the Platreef 2015 PFS mining layout and sequence to determine the risk of stress damage. The results of the analysis were used to inform the geotechnical design parameters.

Geotechnical design parameters were derived to manage potential geotechnical risks that the mine may face. Parameters were outlined for the mine design and mine access design and include the backfill and support requirements.

The Platreef 2022 FS utilises the April 2016 mine design. It was reviewed to ensure that it complied with the geotechnical design parameters and assessed using three-dimensional elastic modelling.

16.1.2 Geotechnical Investigation Results

From the stress testing conducted on samples obtained from seven boreholes spread across the project area, it is observed that the pre-mining stress state at all locations tested is at a moderate to low level, relative to the strength of the rock. The maximum horizontal to vertical stress is 1.3. While the major geological structures identified on the Platreef property have not been associated with any large magnitude tectonic earthquakes, it should be kept in mind that, the planned mining will cause significant stress changes along the major geological structures, which are likely to induce seismicity.

Rock strength properties have been determined for each stratigraphic unit within the project area, which form a comprehensive data set (Table 16.1 and Table 16.2). The results from testing indicate that the mean strength for the Main Zone (MZ), NC1, T1, T2U, T2L, NC2, FAZ and PNZ is 241 MPa, 264 MPa, 189 MPa, 166 MPa, 163 MPa, 184 MPa, 173 MPa and 234 MPa, respectively.

Table 16.1 Design Rock Properties

Lithology	MZ	NC1	T1	T2U	T2L	NC2	FAZ	PNZ
No. of Samples	102	49	68	58	47	14	87	43
UCS Mean	241	264	189	166	163	184	173	234
UCS Mean – Std. Dev.	178	204	148	117	127	157	125	178
UCS Mean + Std. Dev.	304	325	230	214	199	212	222	290
mi	20	21	10	10	10	14	11	17

An important input parameter in the analysis of rock mass behaviour in numerical models is the rock mass modulus, which was calculated from the intact elastic or Young's Modulus (Table 16.2). To give a practical representation of the mine environment the Young's Modulus was downgraded to the rockmass modulus using the relationship published by Hoek & Diederichs (2006). A value of 68 GPa was used in the three-dimensional elastic modelling.

Table 16.2 Intact Elastic Material Properties

Strat	No. of Tests	Results	Minimum	Mean	Maximum	Std. Dev.
MZ	17	Poisson's Ratio	0.16	0.30	0.37	0.05
		Young's Modulus (GPa)	52	89	103	13
NC1	12	Poisson's Ratio	0.27	0.33	0.39	0.03
		Young's Modulus (GPa)	86	96	130	12
T1	18	Poisson's Ratio	0.23	0.29	0.36	0.04
		Young's Modulus (GPa)	71	104	136	23
T2U	13	Poisson's Ratio	0.12	0.28	0.38	0.07
		Young's Modulus (GPa)	57	97	140	28
T2L	9	Poisson's Ratio	0.27	0.30	0.31	0.02
		Young's Modulus (GPa)	59	67	77	8
NC2	0	Poisson's Ratio	*	*	*	*
		Young's Modulus (GPa)	*	*	*	*
FAZ	6	Poisson's Ratio	0.24	0.31	0.43	0.07
		Young's Modulus (GPa)	62	102	131	29
PNZ	12	Poisson's Ratio	0.14	0.28	0.37	0.07
		Young's Modulus (GPa)	61	98	134	27

* No UCM test results.

The Norwegian Geotechnical Institute's Q-system was utilised to facilitate the derivation of Q' values for the rock mass per geotechnical interval for each stratigraphic unit. Q' values were determined for use in the stope design and for the development support design. Based on the use of this rock mass classification system, it is observed that there is little variability in the overall quality of the rock mass, and that the rock mass can generally be described as "good". The frequency number and percentage of data are expressed in terms of the sum of the logged core intervals satisfying the bin criteria, to normalise the varying core intervals while providing accurate statistical reporting. The Q' distribution in each mining domain shows that data is concentrated between the "fair" to "very good" classes with the orebody and footwall domains extending into the "poor" classes.

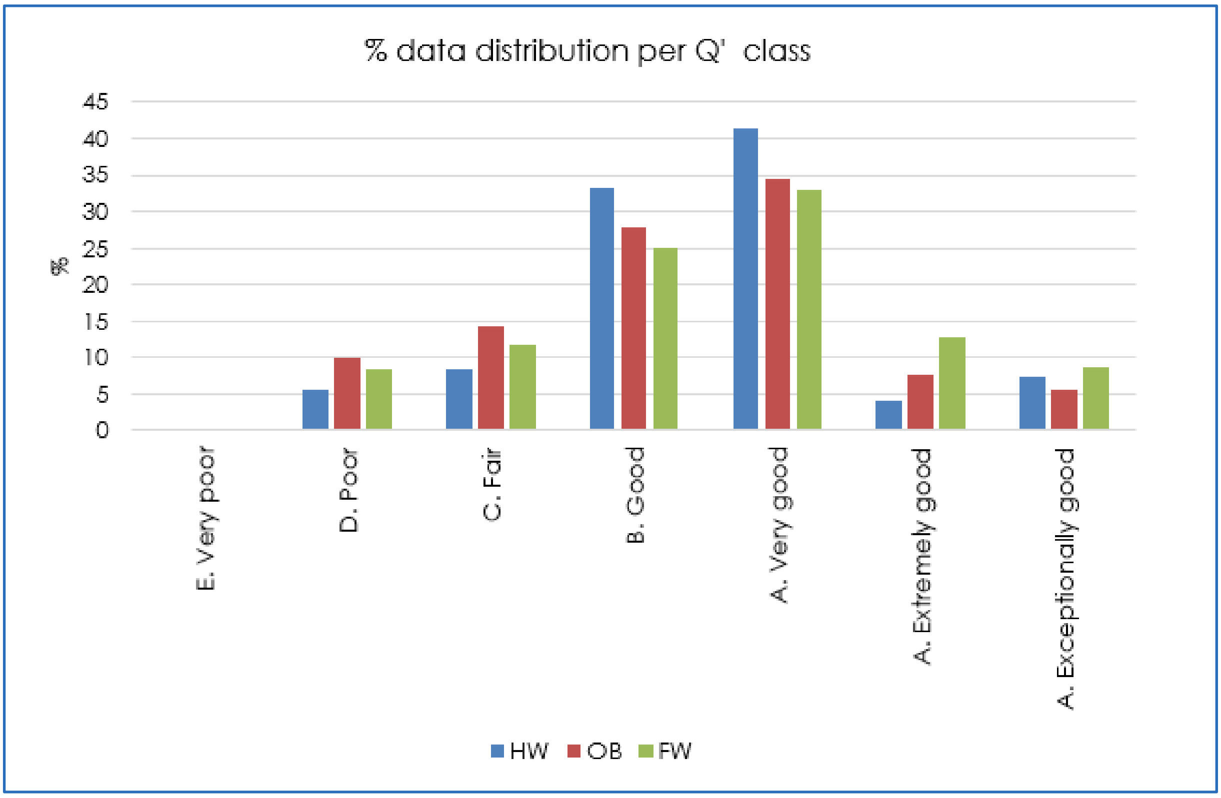
A summary of the rock mass classification results are presented in Table 16.3 and a graphical representation of the data distribution per Q' class is presented in Figure 16.2.

While the rock mass is generally of a "good" quality, geological faults and areas of poor ground do exist. A geotechnical block model has therefore been created to identify areas of "poor" ground Figure 16.3. Placement of permanent structures within these areas should be avoided.

Table 16.3 Summary of Rock Mass Classification Results

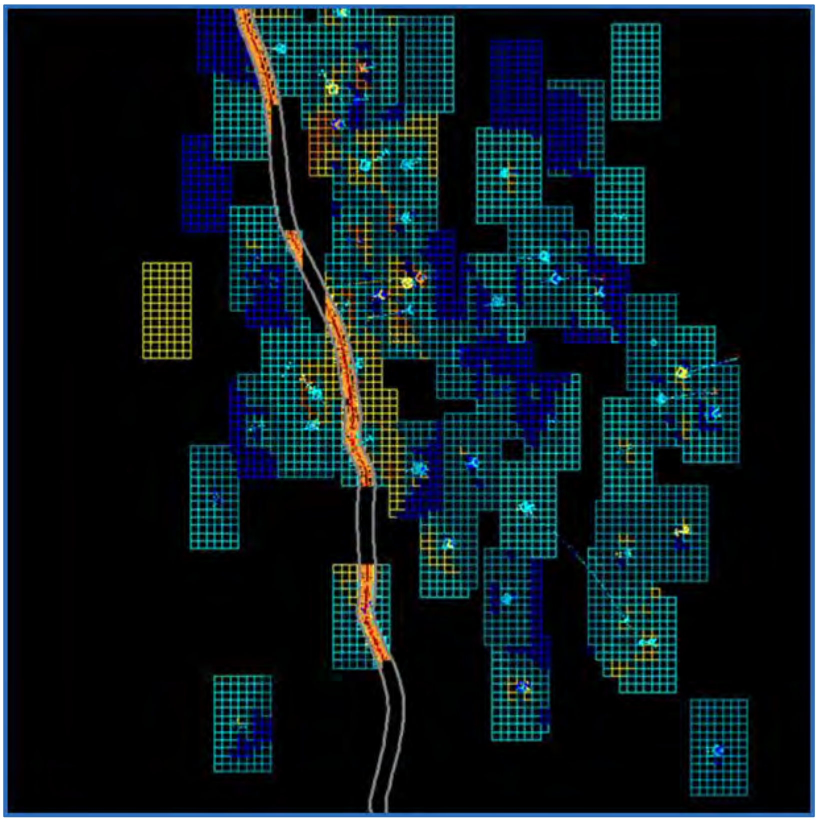
Domain	No. of Intersections	Q'					
		Min	Mean	Max	50th Percentile	20th Percentile	80th Percentile
HW	581	0.2	45.6	150	20.9	7.0	100
OB	197	0.3	41.8	150	17.2	5.9	75
FW	599	0.1	55.6	150	33.3	8.6	133

Figure 16.2 Distribution of Q' Classes per Mining Domain



SRK, 2021

Figure 16.3 Geotechnical Block Model

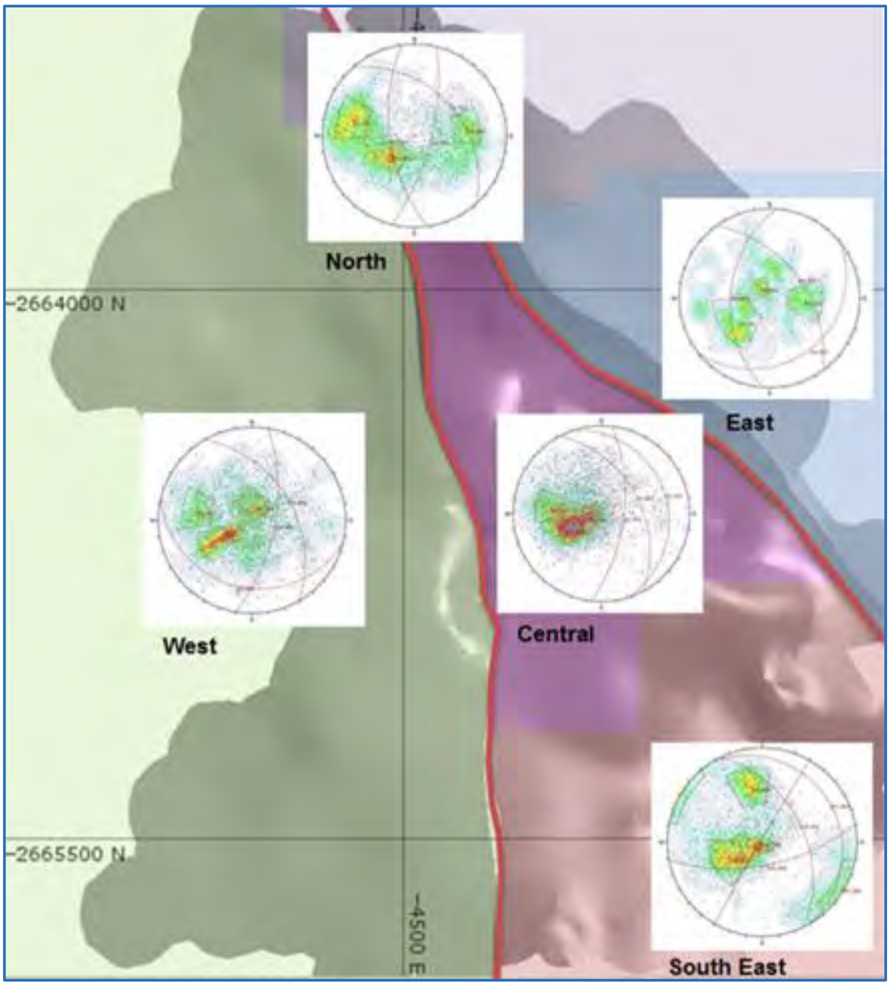


SRK, 2017

From the structural analysis five structural domains (north, west, central, south-east and east) and six joint sets have been identified for the Platreef project (Figure 16.4). Based on the analysis the following is observed:

- Joint Set 1 strikes parallel to the Nkwe fault system; however, Joint Set 1 is flatter dipping at 20–65°.
- Joint Set 2 is an easterly dipping joint set present in the north, west and central domains. Joint Set 2 is parallel to the Kibaran-aged structures which form part of a system of closely spaced extensional faults that are developed across the central and western part of the project area. The Tshukudu and Nyati faults are included in this group.
- Due to the prominence of Joint Set 1 (across the project area) and Joint Set 2 (in the north, west and central domains), stopes mined perpendicular to these orientations will be more stable.

Figure 16.4 Stereographic Projections per Structural Domain



SRK, 2017

16.1.3 Analysis and Design Results

From the keyblock analysis using JBlock, the results for the transverse stopes indicated that the expected linear over break from geological structures is low and is unlikely to influence the overall project. Based on the results, it is concluded that there will be a lower risk in excavating the transverse stopes in the orientations proposed compared with the longitudinal stopes (Table 16.4 and Table 16.5).

Table 16.4 JBlock Results – Transverse Stopes

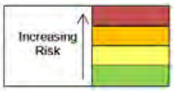
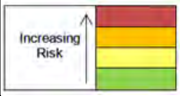
	Total No. of Blocks	No. of Failed Blocks	Failed Blocks (%)	Failed Block Area (%)	Expected Linear Over break (m)	Max Volume (m^3)	Max Area (m^2)	Max Height (m)
SOZ1a_W	751,515	1,754	0.23	0.53	0.004	344	90	8
SOZ1b_W	772,951	5,477	0.71	1.45	0.013	552	143	11
SOZ2_N	440,328	7,712	1.75	4.63	0.046	681	166	11
SOZ3_E	612,899	9,454	1.54	1.92	0.014	388	115	9
SOZ4_SE	663,308	13,129	1.98	4.57	0.034	644	178	9
SOZ4_SE no J6	978,307	7,884	0.81	2.86	0.017	1041	237	11
SOZ4_C	804,185	6,207	0.77	0.64	0.004	120	76	5
SOZ5_W	763,734	2,798	0.37	1.05	0.008	379	129	9

Table 16.5 JBlock Results – Longitudinal Stopes

	Total No. of Blocks	No. of Failed Blocks	Failed Blocks (%)	Failed Block Area (%)	Expected Linear Over break (m)	Max Volume (m^3)	Max Area (m^2)	Max Height (m)
SOZ1b_W	761,415	34,018	4.47	4.52	0.021	255	179	8
SOZ2_N_150	465,636	26,157	5.62	7.74	0.056	625	193	9
SOZ2_N_000	512,056	42,259	8.25	7.63	0.048	463	198	9
SOZ5_W_155	759,584	46,778	6.16	6.41	0.038	438	139	10
SOZ5_W_000	512,465	18,506	3.61	3.16	0.024	369	122	7

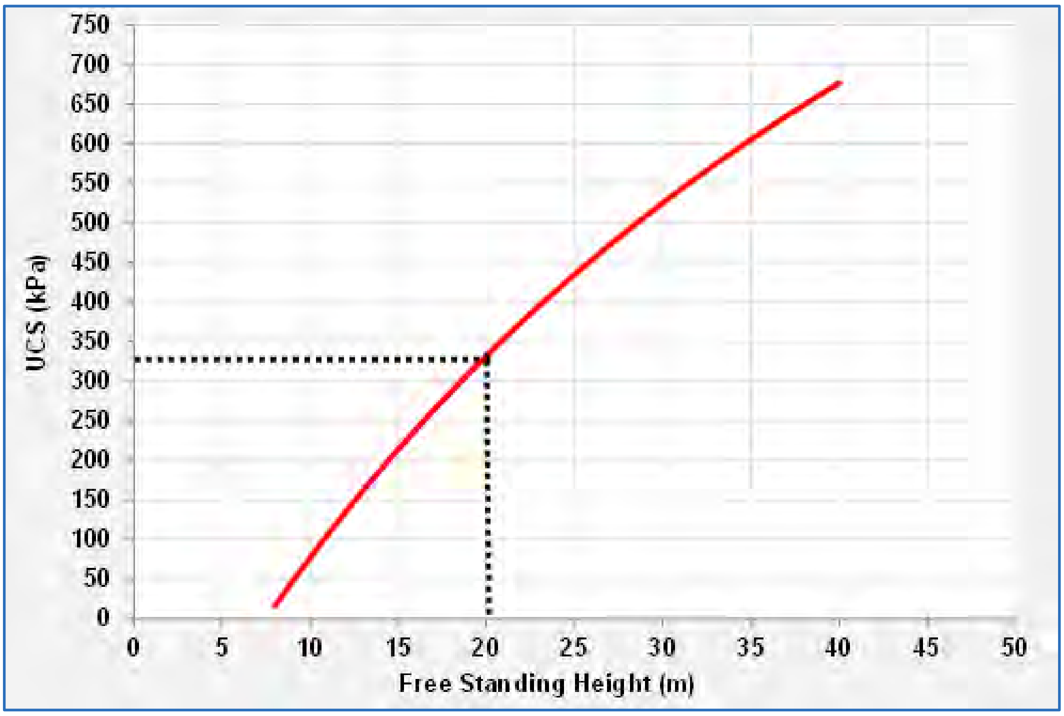
The stability of the mining stopes was assessed using Mathews and Potvin's stability graph method, taking into consideration the rock mass quality (Q'), rock stress, rock strength, joint orientations and the orientations of the stope back and walls. The interaction of joints with the stope walls has a significant influence on stope stability. The recommended unfilled stope lengths are shown in Table 16.6. with the transverse stope lengths likely to range between 20 m and 60 m for a stope height of 20 m. During mining, the unfilled stope length will vary depending on the assessment of local ground conditions and stope overbreak experienced. Backfill will need to be placed in stopes before the maximum wall hydraulic radius is reached.

Table 16.6 Unfilled Stope Length

Stope wall	Q'	N'	HR (m)	Unfilled stope length (m)
SOZ2_N Back (OB)	6.9	1.6	3.6	14
SOZ2_N Back (HW)	10.3	4.0	4.8	26
SOZ2_N North wall	6.9	4.4	4.9	19
SOZ2_N South wall	6.9	15.1	7.7	60
SOZ A1_W Back (OB)	6.9	0.8	2.6	8
SOZ A1_W Back (HW)	10.3	2.1	3.9	17
SOZ A1_W North wall	6.9	10.9	6.8	42
SOZ A1_W South wall	6.9	19.1	8.4	60
SOZ4_C Back (OB)	6.9	1.6	3.5	13
SOZ4_C Back (HW)	10.3	4.1	4.8	26
SOZ4_C North wall	6.9	3.4	4.5	16
SOZ4_C South wall	6.9	18.2	8.2	60
SOZ3_E Back (OB)	6.9	1.7	3.5	13
SOZ3_E Back (HW)	10.3	4.2	4.8	27
SOZ3_E North wall	6.9	7.3	5.9	29
SOZ3_E South wall	6.9	7.0	5.9	28
SOZ4_SE Back (OB)	6.9	1.5	3.4	12
SOZ4_SE Back (HW)	10.3	4.0	4.7	26
SOZ4_SE North wall	6.9	3.5	6.7	40
SOZ4_SE South wall	6.9	1.3	6.7	40
SOZ B1_W Back (OB)	6.9	0.8	2.6	8
SOZ B1_W Back (HW)	6.9	1.5	3.4	12
SOZ B1_W North wall	6.9	22.3	8.8	60
SOZ B1_W South wall	6.9	20.3	8.5	60
SOZ5_W Back (OB)	6.9	0.8	2.7	8
SOZ5_W Back (HW)	6.9	1.5	3.3	12
SOZ5_W North wall	6.9	21.8	8.7	60
SOZ5_W South wall	6.9	21.5	8.7	60

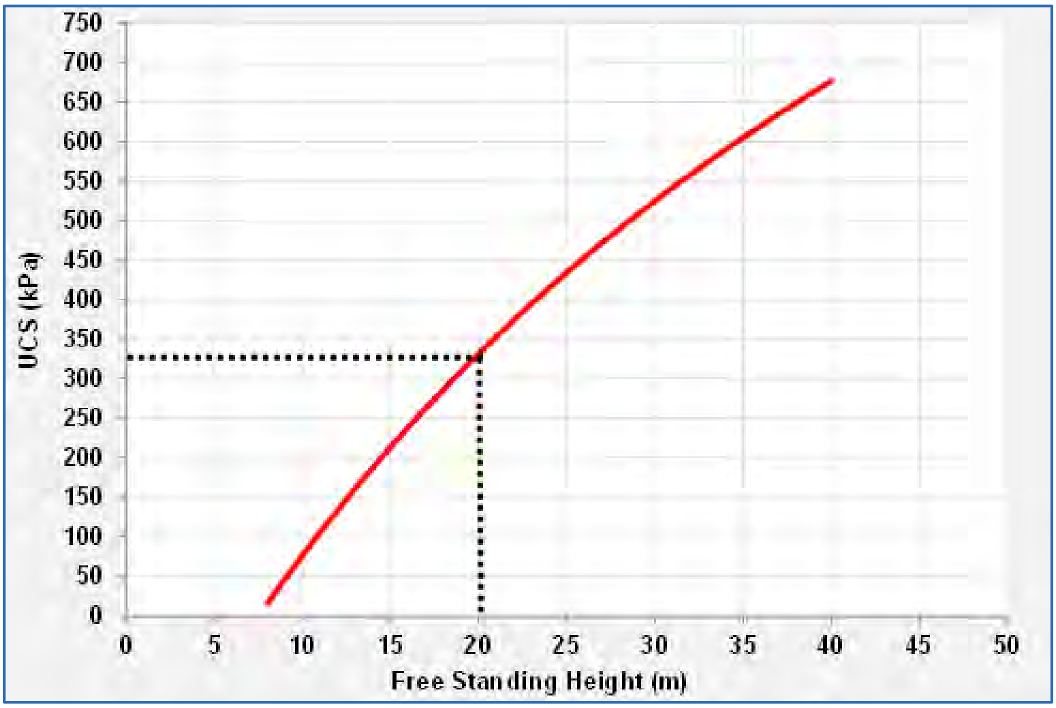
Figure 16.5 provides the strength requirement for Drift-and-Fill mining. Figure 16.6 and Table 16.7 indicates the backfill strength requirements for longhole stopes. The bulkhead specifications are listed in Table 16.8.

Figure 16.5 Proposed CRF Free-standing Strength Requirements for the Drift-and-Fill



SRK, 2021

Figure 16.6 CPF Free-standing Strength Requirements for the Longhole Stope Sidewall



SRK, 2021

Table 16.7 Summary of the Backfill Strengths for the Longhole Stopes

Type	Strength and curing time	% Volume
Primary plug pour	150 kPa at 2 days	30% of primary volume
Primary bulk pour	335 kPa at 28 days	45% of primary volume
Secondary plug pour	150 kPa at 2 days *	30% of primary volume
Secondary plug pour	150 kPa at 28 days	45% of primary volume
Capping pour all stopes	500 kPa at 28 days	25% of primary volume

Table 16.8 Bulkhead and Backfill Specifications for Longhole Stoping

Bulkhead specification	Value
Maximum stope entrance dimensions (with overbreak)	5 m x 5 m
Plug pour height	6 m (1 m above stope drive)
Plug length	6 m
Shotcrete compressive strength at the time of backfill placement	25 MPa @ 7 days
Shotcrete curing time	48 hours
Plug pour curing time	48 hours
Plug pour binder content	5 %
Backfill design parameters	Value
Saturated density	2.12t/m^3
Backfill friction angle	17°

16.1.4 Geotechnical Design Parameters

Longhole stoping will be used where the thickness of the ore zones exceeds 18 m, with Drift-and-Fill being used in narrower portions (less than 10 m thick). Longhole stopes should not exceed a span of 15 m. Drifts should not be more than 6 m wide and should be developed in an arched profile to limit support requirements. The maximum height of drifts is dependent on safe operating practice. Mining will be performed using highly productive mechanised methods with Cemented Rock Fill (CRF) for the drift and fill stopes and Cemented Paste Fill (CPF) for the longhole stopes.

Backfill will need to be placed in stopes before the maximum wall hydraulic radius is reached Table 16.6. Rock mass ratings can be obtained from the geotechnical block model. During mining, the unfilled stope length will vary depending on the assessment of local ground conditions and stope over break experienced.

Backfill in primary longhole stopes must have a minimum strength as specified in Figure 16.5and Table 16.7 to ensure that the backfill is stable during the mining of secondary stopes. There are no strength requirements for secondary stopes and either low strength paste fill of waste rock can be used.

The Drift-and-Fill mining layout requires efficient tight filling to ensure that stopes with more than one lift do not experience roof failure. Loose fill will necessitate the use of long anchors. The inelastic analysis of a conceptual Drift-and-Fill layout highlights the importance of tight filling. Where imperfect tight filling has occurred, it will be necessary to install intense support comprising 6 m long anchors to ensure the safety of personnel in the drift. It will be important to make sure that additional support is always available. If the filling is completely ineffective, it will not be possible to extract tertiary drifts safely. It is also important to ensure that the outer drifts in a block are always mined first.

During the early phases of mining, stress damage will be very limited, and the standard support will suffice. As the overall extraction increases, stress concentrations will increase, and significant stress damage and seismicity can be anticipated, which will need to be managed.

A mine-wide seismic system should be installed within the first five years of mining to monitor the seismic response to mining.

Stope development (drilling access, mucking access, drilling levels and drifts) should be carried out "just in time". This is necessary to avoid damage due to changes in stress, particularly in secondary stopes and benches.

Stope brow support may be required to cater for stress and blast damage (see Table 16.9 and Table 16.10).

Excavations should be separated by twice the combined width to prevent excessive stress interaction.

Tunnels at depth will require S2 support during excavations and S3 support during stoping (see Table 16.9 and Table 16.10).

Long-term service excavations such as ramps need to be sited away from the isolated pillars.

The mining access sublevels must be horizontally offset at least 15 m away from mining. Longer term excavations such as ramps and main level haulages must be sited at least 30 m beneath planned mining.

During the mining of closing pillars, the mining access sublevels will be subjected to significant stress damage (0.5–1.5 m depth of fracturing) and associated seismicity. In these instances, additional dynamic support (see Table 16.9 and Table 16.10) will be required, unless the closing pillar can be avoided through the mining sequence.

Isolated pillars will be highly stressed and excavations beneath these pillars will be subjected to intense stress damage (depth of fracturing >1.5 m) and a high potential risk of seismicity. These should be minimised through improved mining sequences.

Shaft pillars must be large enough that the stress levels in the centre of the pillar do not cause damage to the shaft and critical excavations. The combined shaft pillar radius should be 250 m for Shafts 1 and 2 and 200 m for Ventilation Raise 1, Ventilation Raise 2 and Ventilation Raise 3. The vertical induced strain calculated along the shafts and raises for each mining step indicates that it will be stable throughout the life of mine and no damage is anticipated according to the strain criterion.

The shaft pillars may be extracted at the end of the life of mine. Approximately 60–70% of the shaft pillar can be extracted using the infrastructure within the pillar with a controlled mining sequence and modification of the shaft steel work and lining. This can be increased to about 80% if alternate infrastructure can be used. Stringent planning is required as the highly stressed pillar has to be treated as if mining out a seismic-prone remnant. Detailed rock engineering designs are required and will have to cover the entire stoping sequence, trade-offs between the use of backfill or crush pillars and cater for the displacements or shaft dislocations.

The Tshukudu fault has been identified as a major fault zone (8 m wide) with very poor-quality rock and must be avoided. Development through the fault should be planned carefully to avoid delays and costs. There should be at least two main haulages developed though the fault, to ensure two means of egress. Reduced development rates should be anticipated in this area and included in the mine design. It is recommended that access drives and the T-intersections be sited outside the fault influence zone to improve stability during their operational life. Findings from the seismic potential analysis and the remodelled Tshukudu fault influence zone, recommend bracket pillars of 10 m and 15 m on either side of the Nkwe and Tshukudu fault wireframes, respectively.

A potential for seismicity exists on the geological structures in some stopes and is likely to affect nearby access developments. Additional S3 support will be required to cater for dynamic loading.

The proposed support standards and specifications for Platreef are listed in Table 16.9 and Table 16.10.

Table 16.9 Support Standards for Excavations

Excavation Type	DF (from numerical modelling)	Q rating (from rock mass classification)	Support Standard
Shafts (blind sink)	-	-	Primary Support: Minimum 1.8 m long SS 39 split sets in a maximum 1.0 m x 1.5 m pattern with mesh. Secondary support: 300 mm concrete lining.
Vent shafts (raisebore)	-		Minimum 50 mm shotcrete or concrete lining as required, determined from raisebore study.
Tunnel support for normal conditions (S1A)	DF < 0.5 m	Q>10	Primary Support: 2.4 m long, tensioned resin rebars in a 1.8 m x 1.8 m pattern in crown and down to 1.5 m from floor (four rebars in back and two rebars in each sidewall per row). Mesh to be installed in the crown only. Support installed to face prior to face drilling.

Excavation Type	DF (from numerical modelling)	Q rating (from rock mass classification)	Support Standard
Tunnel support for normal conditions (S1B)	DF < 0.5 m	4<Q<10	Primary Support: 2.4 m long, tensioned resin rebars in a 1.8 m x 1.8 m pattern with mesh in crown and down to 1.5m from floor (four rebars in back and two rebars in each sidewall per row). Support installed to face prior to face drilling. Support pattern may be denser depending on ground conditions.
Tunnel support for high stress, dynamic conditions (S2A)	0.5<DF<1.5m	1<Q<4	Primary Support: 3.0 m long, tensioned resin grouted yielding bars in a 1.2 m x 1.2 m pattern with 50 mm shotcrete in crown and down to 0.5 m from floor (five yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling. All mining access sublevels will require S2 support after ten years of mining. Shotcrete required for main level haulages and ramps (long term).
Tunnel support for high stress, dynamic conditions (S2B)			Primary Support: 3.0 m long, tensioned resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh in crown and down to 0.5 m from floor (five yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling. All mining access sublevels will require S2 support after ten years of mining. Not in main level haulages and ramps (long term).
Tunnel support for extreme high stress, dynamic conditions and rehabilitation (S3)	DF >1.5 m	Q<1	Minimum 3.0 m long grouted, resin yielding bars in a 1.0 m x 1.0 m pattern with mesh and Osro straps across the drive, in crown and down to 0.5 m from floor (six yielding bars in back and four yielding bars in each sidewall per row).
Production drifts	-	Q>10	Minimum 2.0 m long SS 39 split sets in a 1.8 m x 1.8 m pattern with mesh (in the crown only), down to 1.5 m from the footwall + Backfill support.
Production drifts		4<Q<10	Minimum 2.0 m long SS 39 split sets in a 1.8 m x 1.8 m pattern with mesh, down to 1.5 m from the footwall + Backfill support.
Production drifts		1<Q<4	Minimum 2.0 m long SS 39 split sets in a 1.2 m x 1.2 m pattern with mesh, down to 1.5 m from the footwall + Backfill support.
Production drifts		Q<1	Minimum 2.0 m long SS 39 split sets in a 1.0 m x 1.0 m pattern with mesh, down to 1.5 m from the footwall + Backfill support.

Excavation Type	DF (from numerical modelling)	Q rating (from rock mass classification)	Support Standard
Support for hazardous geological structures (shear zone area)			Primary support + 6.0 m long pre-tensioned, grouted, cable anchors installed in a 1.5 m x 1.5 m pattern in the affected area at time of development.
Support for LAF and chrome stringers (stope drives & development)			Primary support + pre-tensioned, grouted, cable anchors installed in a 1.5 m x 1.5 m pattern in the affected area at time of development (minimum 4 cable anchors in a row). Avoid tunnel intersections on these structures. Cable length = 4.5 m / 6.0 m in stope drive and development, respectively.
Support for LAF and chrome stringers (drift and fill)			Primary support + minimum 4.5 m long pre-tensioned, grouted, cable anchors installed in a 1.5 m x 1.5 m pattern in the affected area at time of development (minimum 4 cable anchors in a row). Avoid tunnel intersections on these structures.
Support for 3 - way intersections	-	-	Primary support + 6.0 m long pre-tensioned, grouted, cable anchors installed in a 2.0 m x 2.0 m pattern in the roof of the intersecting excavations. Support must be installed in conjunction with development of the breakaways. (Approximately 8-10 anchors in tunnel and breakaway position).
Support for 4 - way intersections	-		Primary support + 6.0 m long pre-tensioned, grouted, cable anchors installed in a 2.0 m x 2.0 m pattern in the roof of the intersecting excavations. Support must be installed in conjunction with development of the breakaways. (Approximately 12-15 anchors in tunnel and breakaway position).
Bull nose			Primary support + Osro straps across the bull nose for confinement, from the shoulder down to 1.0 m from the footwall.
Development through Tshukudu fault	-		Spiling (12° to 15° to the horizontal) rings of 6.5 m long self-drilling anchors (SDAs) every 1.5 m (spacing 0.5 m to 1.0 m). Resin injection through the SDAs for consolidation. Steel arches spaced 0.5 m to 1.8 m apart depending on rock quality (expected 1.0), void filling between arches and tunnel walls. Resin consolidation and reinforcement may also be required in the face.
Large excavation support			Primary support + pre-tensioned, grouted, cable bolts (minimum length = half excavation span), maximum spacing = 0.5 x length. Shotcrete required in crown and sidewalls down to 1.5 m from footwall.

Excavation Type	DF (from numerical modelling)	Q rating (from rock mass classification)	Support Standard
Large excavation brow support			Primary support + three rows of pre-tensioned, grouted, cable bolts, in a 1.5 m x1.5 m pattern or denser, depending on mass to be supported. Length = brow height + 0.5 m. Installed within 1.0 m of the brow. Shotcrete required.
Critical ore passes			3.0 m long pre-tensioned, grouted, cable anchors installed in a 2.0 m x 2.0 m pattern.
Stope drive support		-	Primary support: 3 m long resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh or 50 mm shotcrete in crown and down to 0.5 m from floor (seven yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling.
Stope brow support (where necessary)			Primary support + three rows (1.0 m apart) of three 6.5 m long grouted, cable anchors installed within 1.0 m of planned brow position.

Table 16.10 Support Specifications

Support Type	Minimum Specification
Rebar	Minimum yield strength 500 MPa black steel, minimum 20 mm diameter, hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing must be demonstrated through approved testing).
Yielding bar	Minimum yield strength 500 MPa black steel, minimum 20 mm diameter, minimum energy absorption 30 kJ within 300 mm, hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing must be demonstrated through approved testing).
Split set (SS-33)	Minimum outer diameter 33.5–34.2 mm, minimum 420 MPa (yield stress) black Supraform steel, minimum steel thickness 2.3 mm, hole size 30–32 mm.
Self-drilling anchor	Minimum 32 mm diameter black steel, 360 kN ultimate load, hollow self-drilling anchors (R32s).
Cable anchor	Minimum 18 mm diameter black steel, 380 kN ultimate load.
Mesh	Black weld mesh, minimum 5 mm gauge, maximum 100 mm aperture, blast resistant.
Shotcrete	Minimum 25 MPa (28-day strength) fibre reinforced shotcrete.
Osro straps	300 mm wide straps with five 10 mm rods, minimum 500 MPa black steel.
Capsule resin	Two component urethane silicate resin capsules. Fast (< 30s) and slow (5–10 min) setting.
Injection resin	Two component urethane silicate injection resin with water sealing properties.
Cable grout	Minimum 40 MPa Ordinary Portland Cement, water cement ratio 0.35:0.40.
Steel arches	TH29 (29 kg/m), A3Y profile made to required tunnel dimensions.
Void filling	Low density aerated cement (0.35 MPa), pumped into woven polypropylene bag.

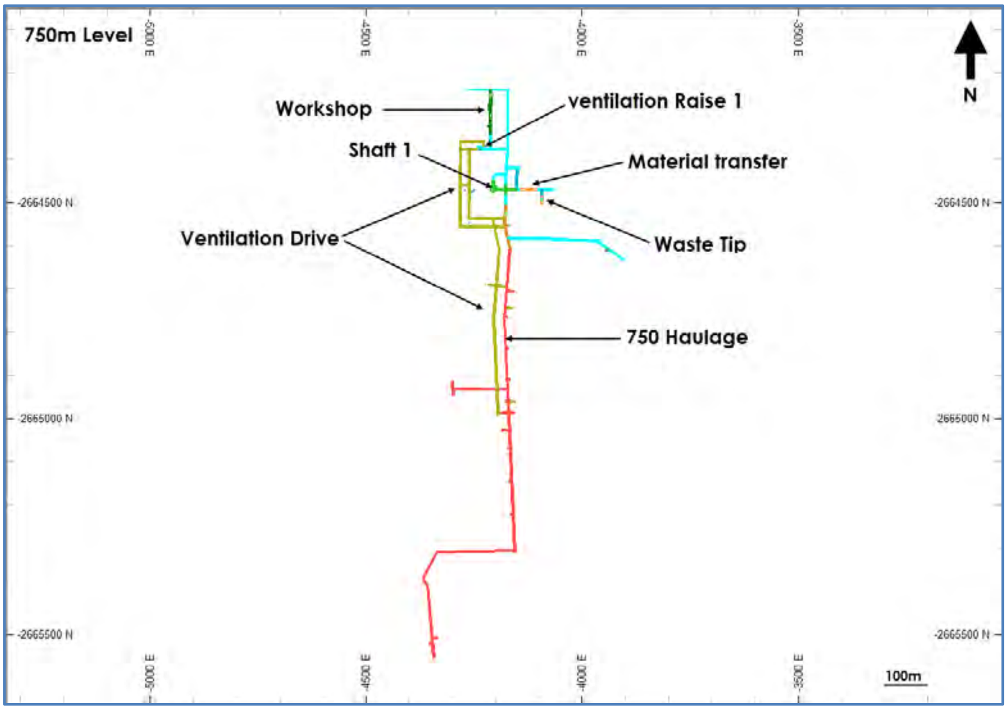
16.1.5 Review of the Mine Design

For the Platreef 2022 FS, changes made to the mine designs to meet the requirements for the Phased Development Plan including additional ventilation and exploration development and waste and ore handling system. Changes also made to avoid developing through the Faults. However, the changes did not exceed more than 10% and the majority of the mine designs are the same as the Platreef 2017 FS. The development and the production schedules have been adjusted to reach the high-grade profiles in the early years and then ramp up to the steady state ore production of 5.2 Mtpa.

The Platreef 2022 FS is a phased development plan trying to make revenue from the smaller plant in the early years to be used as funding for the expansion phase. For this reason, high-grade stopes with the appropriate underground infrastructure around Shaft 1 but additional exploration and ventilation development compared to the 2017 FS mine design was taken into consideration.

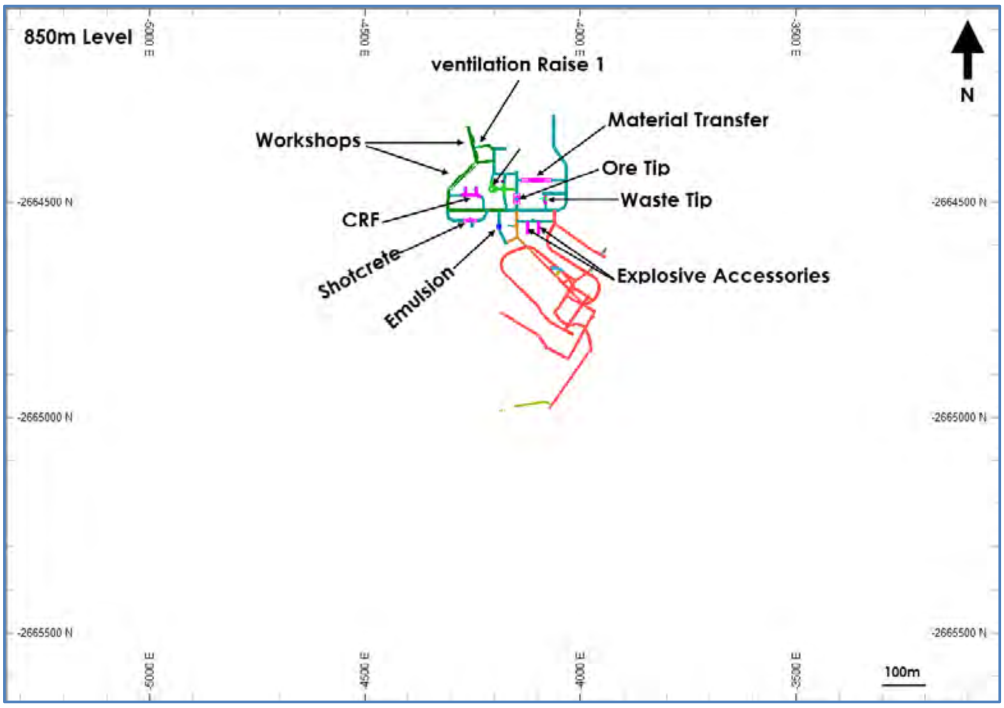
Figure 16.7, Figure 16.8 and Figure 16.9 show the plan view of the initial development on three main levels of 750 m, 850 m, and 950 m. A North-South view of the initial development is shown in Figure 16.10.

Figure 16.7 Platreef 2022 FS – Initial (Pre-Production) Development on 750 m Level



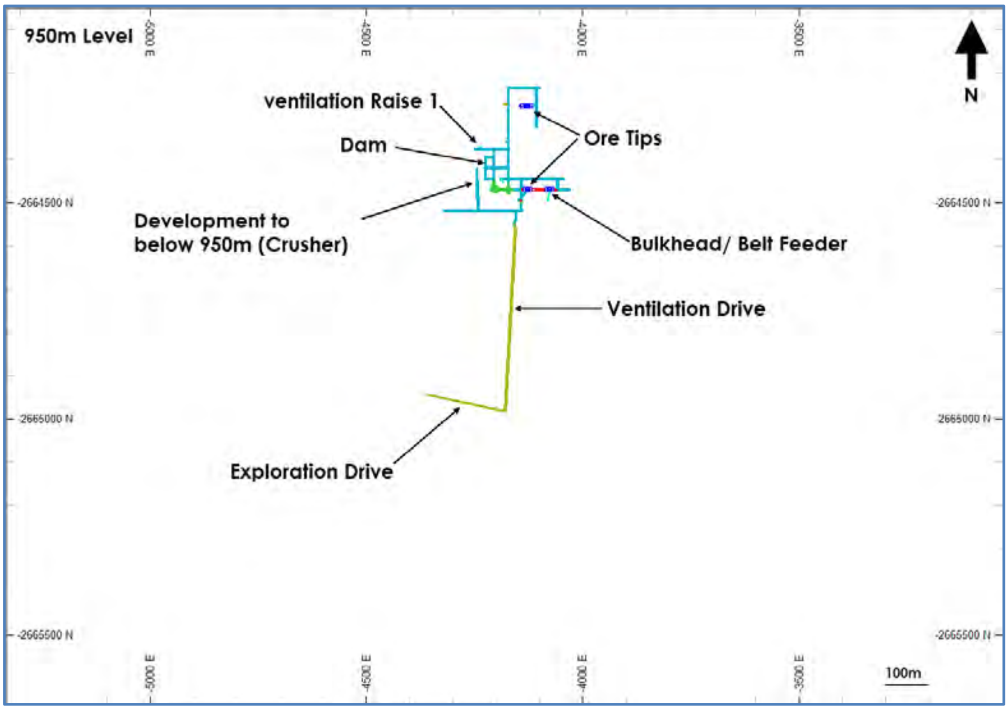
OreWin, 2021

Figure 16.8 Platreef 2022 FS – Initial (Pre-Production) Development on 850 m Level



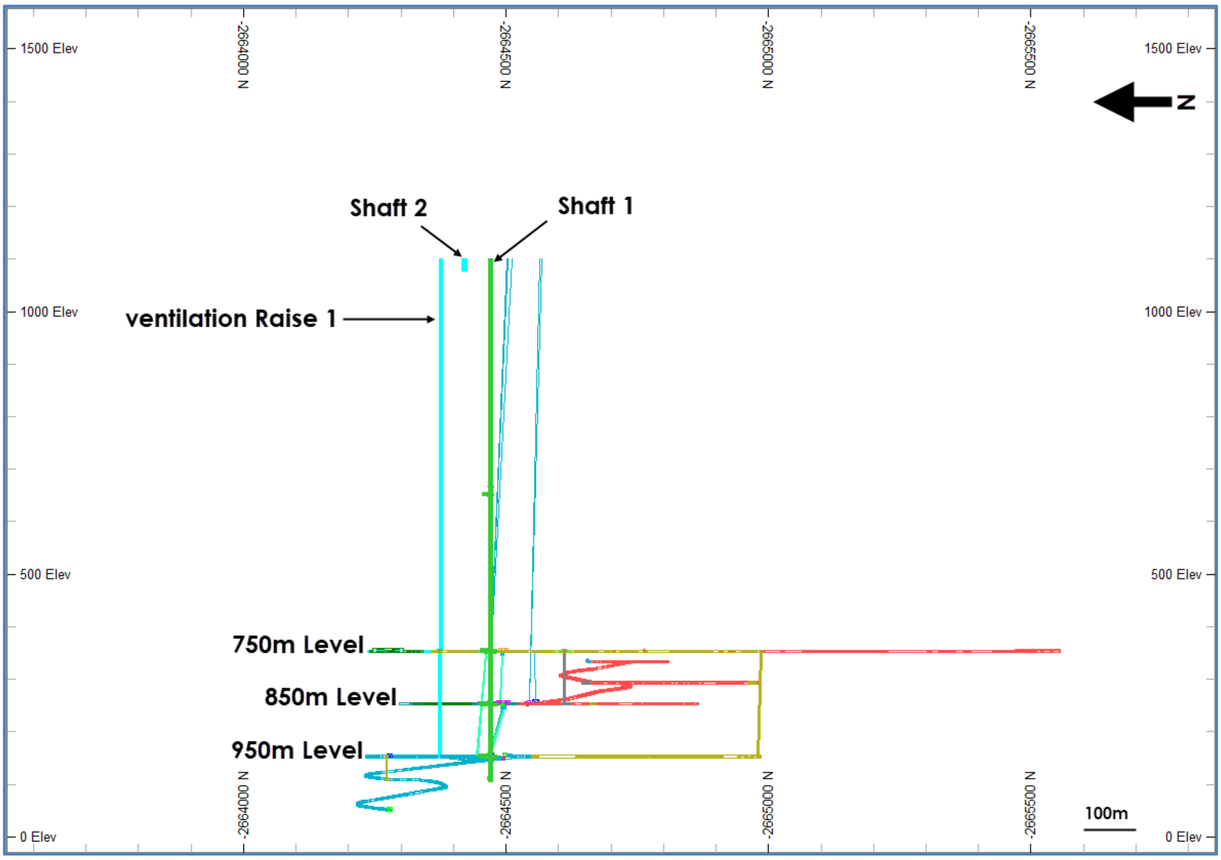
OreWin, 2021

Figure 16.9 Platreef 2022 FS – Initial (Pre-Production) Development on 950 m Level



OreWin, 2021

Figure 16.10 Platreef 2022 FS – Initial (Pre-Production) Development



OreWin, 2021

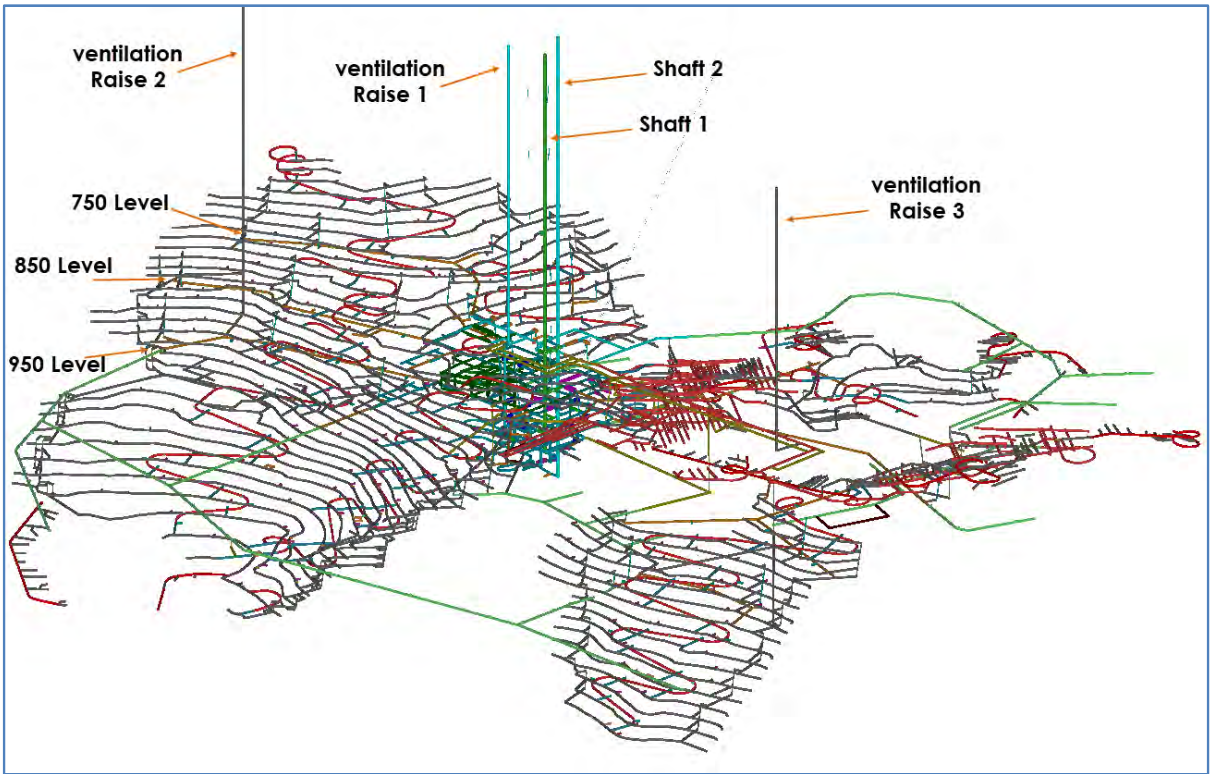
The Platreef Project is designed based on highly mechanised Longhole Stoping and Drift-and-Fill mining methods.

The Platreef 2022 FS evaluates a phased development of Platreef, with an initial 700 ktpa underground mine and a 770 ktpa capacity concentrator, targeting high-grade mining areas close to Shaft 1, with a significantly lower initial capital cost of $520M (including $50M in Shaft 2 and $32M in contingencies). First concentrate production for this option is targeted in 2024, with the sinking of Shaft 2 recommencing in Q3'23, to coincide with the construction of two 2.2 Mtpa concentrators to be completed by 2028 and 2030. This would increase the steady production to 5.2 Mtpa by using Shaft 2 as the primary production shaft.

Primary access to the mine will be by a 1,100 m deep, 10 m diameter production shaft (Shaft 2). Secondary access to the mine will be via a 996 m deep, 7.25 m diameter ventilation shaft (Shaft 1). During mine production, both shafts will also serve as ventilation intakes. Three additional ventilation exhaust raises (Ventilation Raise 1, 2, and 3) are planned. Ventilation Raise 1 will be a 950 m deep, 6 m diameter raise located near the centre of the mining area and adjacent to the two intake shafts. Ventilation Raise 2 will be an 800 m deep, 6 m diameter raise located near the northern edge of the mining area. Ventilation Raise 3 will be a 725 m deep, 6 m diameter raise located near the southern edge of the mining area.

Three main access levels will be established as primary haulage levels. These are the 750 m, 850 m, and 950 m Haulage Levels. Figure 16.11 shows the proposed shaft and raise locations and the main access levels in an elevated view (looking north-east). Mining access ramps will connect the haulage levels with the mining sublevels and other infrastructure. The mining sublevels will be developed from the ramps at regular vertical intervals in the production areas. Drilling and extraction levels for stopes will be driven from the sublevels. Ventilation raises and ore passes will also connect the sublevels with the main haulage levels.

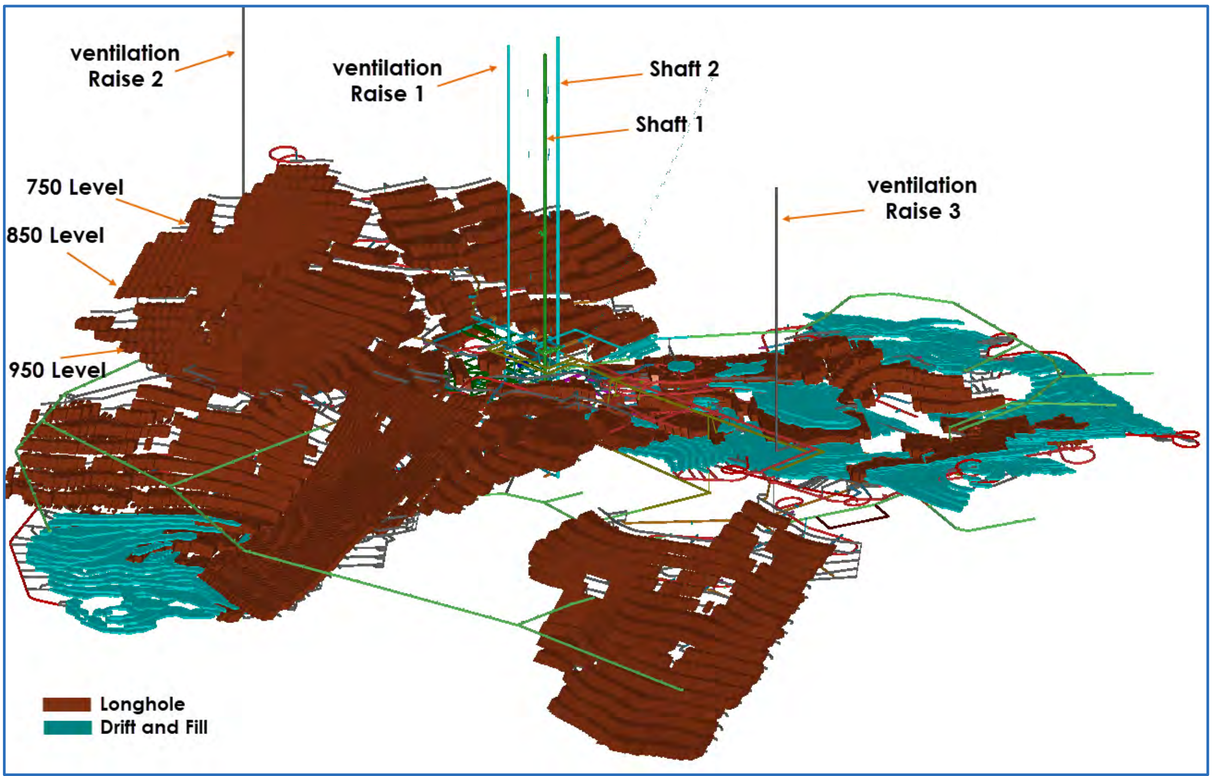
Figure 16.11 Mine Access Layout



OreWin, 2021

The main mining methods will be Longhole Stoping and Drift-and-Fill mining. These methods provide a safe, mechanised, and productive mining plan. The Longhole Stopes were designed at a 20 m height. This modification allowed for an improvement in the overall grade of the mine plan. All Longhole Stoping will be a transverse mining method using 6 m wide top cuts. This change allows for development to be taken off the critical path, as the secondary top cuts can be driven prior to the primary top cuts being mined and filled. Figure 16.12 is an isometric view of the mining areas by method (looking north-east).

Figure 16.12 Elevated View of Mining Areas by Method



OreWin, 2021

A three-dimensional elastic numerical modelling exercise was conducted to assess the updated FS mining layout, the induced stress on the main access levels, sublevels, access ramps and shafts. It was observed that the design generally complies with the geotechnical design parameters. Overall, the following was recommended:

- During extraction, isolated pillars are formed between the stopes and these subject access sublevels to high stresses. S3 support will be required to maintain access during extraction of the secondary stopes.

There are a number of areas where it is planned to undercut previously backfilled longhole stopes and drift and fill sections. Where this occurs, the following should be considered:

- Undercutting old drift and fill stope with drifts will require 1.0 MPa sill strengths. The extraction sequence should be such that the lower drifts are offset to minimise the effect of cold joints and reinforced shotcrete arches must be installed during development to ensure the safety of personnel.
- Undercutting the old backfill drift and fill with long hole stopes will cause dilution of up to 25% for a single cut and 50% for a double cut. This should be included in the mine design.
- For the longhole stope it is recommended to fill the entire stope with addition binder and 2.5 MPa sill strength will be required.

- The empirical method of determining the backfill sill strengths assumes continuous backfill placements and that there are no cold joints or air pockets. These defects affect the stability of the backfill sill.

16.1.6 Geotechnical Risks

The Tshukudu fault has been identified as a major fault which traverses the Project area. This fault presents a major geotechnical hazard as it is often characterised by very poor quality rock. Development through the fault should thus be planned carefully to avoid delays and costs. As the Tshukudu fault strikes across the entire lease area, some development through the Tshukudu fault will be essential to provide access to ore to the west of the fault. There should be at least two main haulages developed though the fault, to ensure two means of egress. Specialised support comprising resin injection, arch sets and void filling (Table 16.9 and Table 16.10) to be carried out by a specialist contractor is recommended for this case. An indication of the slow rate of this development is also provided.

The ingress of water through the Tshukudu fault is another potential risk as the presence of water is likely to negatively influence stability. Modelling by Golder Associates indicates that 80% of the groundwater inflow into the underground workings will be derived from the faults traversing the mine. The Tshukudu fault will produce a greater inflow since it covers more mining area than the other faults (see DRA Report Number: DRA-J0283-STU-REP-909 S04).

Following the identification of the Tshukudu fault it has been established that the type of alteration within the fault and the true thickness of the fault is variable, indicating that improved characteristics of the fault zone in some areas may exist. It is, therefore, possible to develop through the fault in these cases with fewer delays and less intensive support, provided that there is no water ingress.

Overall, the Tshukudu fault has been taken into consideration in the mine design, however mining through the fault will need to be monitored and managed carefully. Geotechnical drilling will be required to delineate and characterise the Tshukudu fault during implementation.

Seismic potential may be associated with underground mining operations at depth around the Nkwe fault zone. This may induce seismic activity leading to falls of ground, damage to equipment and injuries or fatalities. Adherence to the appropriate mining sequence, proper placement of backfill, a seismic monitoring network, and adequate support (S3) are the controls in place.

The mining depth, access ways, LHOS and drift profiles outlined in the Platreef 2022 FS mine design could induce high stress zones in the vicinity of these excavations, resulting in potential damage to excavations and ground support. An elastic modelling assessment of expected stress effects on excavations has been conducted and the use of appropriate support (S2 or S3) was selected for high stress environments. Ground monitoring such as underground inspections, or primary stress measurements at deeper production levels can be implemented at an operational level.

The updated mining layout and schedule includes a few cases where undermining of previously backfilled drift and fill and longhole stopes. Backfill dilution can be mitigated by increasing the cement content in the backfill of the stope to be undermined. However, in the case of undercutting drift and fill stopes, up to 25% for a single cut and 50% for a double cut must be anticipated.

The Platreef project area is traversed by faults, low angled features (LAFs) and weaker chromite partings which have the potential to create adverse ground conditions such as key block creation and falls of ground. Support strategies have been designed to cater for these features.

Tight filling in the drift and fill is essential to ensure successful mining. Instances where poor tight filling has occurred may lead to significant stability problems, particularly in the backs of tertiary drifts. Long anchors will be required to support the backs and if the problem is more widespread, it is likely that some tertiary drifts will be abandoned.

16.2 Underground Mining

16.2.1 Introduction

The Platreef 2022 FS evaluates a phased development of Platreef, with an initial 700 ktpa underground mine and a 770 ktpa capacity concentrator, targeting high-grade mining areas close to Shaft 1. With the sinking of Shaft 2 recommencing in 2023, first concentrate production is targeted in 2024 to coincide with the construction of two 2.2 Mtpa concentrators to be completed by end 2028 and 2030. This would increase the steady production to 5.2 Mtpa by using Shaft 2 as the primary production shaft. The Platreef 2022 FS describes a change in production rate for the project that will require separate capital costs and infrastructure.

Key steps involved in preparing the Platreef 2022 FS are as follows:

- Shaft 1 changeover completed for permanent hoisting in February 2022.
- Start development from the bottom of Shaft 1 in April 2022.
- Shaft 1 capacity is limited to ~700 ktpa ore, plus waste development.
- Initial development focus from Shaft 1 is a ventilation raise, completed by February 2024.
- Reduced initial development, focusing on the nearest, highest-grade stopes.
- Shaft 2 sinking recommences in September 2023. This is a discrete decision, and can be started at any point in time, depending on funding.
- Base case is a 770 ktpa concentrator on site.
- Assumes dry stacked tailings dam (for on-site concentrator).

The Platreef 2022 FS mining method selection focused on mechanised mining with high productivities. Mine design work aimed to maximise production grades and reduce operating costs. Mine schedule focused on optimising stope sequencing, maximising grades in the early years, and removing development from the critical path. The sub-sections below discuss considerations taken in the mining methods selection process and in the final mine designs and schedules.

16.2.2 Mine Design Parameters

The following mine design criteria were used for the Platreef 2022 FS.

Ore and Waste Properties

- Ore in Situ Density:
  - T1 = 3.18 t/m³
  - T2U = 3.19 t/m³
  - T2L = 3.04 t/m³
  - Average =3.14 t/m³
- Waste in Situ Density (Pyroxinites):
  - Hanging wall = 2.91 t/m³
  - Footwall = 3.10 t/m³
- Swell for Development = 40%
- Swell for Production (Longhole Stoping) = 40%
- Swell for Production (Drift-and-Fill) = 40%

Mine Planning

- Lateral Development:
  - Maximum grade will be 15%.
  - Ramps will level off at sublevels to reduce risk of rollovers.
  - Ore passes will be spaced 400 m apart on the levels.
  - Electrical bays will be spaced 400 m apart.
  - Level development will be inclined slightly to have water drain to the sumps (actual incline to be designed during detail design).
- Vertical Development:
  - All raises except Shaft 1 and Shaft 2 will be raisebored.
  - Maximum raisebore diameter is 6 m.
- Longhole Stoping:

  - Bottom-up Longhole Stoping will be used.
  - Minimum inclination of stopes on the footwall side will be 55°.
  - The unfilled stope length for the 20 m high Longhole Stopes should not exceed 60 m.
  - As much as possible, stopes should be aligned to minimise the risk of failure.
  - A primary-secondary stoping sequence is recommended.
  - Adjacent secondary stopes are not mined simultaneously.
  - Secondary stope development can be carried out at any stage, regardless of whether nearby stopes have been backfilled.
  - Over break thickness for top sides, hanging wall, footwall, end sides, and secondary stope sides is 0.5 m.
  - Over break thickness for the floor is 0.3 m.
- Drift-and-Fill:
  - Overhand Drift-and-Fill will be used.
  - Each horizontal slice is mined completely before mining of the slice above begins.
  - Drifts will be driven with flat backs for recovery purposes.
  - Mining widths should be limited to 5.5 m to reduce the length of ground support in the back and walls to 2.4 m. If mining widths exceed 5.5 m, additional support will be required.
  - Assuming a mining width of no more than 5.5 m, the mining sequence should be primary-secondary-tertiary to ensure that slender backfill ribs are not formed. For this method a 5.0 m width was utilised.
  - An average of 0.15 m of over break will be used to estimate the overall dilution from each paste fill rib.

The stoping designs were based on the Net Smelter Return (NSR), the inputs of which are defined in Section 14.6.

Geotechnical information was used for development designs, ground support, and stope layouts. The ventilation parameters defined minimum air velocities, air quantities, and cooling requirements. Shaft parameters were used to determine the production rate along with the production ramp up profile. The underground infrastructure parameters defined required mine facilities, excavation sizes, and equipment requirements (mobile and fixed).

16.2.3 Mining Method Selection

Sublevel Blasthole (Longhole) Stoping with cemented paste fill / rock fill, supplemented with development waste rock, where possible, to fill open stopes in the thicker ore zones. This method will minimise mining costs and achieve the highest productivity. The remainder of production may come from thinner high-grade zones. Drift-and-Fill methods will be used in these zones.

Two longhole mining scenarios were studied and evaluated for the Platreef 2022 FS:

- Transverse Longhole Stoping
- Longitudinal Longhole Stoping.

Longitudinal Longhole Stoping was considered specifically where deposit thickness could not support stope lengths of 15 m or more, normal to the strike, where Transverse Longhole Stoping would not be a suitable option. From the Vertical Miner software results, three major areas (Zone 1 north section, Zone 4, and Zone 2 central locations), were identified as the ideal candidate for this mining method. Stope shapes were created accordingly. After reviewing the longitudinal stope shapes created by MSO, the entire methodology had to be dismissed due to geotechnical considerations, such as hydraulic radius and mining direction. The ribs of the stope were parallel to the fault planes and the hydraulic radius provided would not support such stope shapes with meaningful sizes.

For the thinner and high-grade areas, two mining methods were evaluated:

- Cut-and-Fill
- Drift-and-Fill.

After reviewing the mining shapes (pancakes), it was concluded that Drift-and-Fill mining would be possible and is the most suitable solution due to higher productivities and performance rate.

16.2.4 Mining Shapes Design

Key criteria considered to determine the best method to extract the ore are as follows:

- Safety,
- Targeting high grade mining areas close to Shaft 1 (most accessible highest grade, earliest tonnes, maximum margin),
- Leverage existing supporting surface infrastructure,
- Maximise productivities by incorporating bulk mining methods and operational flexibility, resulting in low operating costs,
- Maintain high overall recovery rates,
- Minimise overall dilution,
- Prevent surface subsidence,
- Equipment selection and compatibility, and
- Minimum underground infrastructure based on 2017 FS design.

The following mining methods were used in the Platreef 2022 FS. All three methods use cemented paste fill or cemented rock fill supplemented with a minor amount of development waste rock to fill open stopes.

- Longhole Stoping – For ore zones with vertical thicknesses exceeding 15 m, longhole Stoping methods include the following:

  - Transverse Longhole Stoping, and
  - Longitudinal Longhole Stoping.
- Drift-and-Fill Mining (Mechanised Cut-and-Fill) – For thinner portions of the ore zones (less than 10 m thick).
- For zones of intermediate thickness (10–15 m thick), bottom-up mining using 5 m lifts of Drift-and-Fill.

16.2.4.1 Longhole Stoping

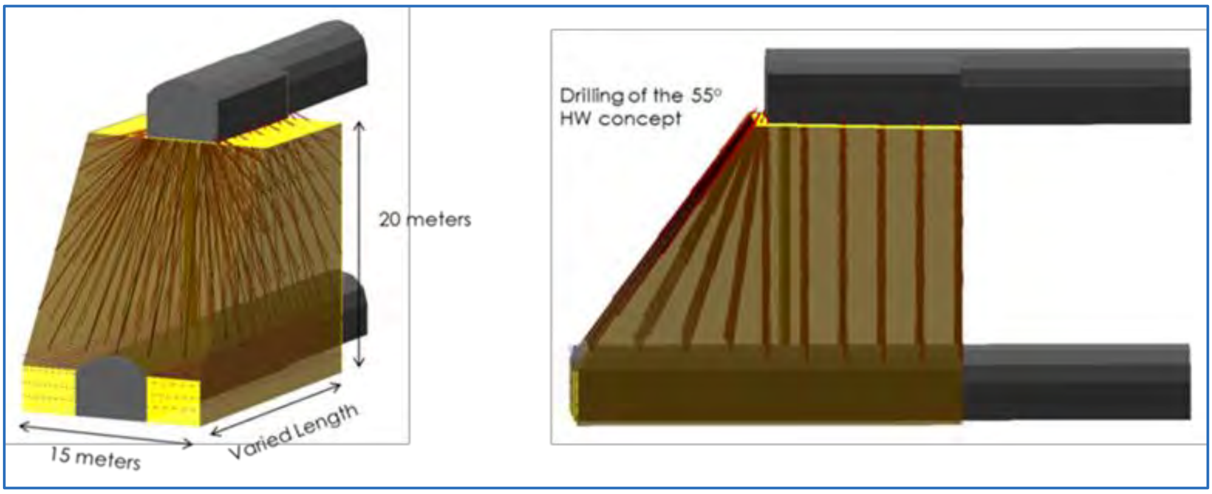
The primary mining method selected for the Platreef 2022 FS is Longhole Stoping. For post-mining support, paste fill will be used in all primary stopes, while paste fill supplemented with development waste rock will be used in the secondary stopes. Longhole Stoping is a highly productive bulk mining method that provides good ore recovery with minimal dilution. Both Transverse and Longitudinal Longhole Stoping methods were evaluated and studied for the project. In the Platreef 2022 FS, Longhole Stoping will be introduced in Phase 1 mining plan to reduce mining costs and ensure constant grades over a higher mining cut.

Transverse Longhole Stoping

Transverse Longhole Stoping extracts ore in blocks oriented perpendicular to the strike of the mineralised zones. Mining access levels are driven subparallel to the strike of the ore zones in the footwall of the deposit. Drilling and mucking access drifts are then driven from the levels to the top and bottom of the stope, respectively. The levels and access drifts up to the ore contact at the stope boundary are driven at 5 m W x 5 m H arched.

The designs assume a stope design width of 15 m for all transverse Longhole Stopes. They also assume that a minimum vertical height of 15 m is required for transverse Longhole Stopes to ensure the stability of the ground between the drill drift and the mucking drift. Stope lengths vary depending on the thickness of the ore zone from a minimum of 5 m to a maximum of 60 m, as governed by the geotechnical hydraulic radius (provided by SRK). The drilling level at the top of the stope is developed at 6 m W x 5 m H arched. The mucking drift along the stope bottom is also developed at 6 m W x 5 m H arched. Finally, the stopes will be designed with a minimum hanging wall and footwall angle of 55°. Figure 16.13 illustrates the concept for a typical transverse stope, demonstrating the fan and 55° wall drilling pattern design.

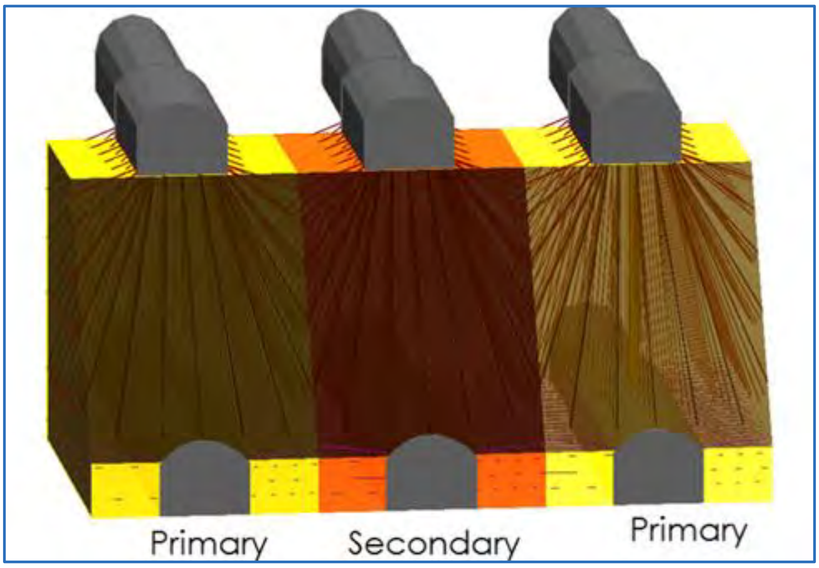
Figure 16.13 Typical Transverse Drill Design



Transverse Longhole Stoping is considered a bottom-up method, whereby the lowest stopes are removed first. In primary stopes, the drill drift and mucking drift will be driven perpendicular to the strike of the ore body. The initial bottom drift will be slashed to the full 15 m stope design width immediately in advance of each stope bench blast. Walls will be slashed using a Jumbo drill. The holes will be angled into the wall such that they slash out to the centre drift and will be drilled at least 22° from perpendicular to the drill drift. Once the slash is taken, no personnel shall be permitted in the area since it does not include entry-quality ground support.

Stopes above the bottom stopes only require a drill drift to be developed; the mucking drift will already be established by the stope below and will subsequently be mined to the required 15 m width with the stope blast. Secondary stopes will be mined similarly to primary stopes; however, they will require that the primary stopes on both sides be mined out and paste filled prior to blasting. Figure 16.14 illustrates the concept of mining the transverse stope with primary and secondary stopes.

Figure 16.14 Mine Sequence Concept



After completion of the development, a slot void must be created prior to blasting the entire stope. These slot rings, when blasted, will provide a large enough void to blast the remaining stope. The slot will consist of three production rings located at the end of the planned stope. These rings will be vertical. Any additional fan production rings will be blasted with the cap blast of the slot. The remainder of the stope will be drilled and blasted on retreat from the brow of the initial slot rings to the stope entrance.

To begin the slot sequence, a drop raise will be centred on the second ring in the centre hole. This raise will be 2.4 m x 2.4 m and will consist of 17 holes, including five centre holes. The five centre holes will be reamed to a larger diameter and will serve as cut holes. These holes will be uncharged. Note an alternative to this type of a drop raise is a large diameter raisebore slot. The drop raise will be blasted in two lifts with the first being half of the length of the drilled raise. Prior to blasting the final drop raise shot, the remaining rings on the three slot rings will be taken to the same height of the blasted portion of the drop raise. The remaining cap of the slot rings will be removed with the drop raise and any fan holes that are drilled out at the end of the stope. This process will end the slot production sequence and will begin the sequence to extract the remaining stope.

After mining of the stope is completed, it will be backfilled with paste except when waste rock is available for inclusion in the secondary stope backfill mix. In Phase 1, Cemented Rock Fill (CRF) will be used for backfill until paste fill is available in 2027. During the backfill portion of the stope cycle, an engineered bulkhead will be constructed at the stope entrance on the lower drift. Paste fill will then be poured from the top drift into the open stope. Once the paste fill is poured, it will require a cure time of 28 days prior to blasting against the backfill. Other activities such as development and drilling in the adjacent stopes may continue during the filling cycle.

Longitudinal Longhole Stoping

Similar to the Transverse Longhole Stoping method, Longitudinal Longhole Stoping consists of a drill development drift and a mucking drift. These drifts are driven parallel to the strike of the ore body. The width of the drift varies depending on the width of the ore body. The drifts are a minimum of 5 m and up to a maximum of 10 m in width. Stopes that are wider than 10 m are evaluated for Transverse Longhole Stoping or Drift-and-Fill mining methods.

This method requires the drifts be driven to a defined boundary; this boundary is normally either a change in mining method or the end of the mineralised deposit. Then, the stopes are mined out in sequence from the end of the drift to the access. Similar in process to Transverse Longhole Stoping, each stope has a drop raise as a point of beginning for the extraction retreat to the access crosscut. Depending on the length of the stope and in accordance with geotechnical design parameters, should the stope length be excessive, it will be necessary to stop, backfill, and re-establish a new drop raise for the recommencement of extraction. Upon completion, all voids will be paste filled and supplemented with development waste rock when applicable.

Longitudinal Longhole Stoping was rejected as viable for the following reasons:

- Geotechnical constraints dictated by the major faults, subparallel subsets, and hydraulic radius.
- Poor rock mass rating in these areas; stope length would be too short to justify.
- The risk of potential hanging wall failure.
- Low productivities due to the small stopes and the limited number that can be in production.
- High costs due to the above factors.

Longhole Stoping Design

Longhole stope orientation zones (SOZs) are based on the topographic surface created by the top of the T2 mineralisation. As shown by the red arrows in Figure 16.15, transverse stopes are mined up-dip and perpendicular to the contours of the ore body surface (illustrated by the blue lines). The map shows six SOZs (1a, 1b, 2, 3, 4, and 5) created by areas that have similar ore body dip directions.

Figure 16.15 Longhole Stope Orientation Zones

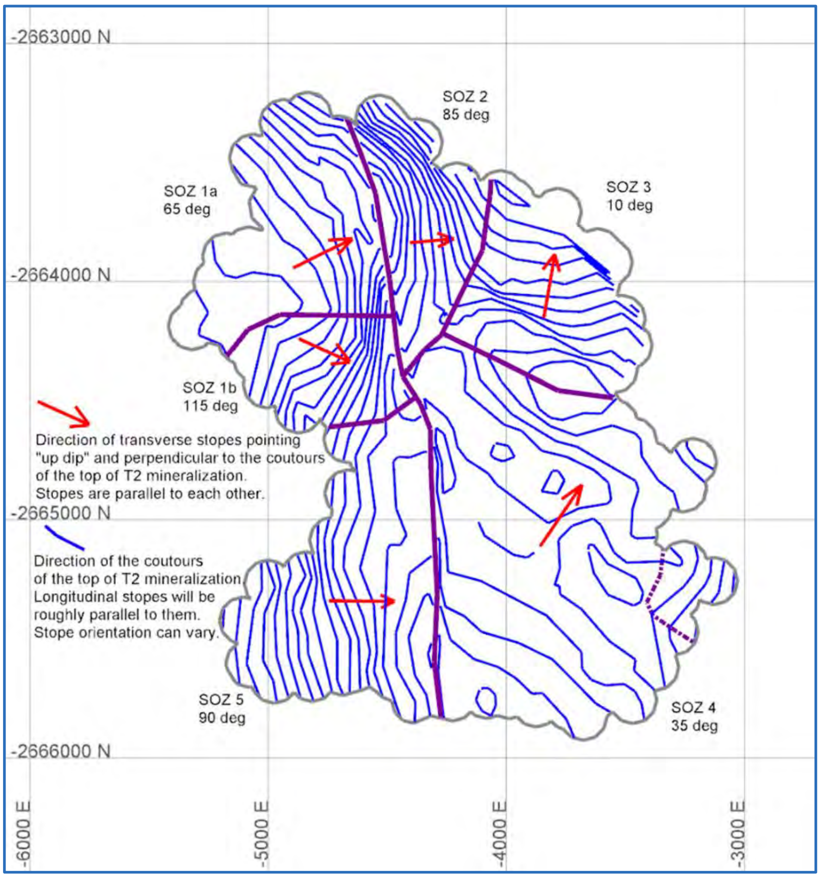
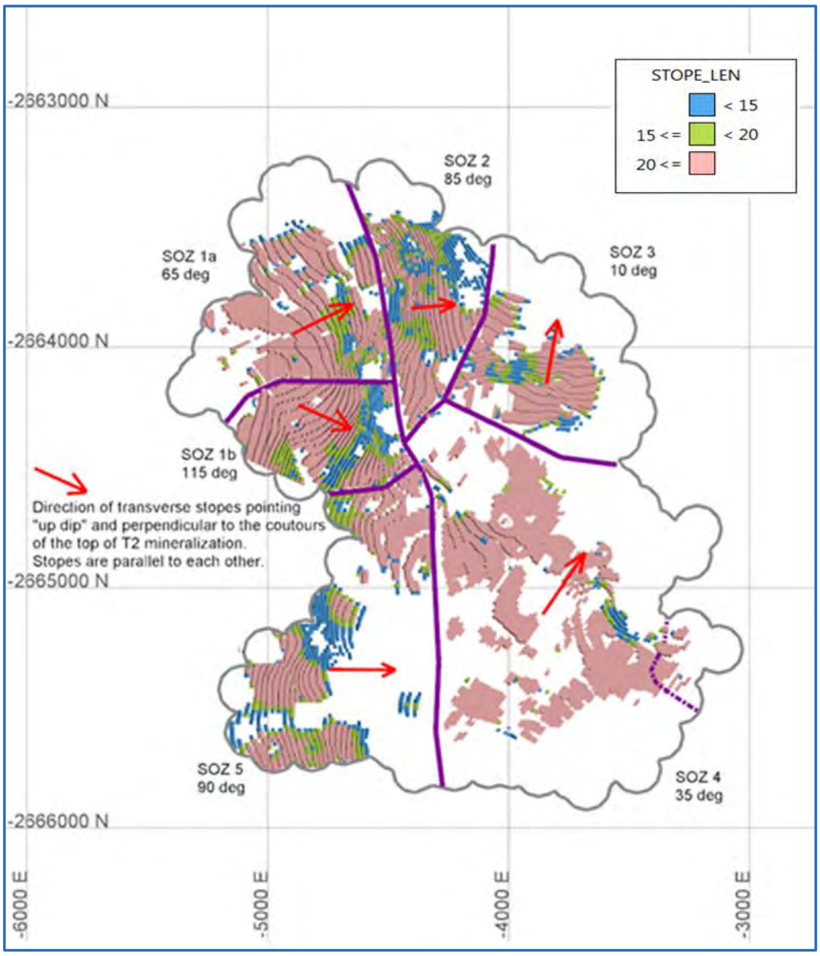


Figure 16.16 illustrates Maptek's Stope Optimiser (MSO) stopes that were generated using a $130/t NSR cut-off for a 20 m minimum ore thickness. Longhole Stopes are 20 m H x 15 m W x a minimum stope length of 5 m. The $130/t and $100/t NSR cut-off stopes are coloured by stope length in Figure 16.16 and Figure 16.17 according to the following:

- Blue: <15 m length
- Green: ≥15 m and <20 m length
- Pink: ≥20 m length

Figure 16.16 Longhole MSO Stopes – $130/T NSR Cut-off



MSO Longhole Stopes generated using a $100/t NSR cut-off are shown in Figure 16.17. (Except for the change in cut-off, stope parameters are the same as in Figure 16.16.).

Figure 16.17 MSO Stopes – $100/t NSR Cut-off

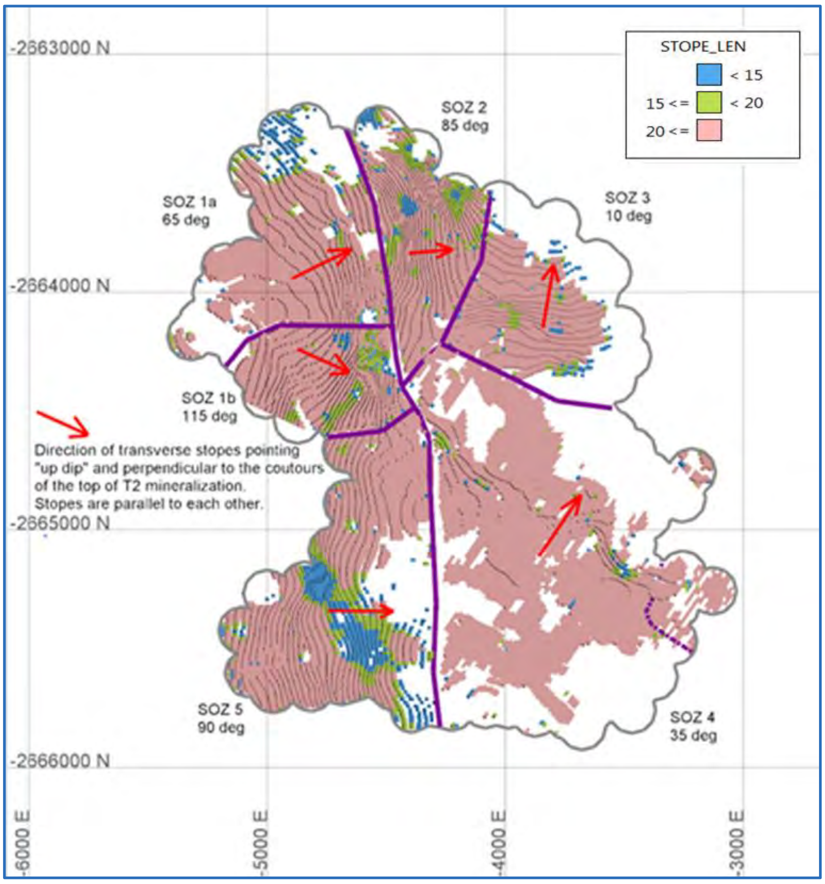
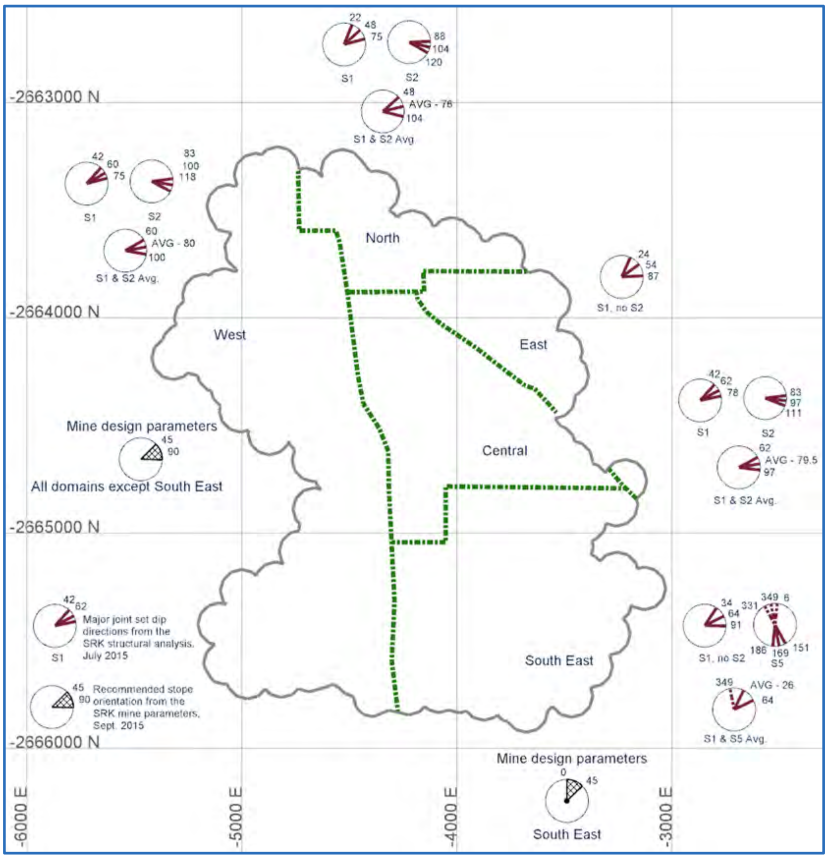


Figure 16.18 shows SRK geotechnical domains with green dashed borders. The SRK geotechnical report lists the major joint sets that would affect stope orientation as S1, S2, and S5, as well as domain averages in areas of multiple joint sets. Maximum, minimum, and mean dip directions are shown for each joint set in the circles. Since dip direction is normal to the strike of the joint set, dip direction is the preferred azimuth for stope orientation. In the cases where two joint sets are present, the average dip direction for both structures is shown in a circle below the two joint set circles, along with the new average dip azimuth for both joint sets. SRK's recommended stope orientations are also shown; an orientation of 45–90° is recommended for all domains except the south-east, which has a preferred orientation of 0–45°.

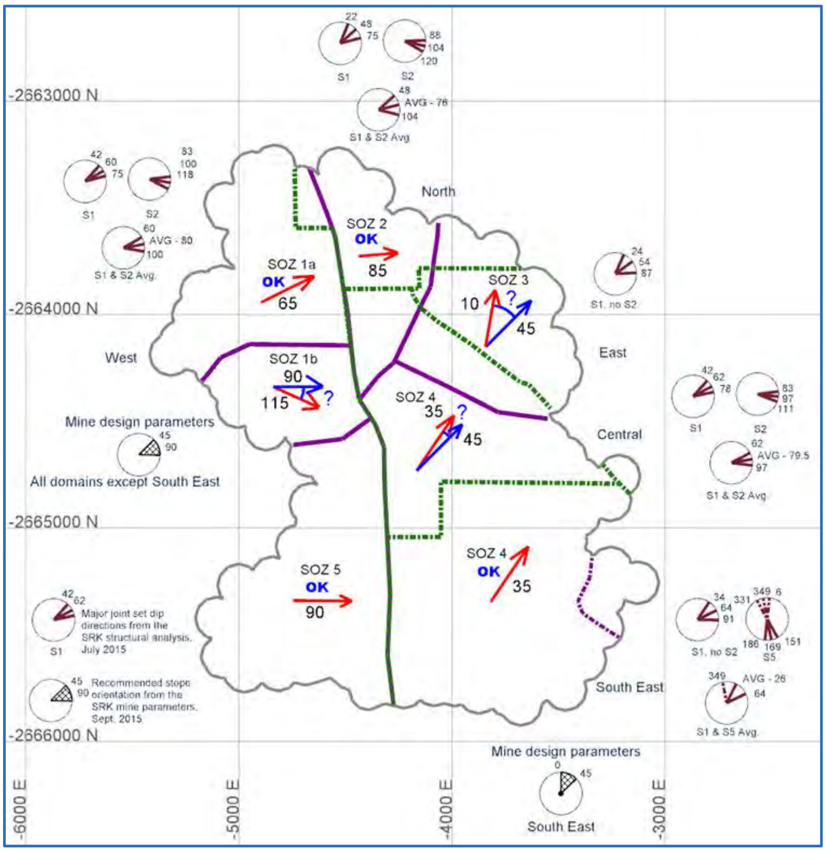
Figure 16.18 SRK Structural Domains and Preferred Stope Orientations



16.2.4.2 Transverse Longhole Stoping

In Figure 16.19, the SRK structural domain and preferred stope orientations are combined with the transverse longhole SOZs (Figure 16.15). For each SOZ, "OK" means that the transverse stope azimuth is within the SRK-recommended stope orientation limits. For the SOZ orientations that are not compliant, a blue arrow shows the corrected azimuth that is required to comply with the limits of the SRK geotechnical parameters. Most areas are "OK," except for SOZ 1b, SOZ 3, and the northern part of SOZ 4. In these cases, the stope orientations were rotated 10–35° to make them compliant with SRK recommendations.

Figure 16.19 Possible Transverse Stope Orientation Conflicts

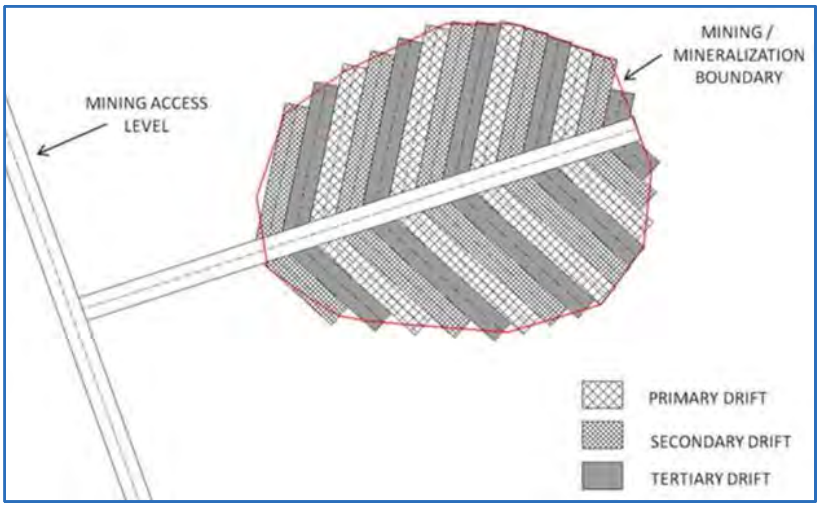


16.2.4.3 Drift-and-Fill Mining

In Drift-and-Fill mining, the ore zone will be divided into 5 m high horizontal slices (or lifts), and 5 m wide ore drifts will be mined and backfilled adjacent to one another in a repeating fashion. Upon completion of each drift, a bulkhead will be constructed, and the void backfilled with paste fill. After the paste fill sets sufficiently to the required strength (28 days), another drift will be driven next to the fill. Mining will progress in this manner in a chevron pattern until the entire slice of ore is depleted. Where ground conditions permit, mining can be performed using a primary secondary or primary-secondary-tertiary sequence, enabling access to multiple mining faces at all times and allowing greater productivity from an individual ore slice.

Drift-and-Fill mining is a flexible mining method that allows near-complete recovery of the ore zone. Mining is completed with the same equipment used for mine development, and dilution from waste external to the ore zone is minimal. Negatively, productivity is lower than Longhole Stoping due to the smaller blast sizes and sequencing of backfill. Good control of drilling, blasting, and mucking is also necessary to minimise backfill dilution.

Figure 16.20 Typical Drift-and-Fill Mining Layout



Ivanhoe, 2021

Drift-and-Fill mining can be performed in either an overhand or underhand fashion. In overhand mining, as each horizontal slice of ore is exhausted, mining progresses to the slice above the previous section. In underhand mining, after each slice is mined and backfilled, mining progresses to the ore slice below and mining takes place beneath the paste fill. The use of overhand methods will lower cement content in paste fill and also the ability to down-break the development onto the paste fill.

Because the underhand method requires a higher percentage of cement to ensure the stability of the back, resulting in increased overall cost, the overhand Drift-and-Fill mining method is considered to be employed in the Platreef 2022 FS.

Ore zones are divided into 5.0 m H horizontal (or lifts), and 5 m wide ore drifts will be mined and backfilled adjacent to one another in a repeating fashion. Individual drift cuts are 5.0 m H x 5.0 m W flat back. Drifts will be driven with flat backs for recovery purposes.

Mining widths should be limited to 5.5 m to reduce the length of ground support in the back and walls to 2.4 m. If mining widths exceed 5.5 m, additional support will be required. Assuming a mining width of no more than 5.5 m, the mining sequence should be primary-secondary-tertiary to ensure that slender backfill ribs are not formed. For this method a 5.0 m width was utilised. An average of 0.15 m of over break will be used to estimate the overall dilution from each paste fill rib, this represents 6% dilution on the heading. Drill patterns will need to be determined and modified in practice to achieve a suitable over break. Overall average dilution for primary and secondary stopes estimated as approximately 5.0% of stope tonnes at 14% of undiluted stope grade.

Upon completion of each drift, the cemented rock fill (CRF) will be pushed into the drifts, filling tight to the back with an attachment to the LHD. After the CRF cures to the required strength, another drift will be driven next to the fill.

There will be no bulkheads to contain backfill, rock bunds may be used.

Stopes are extracted as primary, secondary or tertiary enabling access to multiple mining faces at all times and allowing greater productivity from an individual ore slice (Figure 16.21).

The sequence of primary, secondary and tertiary stopes allows the crews to maintain a 10 m pillar between active stope headings. The continual production rate of the stopes will be based on the rule that secondary stopes cannot be taken until primary stope paste fill set time is complete, and tertiary stopes cannot be taken till adjacent primary and secondary stope paste fill set times are complete. With the exception of the first primary stope the backfill fence construction is not considered on the critical path. It is assumed that the fence construction can be done concurrently with the other mining activities.

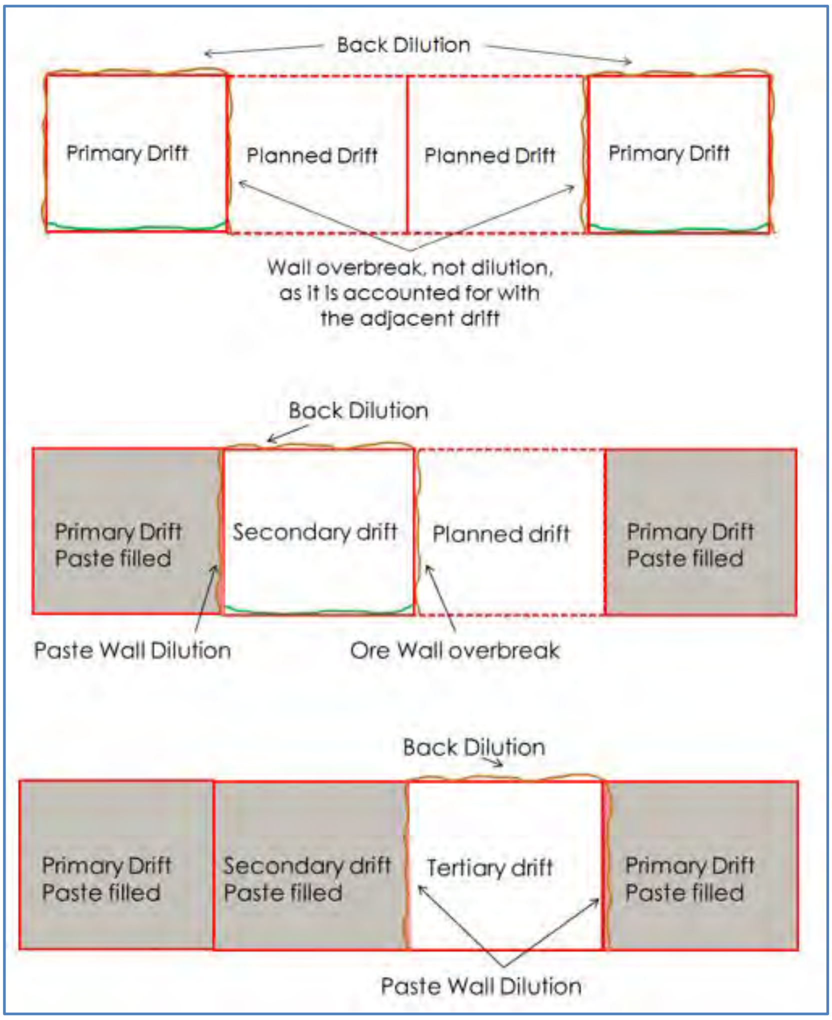
The following criteria was used to establish the production rate for this mining method:

- Each stope is 5 m x 5 m (flat back) in cross section and 15 m long, so using the density of 3.14 t/m^3 the total tonnage per stope will be 1,177.5 t.
- Using a single heading production rate of 354 t/d (4.5 m/d advance rate), one stope will be mined in approximately 3.5 days.
- The cure time for the paste fill is 28 days, before advancing to the adjacent secondary stope then the total active primary faces required for continuous mining will be eight faces. (28 days/3.5 days/stope + the initial panel ~9 panels).
- Each panel (defined as the combination of primary, secondary and tertiary) is 15 m wide (5 m/stope x 3 stopes), nine available panels translate to 135 m required for the total length of one active area, including the 5 m wide access drift (42,390 t).

The maximum allowed rate is based on the number of accesses that are available to a series of active areas. Each access from the ramp will allow for 354 t/d.

Second lifts in the Drift-and-Fill stopes will be accessed by slipping of the backs in the access ramp and installing new ground support.

Figure 16.21 Primary, Secondary and Tertiary Stoping

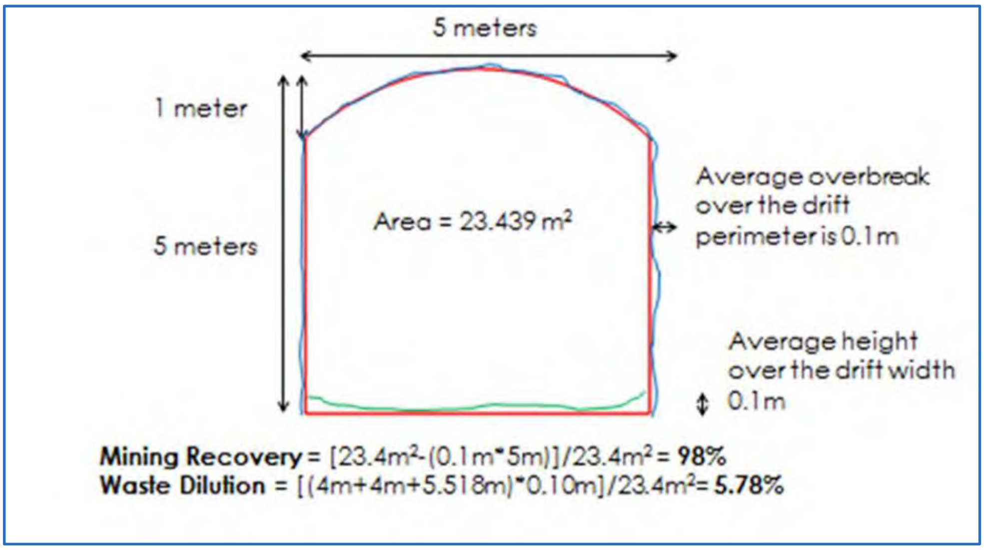


16.2.5 Dilution and Recovery Factors

16.2.5.1 Waste Drifts

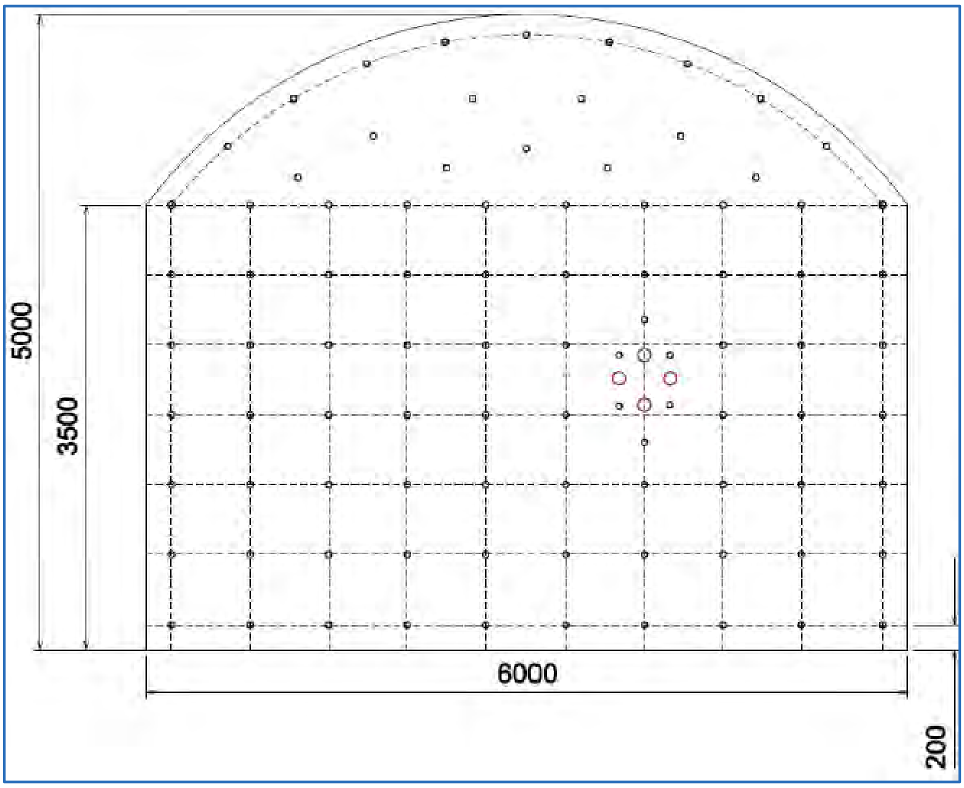
Waste drift recovery and dilution are 98.00% and 5.78% as shown in Figure 16.22.

Figure 16.22 Waste Drift Recovery / Dilution



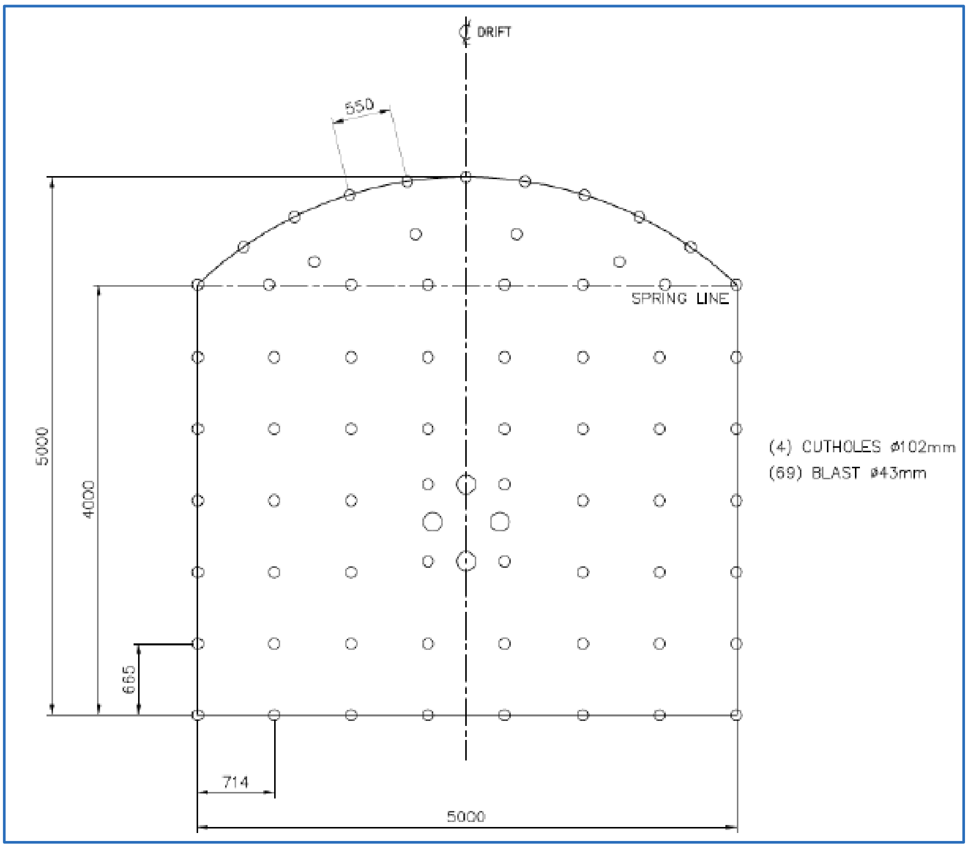
The drilling cycle for access development is based on the blast design pattern illustrated in Figure 16.23. All ore development will be done using a 4.66 m round length. Drilling will be done with an automated drilling system and drilling accuracy will be good to achieve a 4.51 m advance.

Figure 16.23 Ore Access Drift – Blast Design



The drilling cycle for waste development is based on the blast design pattern illustrated in Figure 16.24. The same blast pattern will be applicable to the 4.66 m round and the 3.05 m round. Allowance is made for probe drilling (pilot holes) and 5% for stuck rods and design growth. Development drilling will be done by using an automated drilling system with good accuracy. The more accurate drilling will result in an improved advance per blast, leaving only 150 mm sockets (bootlegs).

Figure 16.24 5 x 5 Access Drift – Cross Section Drill Pattern



16.2.5.2 Longhole Stopes

Grade and tonnes of each stope are calculated using dilution shells of 0.5 m thickness. Over break thickness for top, hanging wall, foot wall, end sides and secondary stope sides is 0.5 m and over break thickness for the floor is 0.3 m.

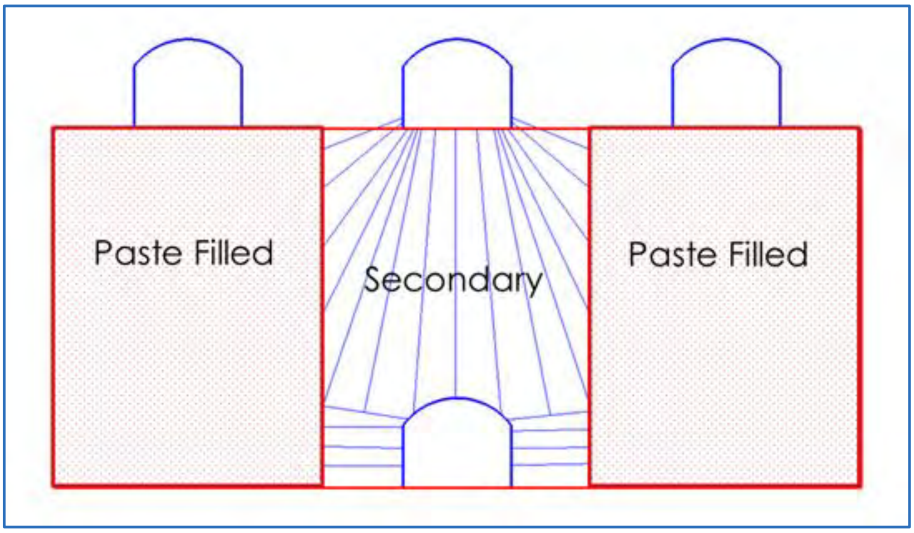
Table 16.11 lists the different stope sizes and the dilution percentages that will be applied to each primary and secondary stope.

Table 16.11 Dilution Percentage by Stope Length

Stope Length (m)	Primary Stope Dilution Factor (%)	Secondary Stope Dilution Factor (%)
10	8.2	14.0
15	5.9	11.7
20	4.1	9.9
25	3.6	9.5
30	3.3	9.1
35	3.1	8.9
40	2.7	8.6
45	2.6	8.4
50	2.5	8.3
55	2.4	8.2
60	2.3	8.1

The Platreef 2022 FS stoping design allows for blast holes to be fanned from a centre drift that is 6.0 m wide. This allows for accelerated topsill development but requires a pattern that drills toward the paste wall, thus increasing the chance that energy from the blast will be directed into the paste fill and increasing the chance for dilution. With proper drilling accuracy and explosive loading this dilution should be minimized thus making the 6.0 m wide development the preferred option. A typical drill pattern is illustrated in Figure 16.25 The resulting calculated thickness of overbreak for paste walls is 0.5 m.

Figure 16.25 Typical Drill Pattern for Longhole Stoping



Ivanhoe, 2021

16.2.5.3 Drift-and-Fill Stopes

The dilution from Drift-and-Fill stoping is an average between the primary, secondary, and tertiary drifts. The primary drift will have zero rib dilution, as the over break is assumed to be ore. The secondary drift will have one rib in ore and the other along a paste fill wall from the primary drift that was mined and filled next to it. The tertiary drift will be driven between paste fill walls from the adjoining primary and secondary drifts. An average of 0.15 m of over break will be used to estimate the overall dilution from each paste fill rib.

The 4.9% represents the average dilution that is applied to the Drift-and-Fill. Dilution grade estimated using average grade of material in immediate hanging wall. Overall average dilution for primary and secondary stopes estimated as approximately 5.0% of stope tonnes at 14% of undiluted stope grade. Dilution percentage for Drift-and-Fill stopes is shown in Table 16.12.

Table 16.12 Drift-and-Fill Stope Dilution Percentages

Drift Type	Paste Dilution (%)	Paste Floor Dilution (%)	Over break Dilution (%)	Average Dilution (%)
Primary	0.0	0.7	2.0	2.7
Secondary	2.2	0.7	2.0	4.9
Tertiary	4.4	0.7	2.0	7.1
Summary	2.2	0.7	2.0	4.9

16.2.6 Mining Recovery

A mining recovery factor of 96% was calculated for the Longholes Stopes which includes allowances for the following:

- Unrecoverable ore from design blast,
- Stope clean out, and
- Oversized muck left in stope (Including mine design recovery and oversize rock/unblasted wall ore).

A mining recovery factor of 98% was applied to Drift-and-Fill stoping areas and to stope end slashes.

16.2.7 Mine Access Designs

Mining zones included in the mine plans for the Platreef 2022 FS, occur at depths ranging from approximately 570 m to 1,170 m below surface. Shaft 1 and Shaft 2 are the primary access points to the mine, along with three ventilation raises.

Access from Shaft 1 and Shaft 2 to the mining areas will be via the three-principal access and haulage levels (the 750 m, 850 m, and 950 m Levels) and a series of interconnecting ramps. Additional mining sublevels will be developed as necessary from the ramps.

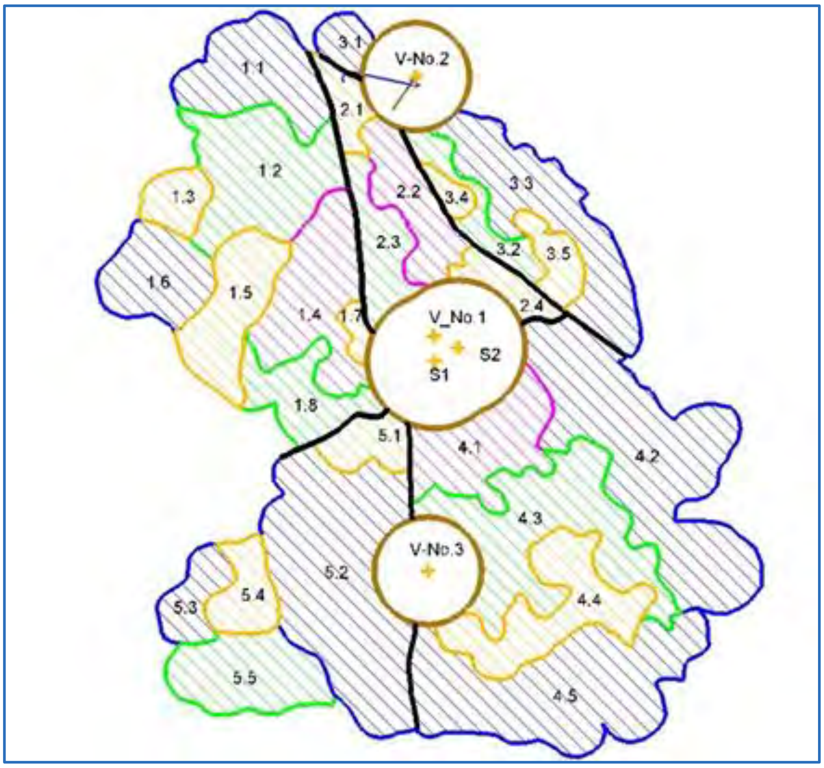
16.2.7.1 Shafts

Shaft functions and design parameters are summarised in Table 16.13. Shaft locations are shown in Figure 16.26.

Table 16.13 Shaft Functions and Design Parameters

Shaft	Function	Diameter (m)	Depth (m)
Shaft 1	Early Mine Development / Escape / Ventilation Intake	7.25	996
Shaft 2	Production – 6.19 Mtpa (Labour and Material / Mine Services / Ventilation Intake)	10.0	1,100
Vent. Raise 1	Ventilation Exhaust	6.0	950
Vent. Raise 2	Ventilation Exhaust	6.0	800
Vent. Raise 3	Ventilation Exhaust	6.0	725

Figure 16.26 Shaft Locations



Shaft 1

Shaft 1 has a 7.25 m diameter and sinking commenced in October 2015 and reached a planned depth of 996 m below surface in 2020. Essential mine services and auxiliary cage guides, for emergency egress, will be extended into the mine as the sinking progresses. This auxiliary cage will support initial mine development and later be used for emergency egress during mine production. Shaft services will be installed to supply the mine with essential utilities, including a temporary fuel line for early mine development. Shaft 1 mine services will provide a redundant backup once Shaft 2 mine services are installed and functional and will provide chilled intake ventilation to the deeper and hotter portions of the mine.

Shaft 1 sinking will include initial lateral development at the 450 m, 750 m, 850 m, and 950 m Levels. Once shaft sinking is complete, there will be a changeover from sinking buckets to a loading arrangement that will allow for 2,500 t/d of hoisting.

A raise borehole muck pass will be pulled to the 750 m Level through the 850 m Level from the 950 m Level to support fast track development to the high-grade ore zones.

Development will be scheduled to tie into Shaft 2 completion as soon as is practical.

Shaft 1 will be the dedicated cooling and refrigeration ventilation intake that will predominantly service the lower mining zones and have a maximum velocity of 15 m/s (620 m^3/s) to limit losses.

A schematic section of the Platreef Mine, showing Flatreef's thick, high-grade T1 and T2 mineralized zones, underground development work completed to date in shaft 1 and Shaft 2 and planned development work, is depicted in Figure 16.27.

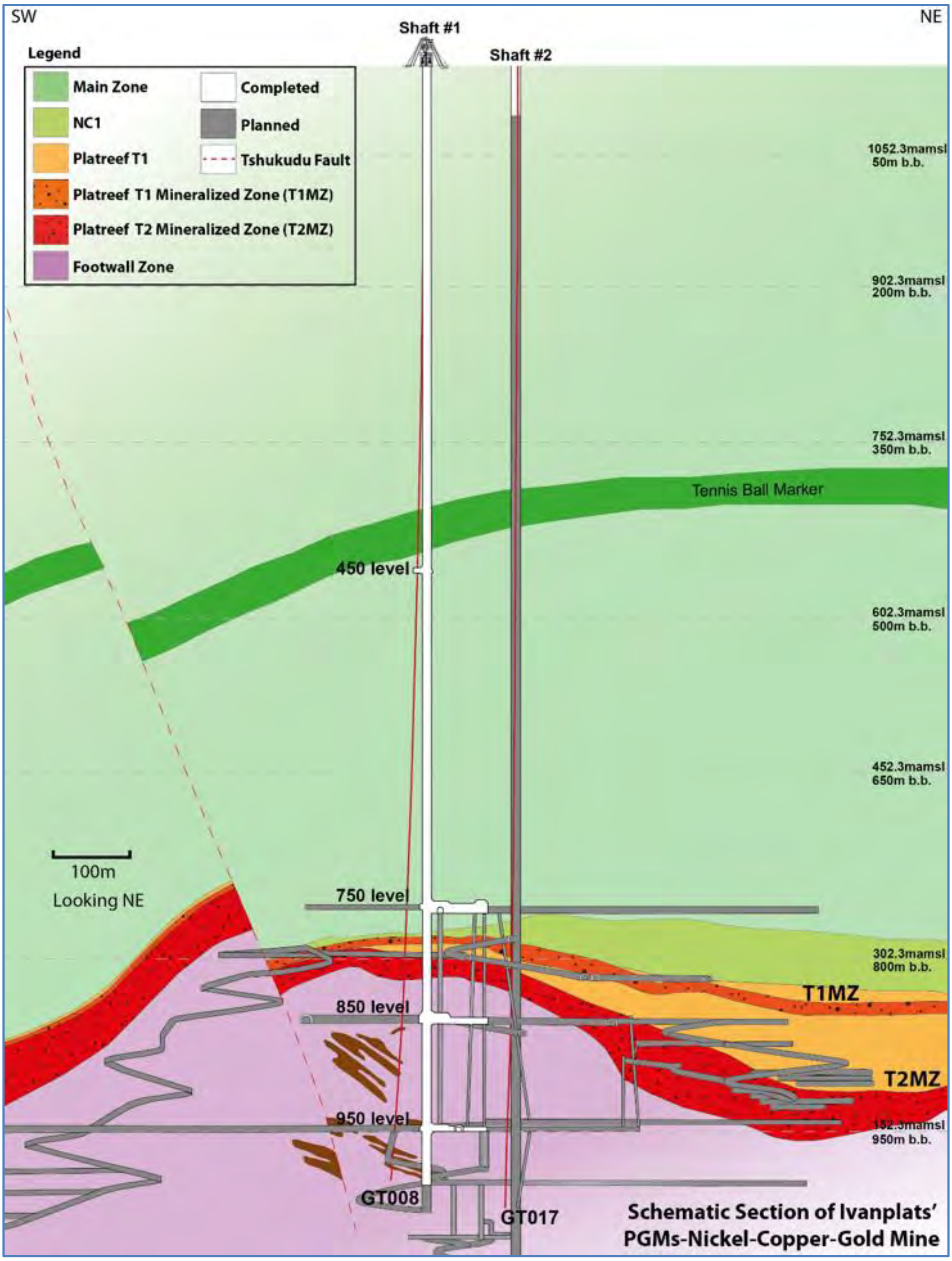
Shaft 1 main features are summarised below:

- Shaft functions – bulk sampling / ore delineation / Phase 1 mine development and stoping / ventilation intake / bulk air cooling / secondary escape
- Shaft location – N 2,664,470.563, E 4,199.930, with a collar elevation of approximately 1,102 mean sea level (MSL)
- Total depth – 996 m (has been sunk to the final depth in June 2020)
- Shaft diameter – 7.25 m (finished)
- Lining – 300 mm concrete (minimum)
- Hoisting / conveyance arrangement – sinking
  - Double drum hoist – 4.88 m diameter, 2,322 kW
  - Hoist rated line pull:
    - 215 kN (Kibble)
    - 250 kN (Jumbo)
    - 172 kN rated
  - Bucket payload – two 10 t buckets

  - Hoist speed – 10 m/s
  - Hoisting distance – 977 m
  - Hoisting capacity (dry) – 1,800 t/d at 16.0 h/d
  - Rope diameter – 43 mm
- Hoisting / conveyance arrangement – early mine development – Modify stage for skip loading. Require Skip/Cage change out in single compartment:
  - Double drum hoist – 4.88 m diameter, 2,322 kW
  - Headframe changeover required – 3 weeks (Will completed during 2021)
  - Hoist rated line pull – not available kg
  - Skip payload – two 12.5 t skips to match load of an LHD with 7 m^3 bucket
  - Hoisting capacity – 2,500 t/d at 11 h/day
  - Rope diameter – 46 mm
  - Skip discharge onto a loadout conveyor (Nominal 250 t/hr) that runs North to a stacker conveyor (1,450 t/hr). Each conveyor will be 1,350 mm wide
  - There is sufficient space in the stacker conveyor footprint to create 10 kt stockpile
- Hoisting / conveyance arrangement – auxiliary / escape:
  - Single drum hoist – 3.75 m (~12 ft.) diameter, 475 kW (640 hp)
  - Auxiliary cage – Double deck, eight people per deck, 3.0 t payload
  - Auxiliary cage hoisting speed – 6 m/s
  - Emergency load capacity – 3,000 kg
- Utility installations – early mine development:
  - Mine return water line – 100 NB
  - Existing slick lines x 2 - 250NB
  - Service water – 100 mm
  - Fire Water – 100 mm
  - Potable Water – 25 mm
  - Fuel line – 100 mm Flanged seamless tube (Shed 40 Grade A API 5L)
  - Vent line – Two, 1,600 mm diameter (63 inches)
  - Main power cables – 11 kV / 50 Hz. (5 x 240 mm^2 XLPE 11 kV feeders)
  - Communications cables:
    - Telephone
    - Leaky Feeder
    - Fibre Optic (single mode redundant pair)
- Intermediate pump station: 450 m Level

- Operating level stations: 750 m Level
- Operating level stations: 850 m Level
- Operating level stations: 950 m Level
- Shaft bottom: 996 m Level
- Downcast at maximum 12.0 m/s (2,360 ft/minute)

Figure 16.27 Schematic Section of the Platreef Mine



Ivanplats, 2020

Shaft 2

Shaft 2 will serve as the production shaft and primary access to the mine. It will be a 10 m diameter, 1,100 m deep concrete-lined shaft and will provide intake ventilation needs for the upper portion of the mine.

Shaft 2 is planned to have three hoisting systems: ore handling and hoisting systems, the main personnel and materials transport cage, and an auxiliary personnel cage. There are additional provisions for primary mine services.

Shaft 2 will feature two 40 t guided skips working in counterbalance. Total rock-hoisting capacity will be 6.40 Mtpa. The skip compartment will be screened off from the remaining shaft at the headframe, major stations, loading pocket, and dump pocket. No services will be installed in this compartment.

The personnel-and-materials cage in Shaft 2 will be a large 225-person, single-deck cage. The main cage is sized to accommodate the majority of the mining equipment with minimal disassembly. Oversize equipment (e.g., 50 t production truck) will be split at the articulated joint. The cage will be designed to allow hoisting of equipment and materials up to a maximum payload of 40 t from a depth of 1,100 m. All equipment and materials will be caged (no slinging). The cage is designed to be chaired with drive on / drive-off capabilities and will be balanced via a counterweight. Shaft 2 includes a 20-person, two-deck auxiliary cage with a 3 t payload.

There will be two main ventilation intake shafts: Shaft 1 and Shaft 2, which will supply fresh air to the mine. Intake from Shaft 2 will be designed for a maximum velocity of 11 m/s (860 m^3/s).

Shaft 2 main features are summarised below:

- Shaft functions – production / service / ventilation intake
- Shaft location – N 2,664420.563, E 4,113.328, with a collar elevation of approximately 1,100 m MSL
- Shaft bottom elevation: 0 m MSL
- Total depth: 1,100 m
- Shaft diameter: 10 m (finished)
- Lining: 300 mm concrete (minimum)
- Surface ore/waste bin: 160 t capacity (four skips)
- Fixed guides
- Skip compartments screened off at the stations and loading station (no brattice wall)
- Friction hoists (Koepe): Tower mounted, production hoist:
  - Hoist type: Friction hoist (Koepe)
  - Number of motors: One
  - Nominal motor power: 9,217 kW (12,359 hp); fed at 11 kV

- Drum diameter: 6.0 m
- Number of head ropes: Four
- Head rope diameter: 54 mm
- Number of tail ropes: Six
- Tail rope diameter: 56 mm
- Skip payload: two 40 t skips in balance
- Moisture content: 3–5% by weight
- Hoisting distance: 1,088.84 m
- Hoisting speed: 18 m/s
- Hoist hours/day: 17 h average
- Hoisting capacity: 1,278.36 t/h
- Hoisting capacity per year: 6.19 Mtpa

- Service hoist:
  - Hoist type: friction hoist
  - Number of motors: One
  - Nominal motor power: 2,515 kW (3,372hp); fed at 11 kV
  - Drum diameter: 6.0 m
  - Number of head ropes: Four
  - Head rope diameter: 54 mm
  - Number of tail ropes: Four
  - Tail rope diameter: 56 mm
  - Maximum cage capacity load: 40 t
  - Single deck cage: Designed capacity of 225 people
  - Actual dimensions of cage deck: 3,450 mm W x 9,112 mm L x 9,424 mm H
  - Hoisting speed: 10.0 m/s
  - Hoisting distance: Up to 1,088.84 m
- Auxiliary hoist: Single drum:
  - Number of motors: One
  - Nominal motor power: Auxiliary hoist – 2.5 m (~12 ft.) diameter, 309 kW (740 hp); fed at not available kV
  - Drum diameter: 2.5 m
  - Rope diameter: 22 mm
  - Two deck auxiliary cage: 20 people per deck
  - Hoisting speed: 6.0 m/s

- Utility installations:
  - Compressed air: 250 NB
  - Service water: 200 NB (~8 inches)
  - Fire water: 200 NB (~8 inches)
  - Potable water: 50 mm (~2 inches)
  - Dewatering: 300 NB (~12 inches)
  - Drain line: 100 NB (~4 inches)
  - Fuel line: 100 mm (~4 inches)
  - Main power cables: 11 kV / 50 Hz. (5 x 240 mm² XLPE 11 kV feeders)
  - Earth cable: 10 x 95 mm² BCEW
  - Communications cables (10 x 35 mm Fibre):
    - Telephone
    - Leaky feeder
    - Fibre optic
- Cable pocket station: 450 m Level
- Operating level stations: 750 m Level, 850 m Level, and 950 m Level
- Crusher and Settler level: 1000 m Level
- Skip loading and Pump station: 1,050 m Level
- Skip tail rope changing level: 1,050 m Level
- Service rope changing level: 1,050 m Level
- Shaft bottom station: 1,100 m Level
- Downcast at maximum 10.0 m/s

16.2.7.2 Ventilation Raise 1

- Raise function – one of the primary exhausts while also serving as a dedicated ventilation exhaust for underground maintenance shops, fuel stations and early mine development.
- Raise location: Surface to 750 m Level (X –4228, Y –2664376) and 750 m Level to 950 m Level (X –4210, Y –2664396). This may be modified slightly to suit actual site conditions.
- Raise diameter: 6.0 m finished (6.1 m reamer head).
- Construction method: Raisebore
  - Pilot from surface to 750 m Level
  - Ream from 750 m Level to Surface
  - Pilot from 750 m Level to 950 m Level
  - Ream from 950 m Level to 750 m Level

- Lining – between 800 m and 850 m deep from collar with 50 mm shotcrete (minimum) (based on Mine Design Criteria Report). There will be an update based on geotechnical borehole during 2021.
- Maximum up cast velocity of 20 m/s.
- This will be equipped for second egress access.

16.2.7.3 Ventilation Raise 2

- Raise functions: Ventilation exhaust.
- Raise location: X –4206, Y –2663486, with a collar elevation of approximately 1,113 m MSL.
- Raise bottom elevation: +150 m MSL.
- Total depth: 963 m.
- Raise diameter: 6.0 m finished (6.1 m reamer head).
- Lining – between 85–86 m below collar, 754–771 m below collar and 929–964 m below collar 50 mm shotcrete (minimum) (based on Mine Design Criteria Report). There will be an update based on geotechnical borehole during 2021.
- Operating level stations: 750 m Level, 850 m Level, and 950 m Level.
- Shaft utilities: Installed communications and control for escape hoist.
- Up cast at maximum: 20 m/s.

16.2.7.4 Ventilation Raise 3

- Raise functions: Ventilation exhaust
- Raise location: N –2,665,250.000, E –4,225.000, with a collar elevation of approximately 1,093 m MSL.
- Raise bottom elevation: +350 m MSL
- Total depth: 743 m
- Raise diameter: 6.0 m finished (6.1 m reamer head)
- Lining: Must be fully lined from surface to bottom with 50 mm shotcrete (minimum) (based on Mine Design Criteria Report). There will be an update based on geotechnical borehole during 2021.
- Up cast at maximum: 20 m/s

16.2.7.5 Internal Ventilation Raises

- Numerous other internal ventilation raises will be required.
- All internal raises will 3 m in diameter.
- Locations and length will vary.
- Most are to be constructed by raise boring.

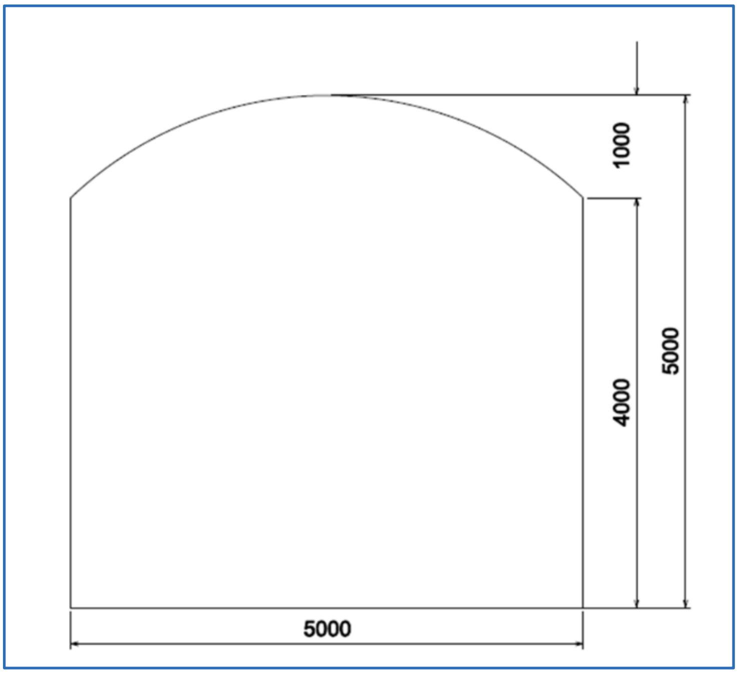
- Shorter raises between sublevels may be constructed using vertical crater retreat (VCR) methods or drop raising.
- Internal ventilation raises will have ladders and landings for secondary means of egress and eliminate the need for raises in Ventilation Raise 2 and Ventilation Raise 3.

16.2.7.6 Access Levels and Ramps

Main Access / Haulage Levels

- Access / haulage drifts connecting to vent shafts: 5 m W x 5 m H
- Other haulage drifts: 5 m W x 5 m H
- Ore pass access drifts: 5 m W x 5 m H
- Ventilation raise access drifts: 5 m W x 5 m H
- Minimum curve radius: 25 m
- All developments are arched, illustrated in Figure 16.28.

Figure 16.28 Lateral Development Profile



Mining Sublevels and Access Ramps (Ore Reserve Development)

- Sublevel lateral drift: 5 m W x 5 m H
- Stope access drifts: 5 m W x 5 m H

- Ore pass access drifts: 5 m W x 5 m H
- Ventilation raise access drifts: 5 m W x 5 m H
- Sublevel access ramps: 5 m W x 5 m H
- Crusher and shaft bottom access ramps: 5 m W x 5 m H
- Minimum curve radius: 25 m
- Maximum grade: 15% or 8.5 degree
- All drifts are fully arched (Figure 16.28)

16.2.7.7 Other Mine Access Criteria

- All drift widths are based on equipment size plus minimum 1.5 m clearance
- Ramps: Maximum Gradient of 15%
- Ramps: Average Gradient of 12.5%
- Ramps: 30 m Radius Curve (minimum)

16.2.8 Ventilation

BBE has been tasked by OreWin (mining engineers) to participate as specialist mine ventilation engineers for a feasibility level study. The objective of the work is to review previous work and optimise current mine planning in-line with mining initiatives to arrive at a fit-for-purpose and cost-effective project. The ultimate outcome of this study is for BBE to determine the primary ventilation and refrigeration requirements and provide CAPEX and OPEX to an appropriate level of accuracy.

The mine will be accessed via a 7.25 mØ ventilation shaft (Service Shaft) and 10.0 mØ production shaft (Main Shaft) used for fresh air. Main fan stations are planned at three 6.0 mØ Ventilation Raises (Vent Raise No.1, 2 and 3) which will be used to exhaust 1 500 m^3/s of air to surface. There will be a need for refrigeration at this depth which will be provided by an air-cooling system on surface.

Platreef Mine is a highly mechanised hot mine where the pre-dominant ventilation design criteria will relate to heat management. The primary and secondary ventilation systems are designed to provide an exhaust system on surface and on each production level. The two main intake shafts (Service Shaft and Main Shaft), located at the centre of the mining blocks, will provide fresh intake air, while ventilation raise No.1 (VR1; near the intake shafts), ventilation raise No.2 (VR2; in the north), and ventilation raise No.3 (VR3, in the south) will serve as exhaust shafts. Fresh air will be distributed via intake airways on 750 m Level, 850 m Level and 950 m Level. Declines will provide ventilation between levels. Ventilation controls will be used to direct air to the active production areas from where sub-level return air raises (RARs) will exhaust air from each level.

Underground mining operations will take place in separate mining orebodies that share a common twin vertical shaft system for ore and personnel transport. Capital development from the Services Shaft will start in 2022 on 750 m Level, 850 m Level and 950 m Level. The production schedule aims to achieve first production in 2025. Production level development will continue until steady state production is reached in 2032 when 5 Mtpa will be mined.

A Ventilation on Demand (VoD) philosophy has been adopted from the early planning phases and ventilation requirements were based on mining activities rather than the available drifts.

Initially primary in-take air will be through the Service Shaft and will use an exhaust ducted system with six axial flow fans installed on surface. These fans will be used for initial development and the establishment of shaft station infrastructure. By 2024 VR1 will be complete which will then be used as the main return system. Temporary fans will be installed underground on 750 m Level in two locations. By 2025 early production commences and addition airflow is required. The 110kW fans will be replaced with two sets of 650 kW fans. By 2029 the Main Shaft and VR2 will be complete and the main exhaust fans (2200 kW) will be installed on surface at VR1 and VR2. In 2032 VR3 will be complete and the remaining surface fans will be installed providing the full design airflow quantity of 1550 m^3/s.

Three exhaust fan stations are planned, one each vent raise (i.e. at VR1, VR2 and VR3). Each fan station will be identical and will comprise a bifurcated centrifugal fan arrangement.

A two cell surface bulk air cooler (BAC) is required to meet the cooling needs underground. Ambient air drawn into the Service Shaft will be cooled using the BAC, an open spray system supplied with chilled water from a refrigeration plant. The cooling demand required of the BAC, and therefore refrigeration plant, changes as the mine expands but the infrastructure can be designed to cater for initial (7.5MWBAC) and LoM (20MWBAC) production needs.

The refrigeration machines will be phased in over 4 years. Four machines are planned for LoM. Two machines will be installed in 2029 for the ramp up production needs of the mine, and the remaining two units will be installed in 2032.

The capital cost for the ventilation and refrigeration infrastructure was determined to be R606M. This included costs of the main fans, refrigeration plant and temporary underground fans.

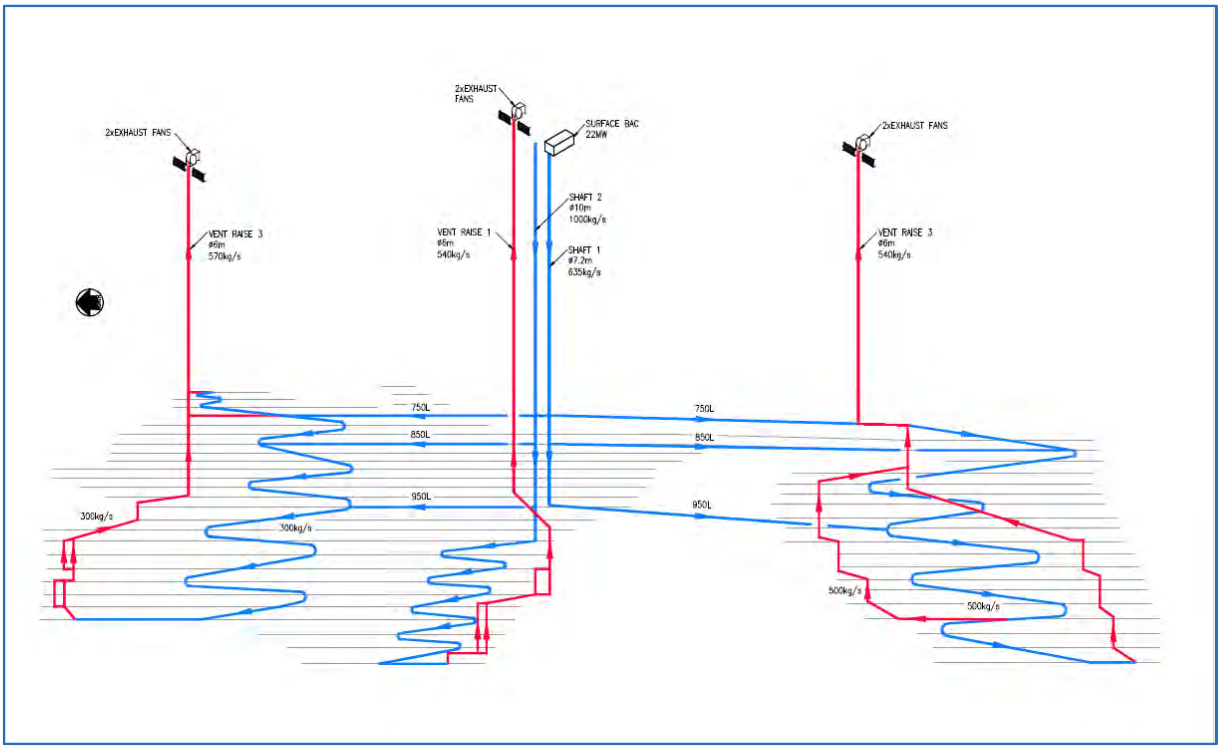
The operating cost was found to be approximately R263m per year during peak production. This included primary and secondary fans and the refrigeration plant power costs.

16.2.8.1 Primary Ventilation

The ventilation design criteria itemize several applicable conditions that are used to determine air quantity requirements for the LoM. These conditions are determined by the presence of diesel engine emissions, airborne dust, gases and blast fumes in the air stream, by minimum airway velocity requirements (personnel exposure) and by heat energy removal rates in production and development drives. Considering their low-level impact in ensuring worker health and safety, administrative controls are kept to a minimum. In addition, ventilation modelling is used to verify the effect on the air quality and air temperatures (wet-bulb and dry-bulb) in production drifts. The primary ventilation layout is shown in Figure 16.29.

An aligned understanding of mining activities and operations is necessary to define accurately air quantity requirements. Underground mining operations at Platreef will take place in separate mining orebodies that share a common twin vertical shaft system for ore and personnel transport. Capital development from the Services Shaft will start in 2022 on 750 m Level, 850 m Level and 950 m Level. The production schedule aims to achieve first production in Q4'24. Production level development will continue until steady state production is reached in 2030 when 5.2 Mtpa will be mined.

Figure 16.29 Primary Ventilation Layout



BBE, 2021

During steady state, fresh air will be supplied from surface through the 7.25 m diameter Services Shaft, and the 10.0 m diameter Main Shaft.

It is intended for fresh air to enter the mine from the shafts to 750 m Level, 850 m Level and 950 m Level. Fresh air will be distributed to the various production areas through a combination of fresh air passes (FAP) and ramp systems. Each production level will be ventilated as separate ventilation districts. Heat, dust and blasting fumes will be extracted directly, the main return system utilizing a forced-exhaust system. This ventilation strategy has major benefits on re-entry time, mining flexibility and workplace conditions in general. If the production levels were to be ventilated by force ventilation systems, the mine's total heat load would be increased by an additional 7.5 MW, necessitating an additional 8.0 MWR refrigeration plant on surface. The benefit of using the extraction ventilation system is therefore evident by the savings in the additional cooling power that would have to be otherwise provided. The Service Shaft (7.25 mØ) can support mining activities during the initial build-up and early production up to Q2'27. Intake capacity from the Main Shaft (10.0 mØ) will be required from Q3'27.

During steady state, return air will exhaust to surface through three (3) exhaust shafts using the main surface exhaust fan stations. Modelling indicates that the reduction in size will have no significant impact on the overall ventilation system. To cater for a second egress VR1 will be equipped for hoisting purposes until Main Shaft is commissioned. This temporary arrangement will make use of a rope guided hoisting system, which will limit the velocity in VR1 to <8m/s when hoisting takes place. To achieve the reduced velocity two of the underground main fans will be stopped, which will limit underground production. For this reason, VR1 hoisting arrangement can be used in emergency situations (not on a permanent basis). Should the VR1 hoisting arrangement be required on a permanent basis, VR3 must be accelerated and commissioned by December 2023. Each production level is connected to a series of return air passes (RAP) which are connected to the main exhaust shafts through several dedicated return airways. Main return airways are strategically selected to ensure each mining area can be ventilation effectively. Return airways are situated on 750 m Level and 850 m Level. Capacities of both return airways must be increased from single 5 m x 5 m drives to dual 5 m x 5 m drives to accommodate the required airflow. In addition, the 750 m Level return airway must be extended beyond the Main Shaft position to ensure used air can return to VR1 without contaminating the intake air.

16.2.8.2 Secondary Ventilation

Ventilation on Demand (VoD) philosophy was adopted from the early planning phases and ventilation requirements were based on mining activities rather than the available drifts. The primary ventilation requirements can therefore be reduced from 20 m^3/s to 0 m^3/s when LHDs or blasted drifts have no production activities taking place or can be reduced to 10 m^3/s when other mining activities take place resulting in a reduction in ventilation infrastructure and operating cost.

The initial ventilation strategy is to exhaust about 120 m^3/s using two sets of three TVT2 fans on surface arranged in series and connected to ø1 500 mm ducting that extends to the three levels. Each duct set delivers about 40 m^3/s to each level. On each level there are two additional 110 kW fans that assist the pressure of the exhausting ducts. The ducts are reduced from ø1 500 mm to ø1 220 mm.

The stoping method will be long hole stoping and drift and fill. The stopes will be drilled between the sub-levels from cross cuts using vertical drill rigs. The broken ore will be removed from extraction drives using diesel LHDs which will transport the ore to stockpile drives situated on every level, before being hauled by trucks to the Service Shaft crusher and silo area.

Intake air will be supplied to the production areas via the decline and Main Shaft until 1,100 m Level, from where the FAR commences and further supports the decline. The FAR follows the production levels to shaft bottom with connections to the decline and perimeter drives on the levels. On production levels air will be supplied to all cross cuts by silenced auxiliary fans and ducting, with in-line regulators determining the quantity of air required per crosscut.

An exhaust-overlap ventilation system will be used in the production drifts. The intake of the exhaust column must be carried to a point not exceeding 30 m from the face. The distance between the discharge of the force column and the advancing face should be designed in such a way to ensure that the ventilation duct will not be more than 20 m from the face after the blast. The minimum overlap distance between the exhaust column intake and the force column intake points should at least be 10 m and not exceeding 25 m. Fans in the exhaust column must be positioned in such manner that the exhaust column will remain under negative pressure, thus ensuring that no exhaust fumes leak back into the intake air flowing to the face.

16.2.8.3 Surface Main Fan Stations

Three exhaust fan stations are planned, one each vent raise (i.e. at VR1, VR2 and VR3). Each fan station will be identical and will comprise a bifurcated centrifugal fan arrangement. A description of the planned fan station, as well as the performance requirements, is detailed in the section below.

A bifurcated centrifugal fan station will be installed. Two fans will be operational to achieve the full duty point. The peak operating duty point per fan station will be 550 m^3/s at a shaft collar pressure of 6.0 kPa at 0.97 kg/m^3. A minimum fan station footprint size of 60 m (l) x 25 m (w) will be required for the fan station as shown in Figure 16.30. The fan performance curve of the centrifugal flow fan is shown in Figure 16.31.

Figure 16.30 Centrifugal Fan Station Footprint

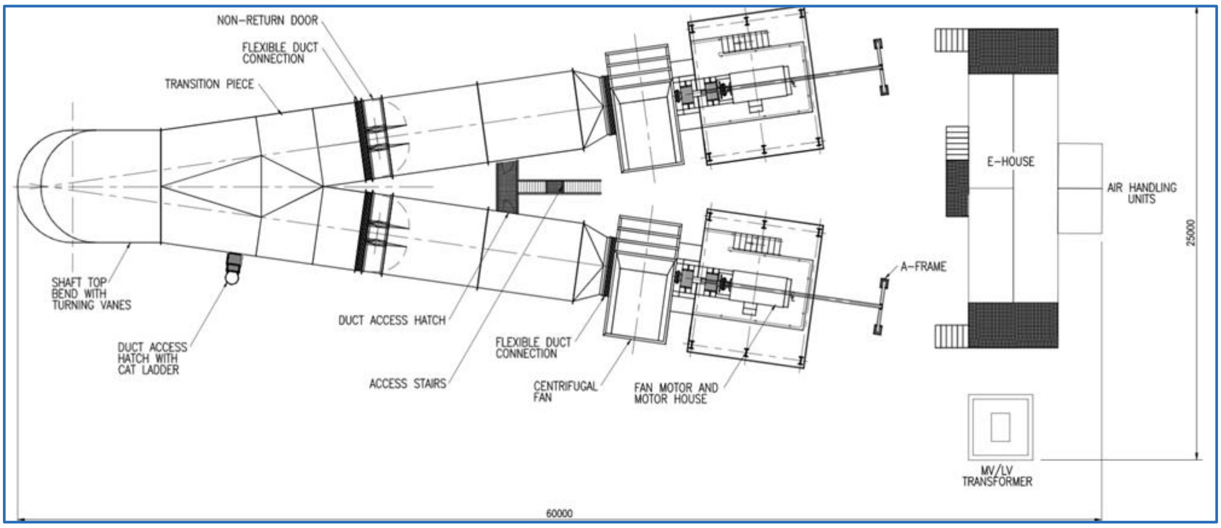
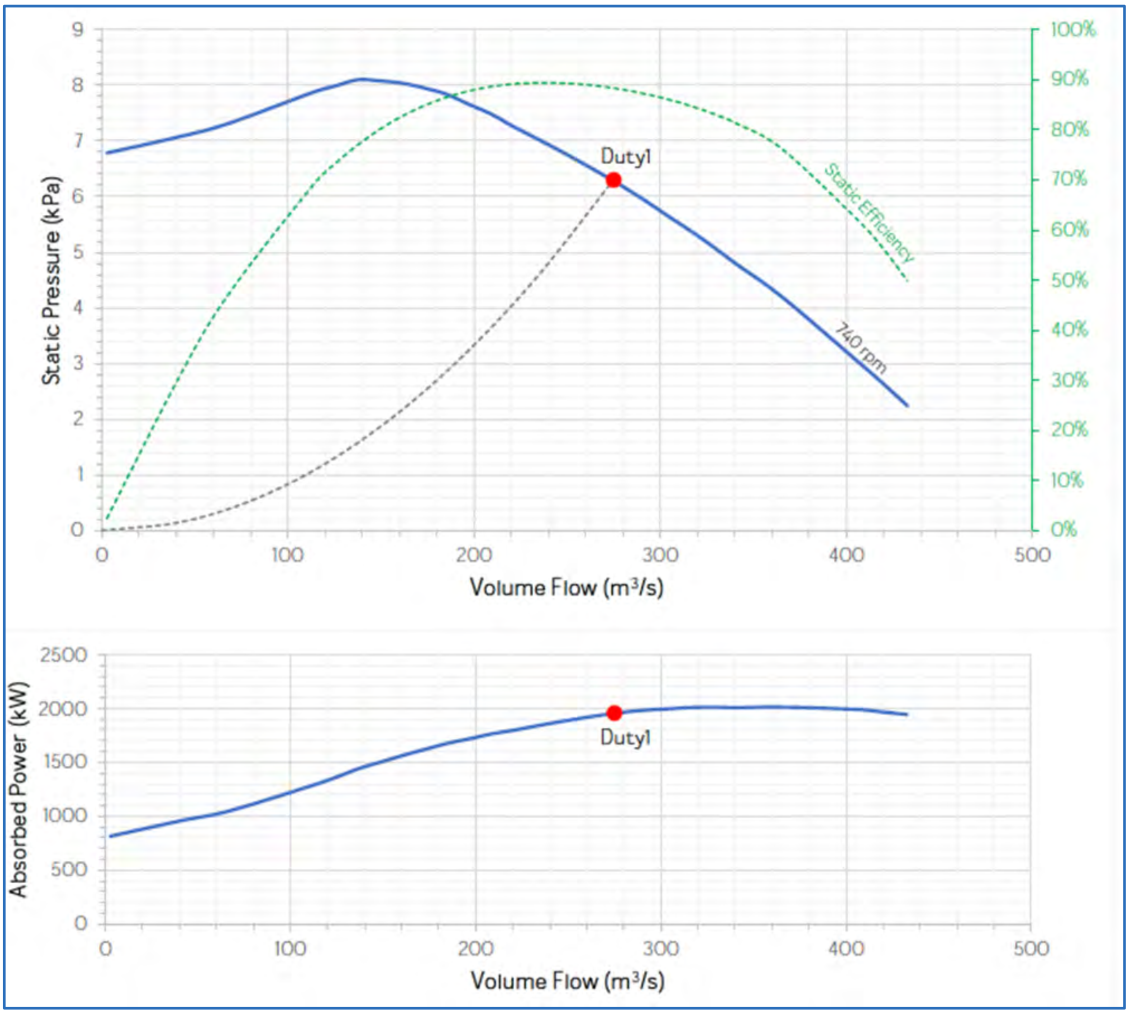


Figure 16.31 Main Surface Fan Curve



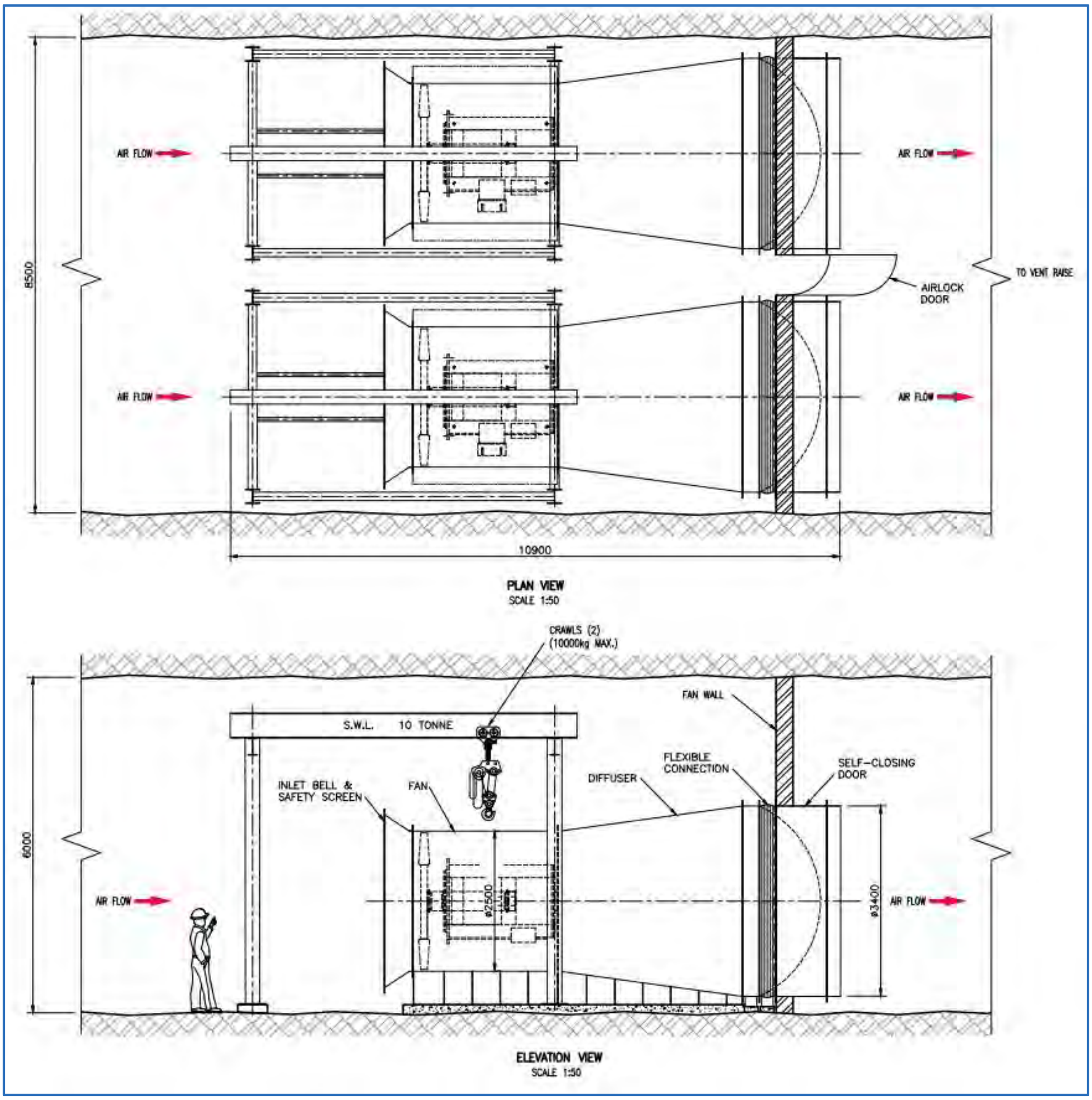
BBE, 2021

In the event of a power failure, one of the main fans can be started via the mine's standby generator set. VSDs will be used to keep the inrush current to a minimum. The fan performance will be modulated to provide sufficient air for emergency requirements.

16.2.8.4 Underground Fan Stations

Two main underground fans stations are planned on 750 m Level. Temporary fans will be installed in two RAWs in a wall. From 2023 to 2025 each fan wall will be equipped with three 110kW fans. In 2026 the 110kW fans will be replaced with 650kW fans (two fans per wall). The 650kW fans will be installed in the same location with a modified fan wall. A typical underground fan wall installation is shown in Figure 16.32.

Figure 16.32 Typical 650 kW Underground Fan Station



BBE, 2021

16.2.8.5 Refrigeration and Air Cooling

A surface bulk air cooler (BAC) is required to supplement the cooling needs underground. Ambient air drawn into Service Shaft will be cooled using the BAC (open spray system) supplied with chilled water from a refrigeration plant. The cooling demand required of the BAC, and therefore refrigeration plant, changes as the mine expands but the infrastructure can be designed to cater for initial and LoM production needs. This section details the requirement and the phase in of the required infrastructure. The BAC duty requirements for the various phases of the mine shown in Table 16.14.

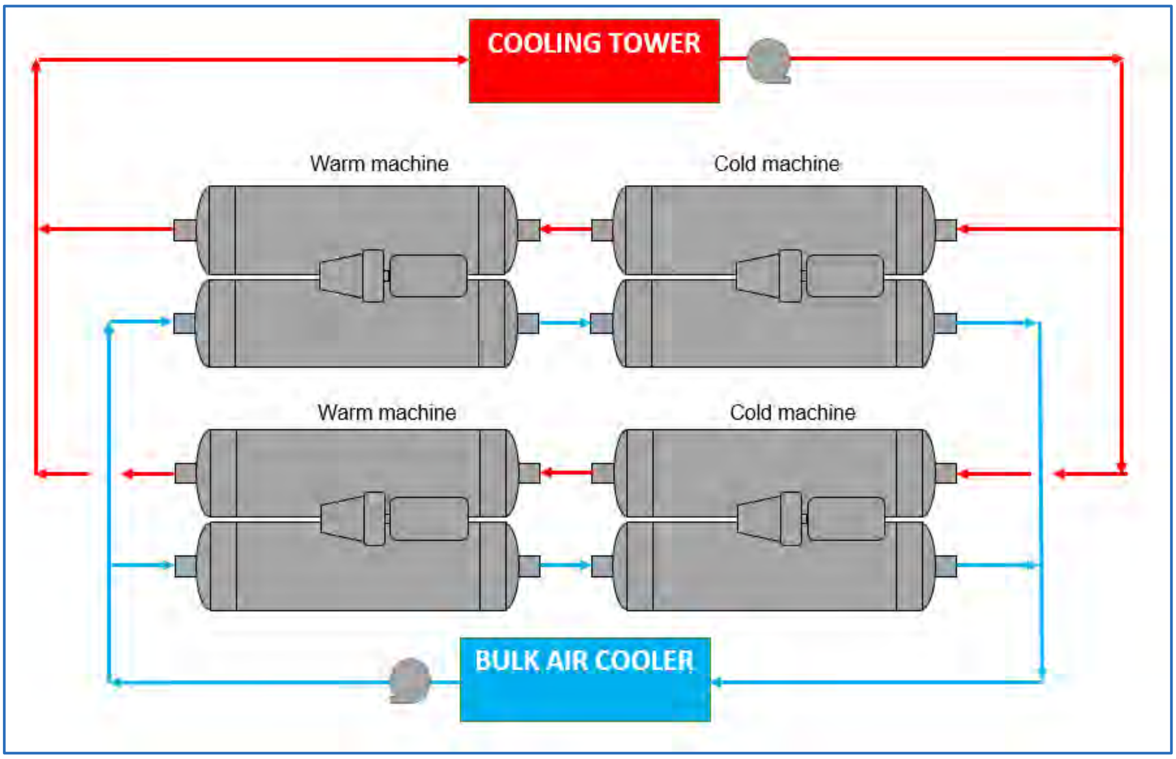
Table 16.14 The BAC Duty Requirements for Each Phase

Description	Year	BAC duty (MW)	Installed Refrigeration (MW)
Phase 3 – Ramp up to 5 Mtpa	2029 – 2031	7.5	~10.5
Phase 4 – LoM	2032 onwards	20.0	~21.0

The refrigeration machines will be phased in to meet the cooling requirement as needed. Four machines are planned for LoM. Two machines will be installed in 2029 for the ramp up production needs of the mine, and the remaining two units will be installed in 2032. The machines will all operate and there will be no installed standby (strategic spares on site). The plant room will be designed and built upfront to house all four machines.

Each refrigeration machine pair will be installed in a series counterflow arrangement, and two pairs will be installed in parallel. The series counterflow arrangement provides optimal thermal efficiency and a favourable capital cost of equipment. Return water from the BAC will enter the first refrigeration machine's evaporator (warm machine) and then enter the second refrigeration machine's evaporator (cold machine) where it will be cooled to the required leaving chilled water temperature. Similarly return water from the cooling tower will enter the cold machine's condenser and then discharge into the warm machine's condenser before returning to the cooling tower. This concept is illustrated in Figure 16.33.

Figure 16.33 Parallel Pair Series Counterflow Arrangement



BBE, 2021

Condenser cooling towers, located in close proximity to the refrigeration plant, will be used for heat rejection purposes. A four cell, counter flow, mechanical draught cooling tower is proposed using splash pack fill. The cooling tower structure may have an FRP structure with a concrete basin or may have an all concrete structure. The choice largely depends on cost and life expectancy (a concrete structure will cost more but have a longer operating life). Each cell will be equipped with a 45kW axial flow fan complete with variable speed capability for head pressure control.

The condenser pumps will be end suction, single-stage centrifugal pumps directly coupled to 150 kW, 4-pole motors operating on 525 V power supply. Two operating pumps will be installed with a fully piped-in standby. The pumps will be protected by strainer screens installed in the sumps and there will be a water strainer station installed before the refrigeration machines.

A two cell surface BAC will be installed in close proximity to Service Shaft. Air will enter a connecting drift, adjacent to the main shaft, and mix with air from surface at a sub-bank located between 4 and 10 m below surface.

A BAC is an open spray system comprising an enclosure complete with sprays, pumps, piping and fans. A concrete BAC is proposed at this stage and both cells will be built when cooling is first required in 2029. During this initial phase it is not necessary to equip both cells with piping or pumps nor will it be necessary to install all four fans (since the balance will bypass and be pulled down the shaft).

16.2.9 Mine Backfill

The mining methods selected for Platreef require backfill as a support medium. The following three backfill products will be used:

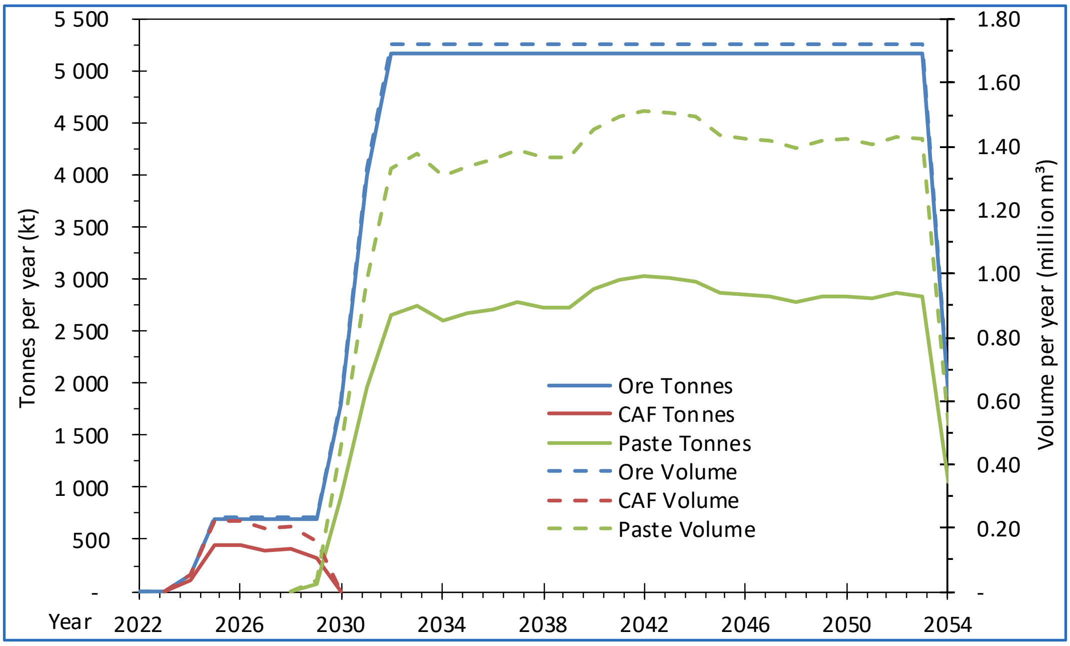
- Cemented Rock Fill (CRF), referred to as Interim Backfill.
- Cemented Paste Fill (CPF), referred to as Long Term Backfill.
- Waste Rock (from mine development used with CRF and CPF).

CRF will be used for the initial 700 ktpa mining production in D&F stopes. Once the production is increased, mining will be via transverse LHOS and filling will transition to CPF.

The detailed CRF system and CPF system design and cost estimates are presented in Paterson & Cooke reports 1051-RP-35-001 and 1051-RP-35-002 (Ivanhoe document numbers).

The backfill schedule and quantities are shown in Figure 16.34.

Figure 16.34 Backfill Schedule



16.2.9.1 Design Criteria

The backfill requirements and main system design criteria for both the CRF and CPF system are presented in Table 16.15 and Table 16.16 respectively.

Table 16.15 D&F Backfill Requirement and CRF Design Criteria

Item	Value/Description
Backfill medium	Cemented Rock Fill
Mining method	Drift and fill only
Operating days	360 days per annum
Hours per day	24 hours
Shifts	2 x 11.5 h backfill shifts per day 2 x 8 h of shifts available for trucking 2 x 10 h shifts per day for crushing
Plant availability	CRF plant availability: 90% Crushing plant availability: 75%
Ore production for D&F	Average: 700 000 t/y Average: 58 500 t/mth Average: 2000 t/d
Filling allowances	Backlog filling allowance: 25% Overbreak allowance: 5%
Backfill voids	100% of D&F (i.e. no waste-rock only filling as most areas will be mined on top)
CRF plant design throughput	55 800 t/mth 2 080 t/d 130 t/h (60 m³/h)
Backfill Panel Size	Drift and Fill: 5m wide x 5 m high (arched), with primary, secondary and tertiary drifts
Required backfill strength	Up to 400 kPa – Drift and Fill

Table 16.16 LHOS Backfill Requirements and CPF Design Criteria

Item	Value/Description	
Backfill medium	Cemented Paste Fill	
Mining method	Long-hole Open Stoping (LHOS)	
Operating days	Process plant: 360 d/y, 93.3% availability (8061 hours) UG mine: 360 d/y Paste plant: 360 d/y	
Paste plant utilization & availability	Availability: 324 d/y, 90% availability Utilisation: 227 - 268 d/y, 70% - 83%utilisation	
Time available for backfill	2 x 11.5 h backfill shifts per day (mining) 2 x 10 h available for filling per day	
Ore production for LHOS	Maximum: 5 200 000 t/y; 433 300 t/mth; 14 500 t/d	
Filling allowances	Backlog filling allowance: 10% Overbreak allowance: 5%	
Paste plant throughput	To match 4.4 Mtpa process plant throughput: 550 dry t/h nominal, 2 modules of 275 dry t/h To match 5.2 Mtpa process plants throughput, including 100 dry t/h of filter cake feed: 650 dry t/h maximum, 2 modules of 325 dry t/h	
Backfill Voids	100% (LHOS areas to be mined in cut and fill method) 50% as primary stopes (TBC) 50% as secondary stopes (TBC)	
Backfill Panel Size	LHOS: Nominally: 15 m on strike, 20 m high, length varies between 10 to 60 m Nominal Stope Size: 15000 m³ (TBC)	
Required backfill strength (SRK 01)	Primary plug pour:	150 kPa at 2 days
	Primary bulk pour:	335 kPa at 28 days
	Secondary plug pour:	150 kPa at 2 days
	Secondary bulk pour:	150 kPa at 28 days
	Capping pour all stopes:	500 kPa at 28 days

16.2.9.2 Cemented Rock Fill (CRF) Testwork

CRF test work was conducted using large cylinders (1.2 m high, 0.6 m diameter) to avoid any boundary and size effects. The waste rock samples received were crushed to two fuller grading curves to determine the effect of size grading within the expected range received from the crushing circuit, presented in Figure 16.35. The UCS versus W:B ratio correlation determined by the test work is presented in Figure 16.36.

For detailed results and data refer to Paterson & Cooke report 1051-TM-35-002 (Ivanhoe report number).

Figure 16.35 Waste Rock Sample Gradation for Tests

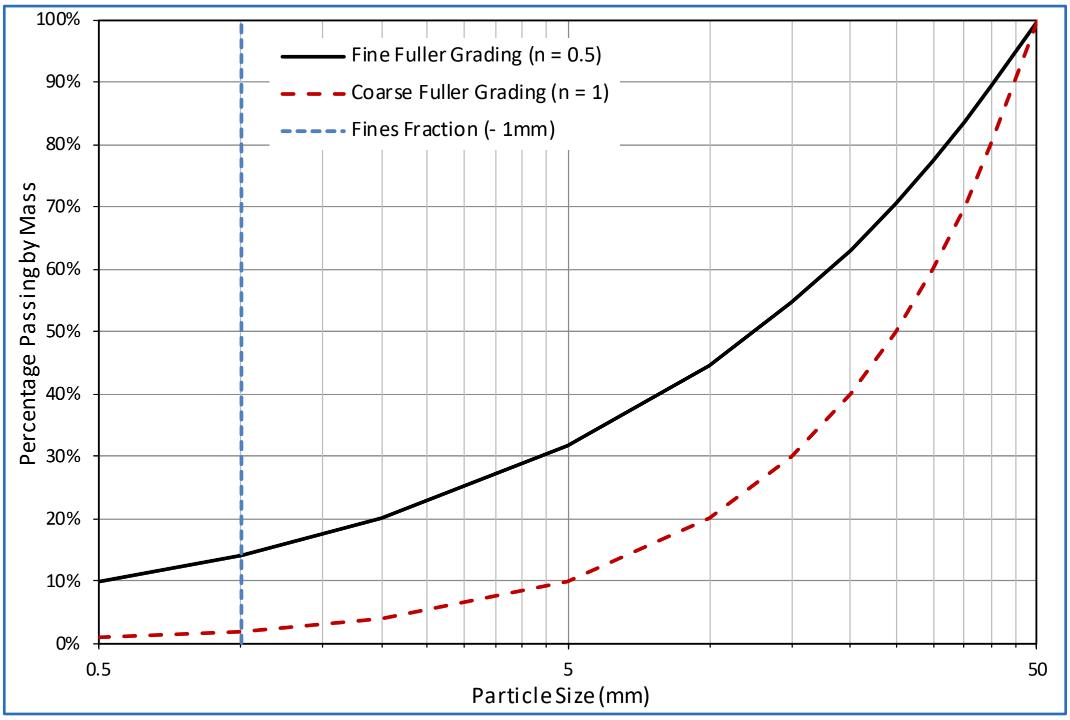
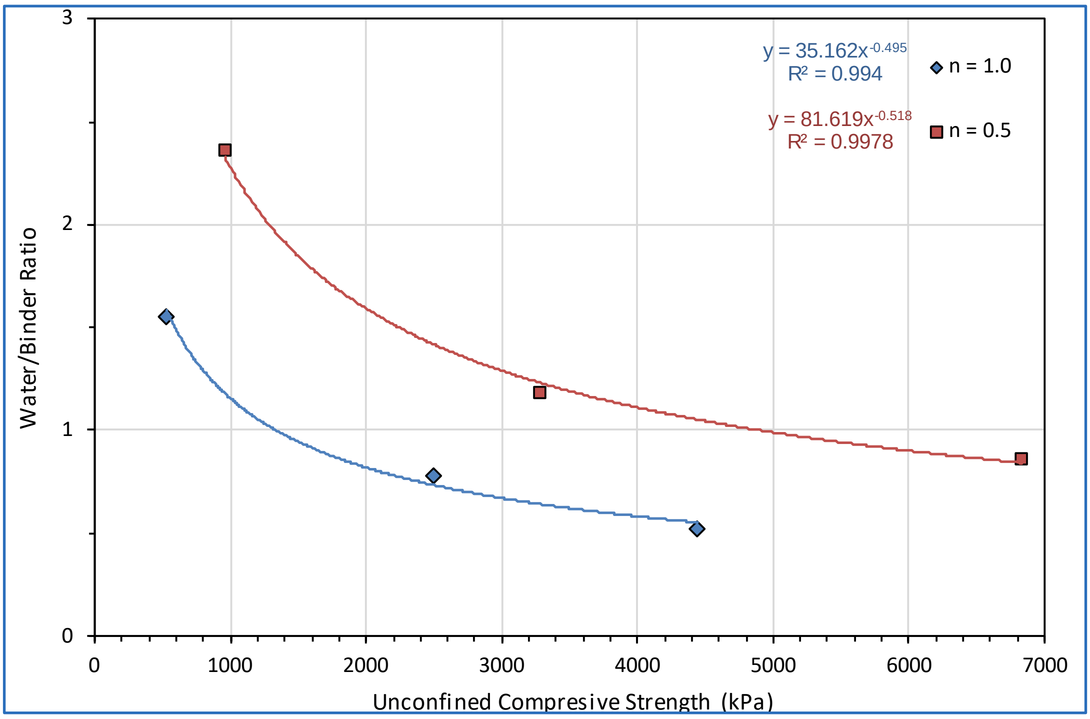


Figure 16.36 UCS Strength versus W:B Ratio Correlation from Testwork



16.2.9.3 Cemented Paste Fill (CPF) Testwork

No additional paste test work was conducted for the BFS study as the CPF will only be requires 6 years after production has started. Fresh tailings will be available to perform detailed test work before the paste plant implementation is started.

The CPF design is based on the testwork that was undertaken by Golder during the Platreef 2017 FS paste fill study. The material properties and relevant test work results utilised for the CPF design are presented in Table 16.17.

Table 16.17 CPF Material Properties

Item	Value/Description	
In-situ Densities	Ore: 3.15 t/m³ to 3.19 t/m³ Hanging Wall: 2.90 t/m³ Footwall: 3.10 t/m³	
Particle Size Distribution	d90 = 100 - 150 µm d75 = 70 - 90 µm d50 = 40 - 45 µm d25 = 15 - 20 µm % passing 20 µm = 25% - 30%	
Solids Density	Full process plant tailings stream: 3.06 t/m³	
Thickener U/F mass concentration	60 - 65%m target for filtration	
Filter cake mass concentration	83%m (achieved during test work by others)	
Binder properties	PPC SUREBUILD CEM II/B-M(V-S) 42,5 N Solids Density: 3 100 kg/m³	
Backfill mass concentration range	75.3 – 77.8 %m (150 Pa – 400 Pa Yield Stress)	
Rheology	Yield Stress Constant A:	750×10^3
	Yield Stress Exponent B:	30
	Viscosity Constant A:	10×10^3
	Viscosity Exponent B:	37
Compressive Strength Test Results	7- day strengths:	$W{:}B = 73.095 \times UCS^{-0.468}$
	28-day strengths	$W{:}B = 67.638 \times UCS^{-0.424}$

16.2.9.4 Cemented Rock Fill System

For details on the CRF system refer to Paterson & Cooke report 1051-RP-35-001 (Ivanhoe document numbers).

The CRF system comprises two distinct systems, the crushing system/circuit and the CRF batch plant, both located on surface. Figure 16.37 shows a schematic of the CRF system.

The crushing circuit is fed by the waste rock and produces the required aggregate for the batch plant. The aggregate presents the majority of the CRF mix (around 93%), and therefore the aggregate particle size distribution is the main influencing factor that determines the CRF quality and strength.

The crushing circuit comprises of the following:

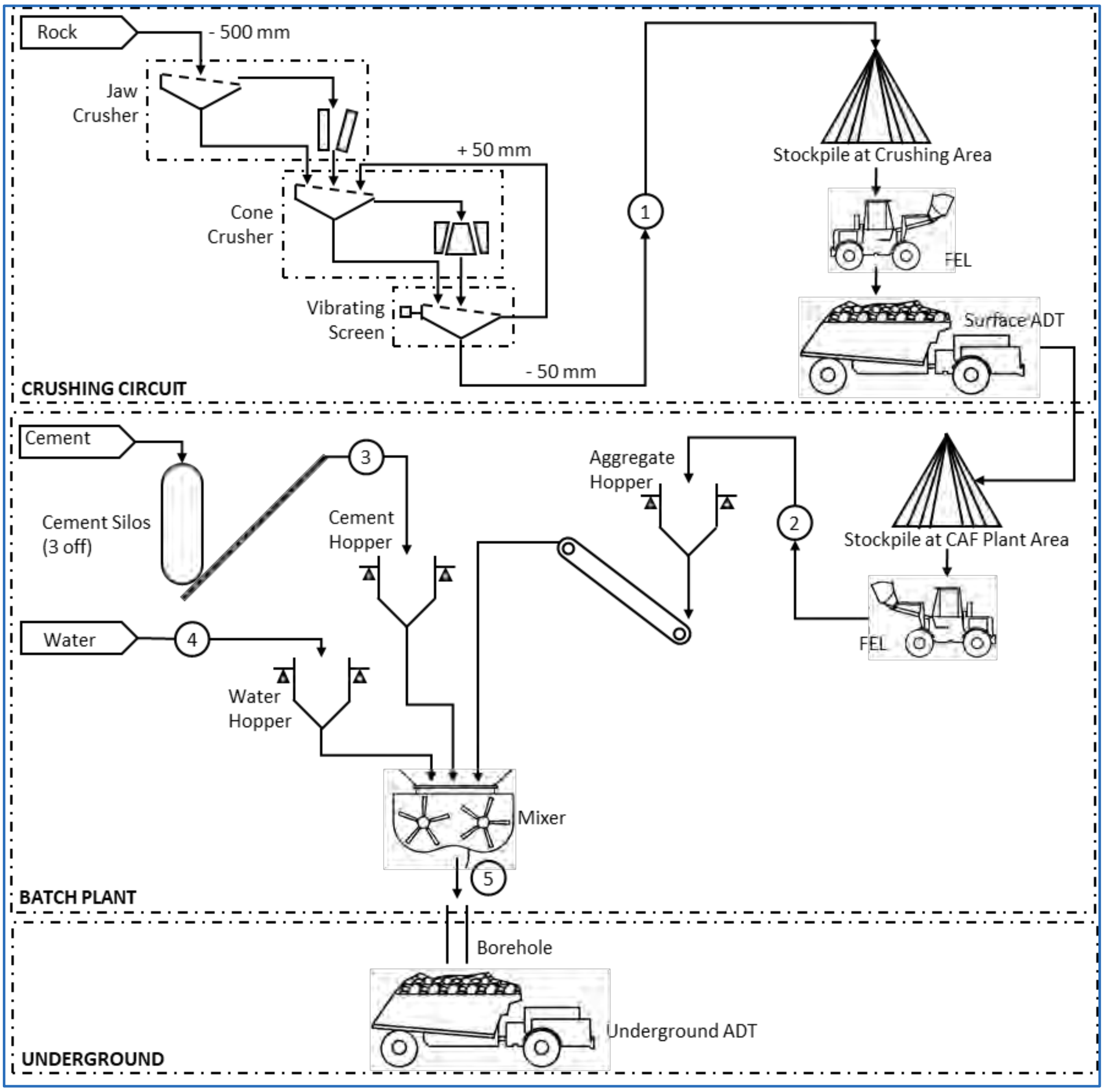
- Primary Jaw Crusher (or primary cone crusher if vendor prefers)
- Secondary Cone Crusher
- Final Sizing Vibrating Screen (multi-deck)

The crushed aggregate is then trucked to the CRF plant area to an intermediate stockpile. From there it will be loaded into the CRF plant aggregate bins with a FEL and mixed with cement and water in batches and sent underground via a CRF borehole.

The CRF plant comprises the following systems:

- Aggregate weigh system with conveyor feed.
- Water weigh hopper.
- Cement silos with screw conveyors and cement weigh hopper.
- Elevated twin shaft mixer with pressurised washing system.
- Control room and MCC.

Figure 16.37 Process Flow Schematic



The summary mass balance is represented in Table 16.18. The detailed mass balance is provided on the Process Flow Diagram in the report.

Table 16.18 CRF Mass Balance and Mix Design

No	Stream Description	Daily Throughput	Hourly Throughput	Mass per m³ CRF	Volume
1	Rock	1880 t/day	126 t/h	-	0.688 m³
2	Aggregate	1880 t/day	118 t/h	2089 kg	0.674 m³
3	Cement	38 t/day	2.4 t/h	43 kg	0.014 m³
4	Water	101 t/day	6.3 t/h	112 kg	0.112 m³
Total	CRF	2020 t/day	126 t/h	2244 kg	1.000 m³

Views of the 3D model of the CRF plant are shown in Figure 16.38 and Figure 16.39. Detail on the borehole delivery to underground can be found in the CRF system design report.

Figure 16.38 Surface CRF Plant 3D Model

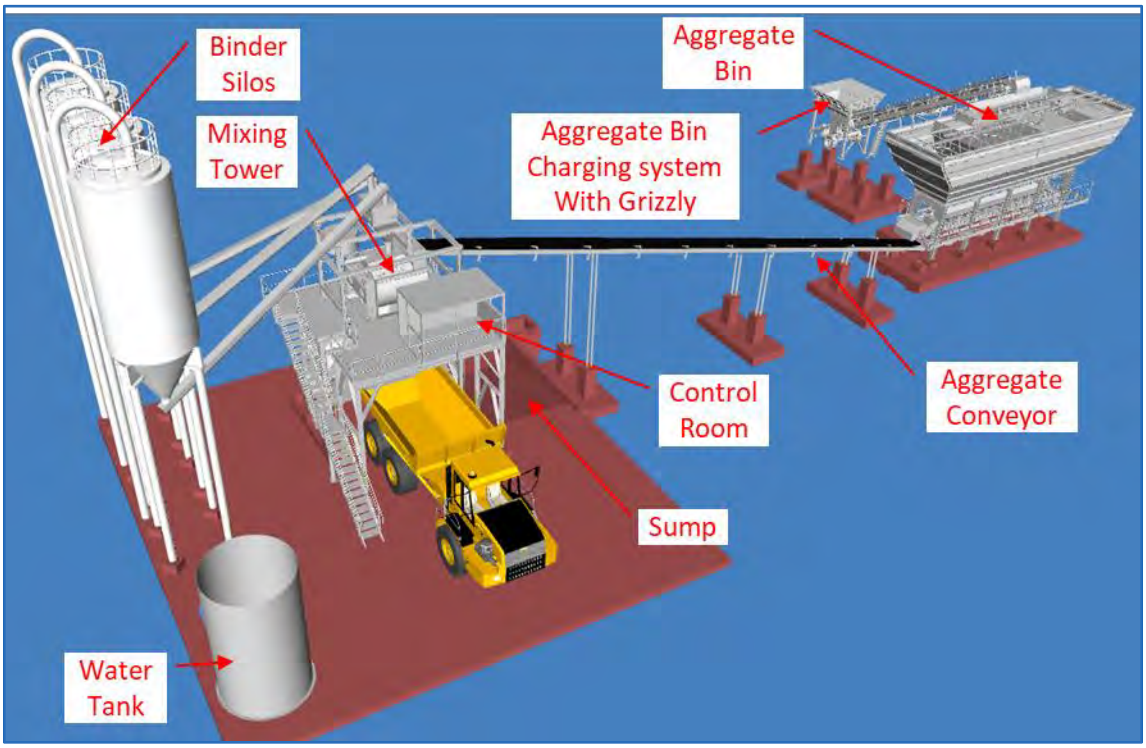
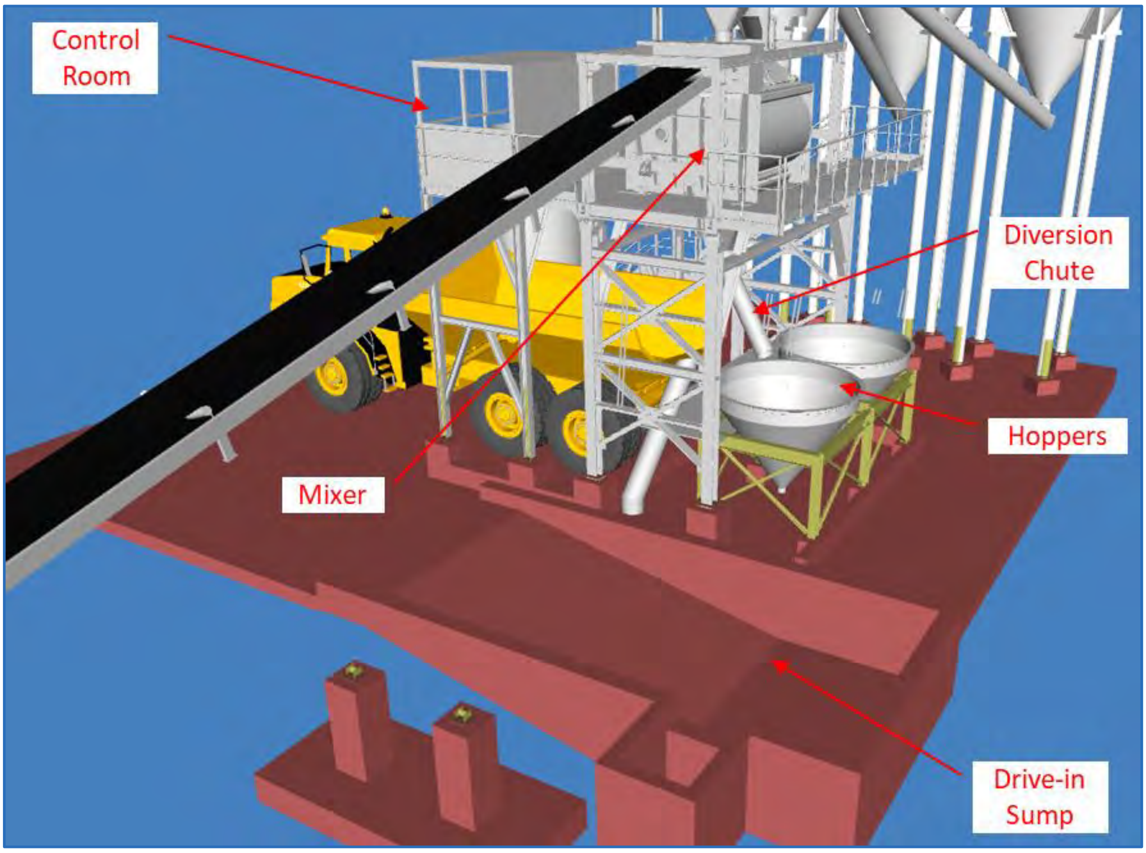


Figure 16.39 Surface CRF Plant Discharge Arrangement



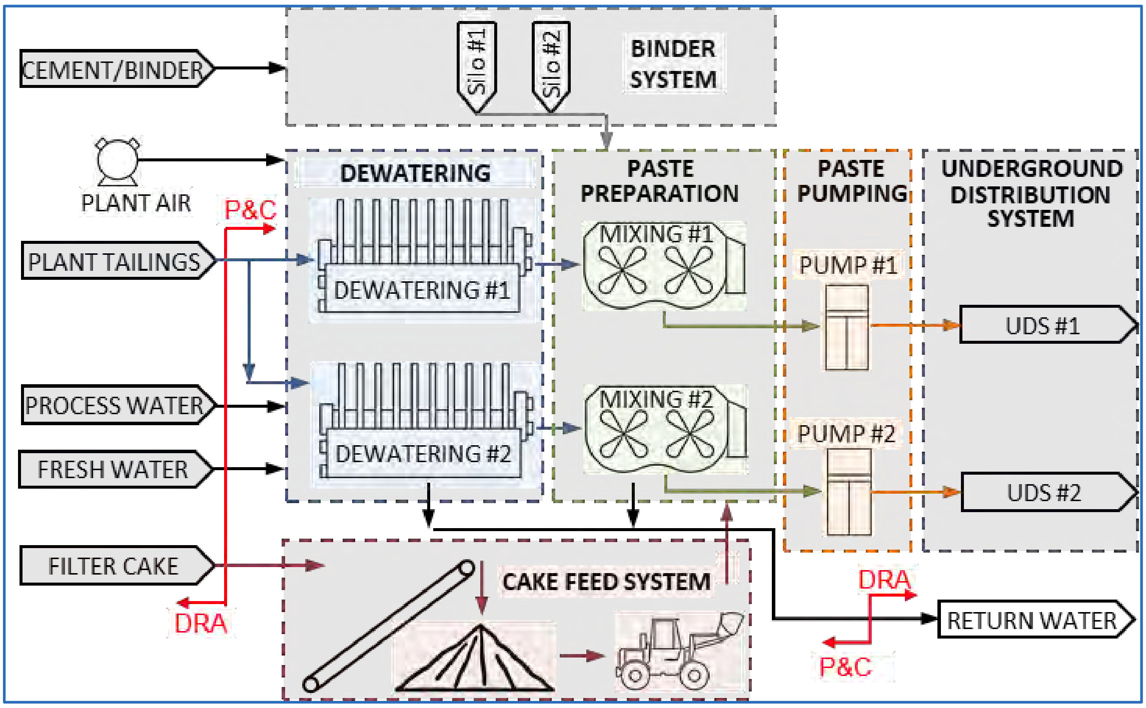
16.2.9.5 Cemented Paste Fill System

For details on the CPF system refer to Paterson & Cooke report 1051-RP-35-002 (Ivanhoe document numbers).

A block diagram for the entire system is shown in Figure 16.40. The detailed process flow diagrams and mass balance are presented the CPF System report.

The tailings feed from the process plant will be stored in filter feed tanks at the paste plant. From there the tailings are filtered. The mixer is fed by filter cake from the dewatering system or from a stockpile with front-end loader. Trim slurry, water and cement is also fed to the mixer. The continuous paste mixer discharges into the paste hopper to provide a buffer for the paste pumping system. From the hopper the PD pump will transfer the paste via surface piping and boreholes to the underground backfill distribution system.

Figure 16.40 Block Diagram of CPF System



The paste backfill will be prepared using full plant tailings and various mix designs will be required for the primary and secondary stopes.

Each stope will require a plug pour that will cure for two days before the main pour can commence. The plug pour reduces the required barricade thickness and associated cost and ensures that the subsequent pours can proceed without and danger of the barricade failing.

The strength of the main pour for the primary stope is determined by the vertical exposure of the stope. The main pour of the secondary stope requires a lower strength than the main pour in order to prevent potential liquefaction. The final capping pour for both primaries and secondaries requires a higher strength to for bogging the next lift.

The mix designs for the various pours are presented in Table 16.19 for a CPF yield stress of 300 Pa.

Table 16.19 CPF Mix Designs

Parameter	Primary Plug Pour	Primary Main Pour	Secondary Plug Pour	Secondary Main Pour	Capping Pour
Yield Stress	300				
Concentration	77%				
Strength (kPa)	150 / 335	335	150	150	500
Curing Time (days)	2 / 28	28	2	28	28
W:B Ratio	5.75	5.75	7.01	8.08	4.85
Binder Content	5.2%	5.2%	4.3%	3.7%	6.2%
Tailings (kg/m^3)	1 515	1 515	1 530	1 539	1 500
Water (kg/m^3)	477	477	477	477	477
Cement (kg/m^3)	83	83	68	59	98
Density (kg/m^3)	2,075				

For a detailed operating philosophy refer to the CPF System report.

A 3D model was developed for the surface paste plant. Figure 16.41 and Figure 16.42 show snapshots of the 3D model developed for the paste plant based of sizing of the major mechanical equipment. The 3D model allowed for extraction of detailed BOQ for costing purposes.

Figure 16.41 View for the Paste Plant from the South

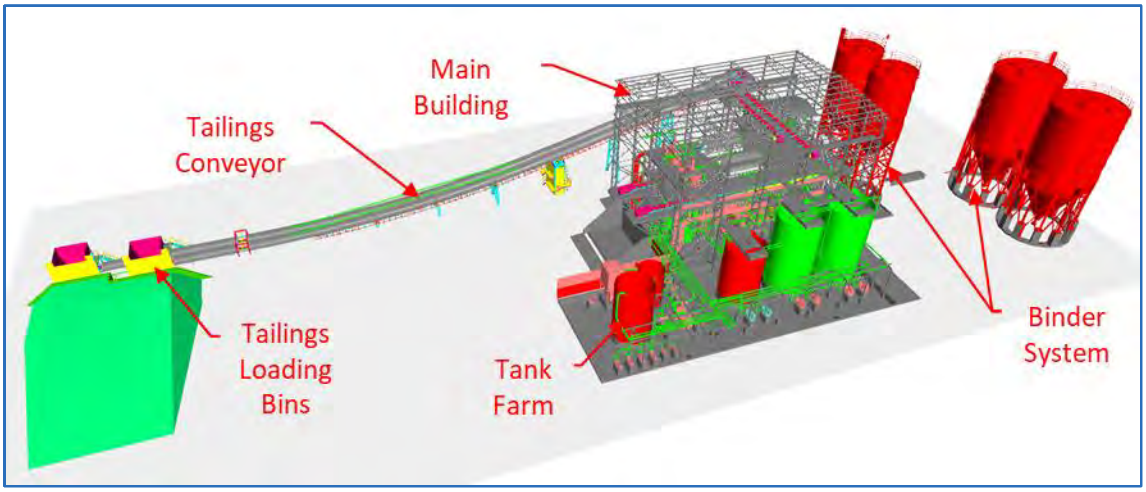
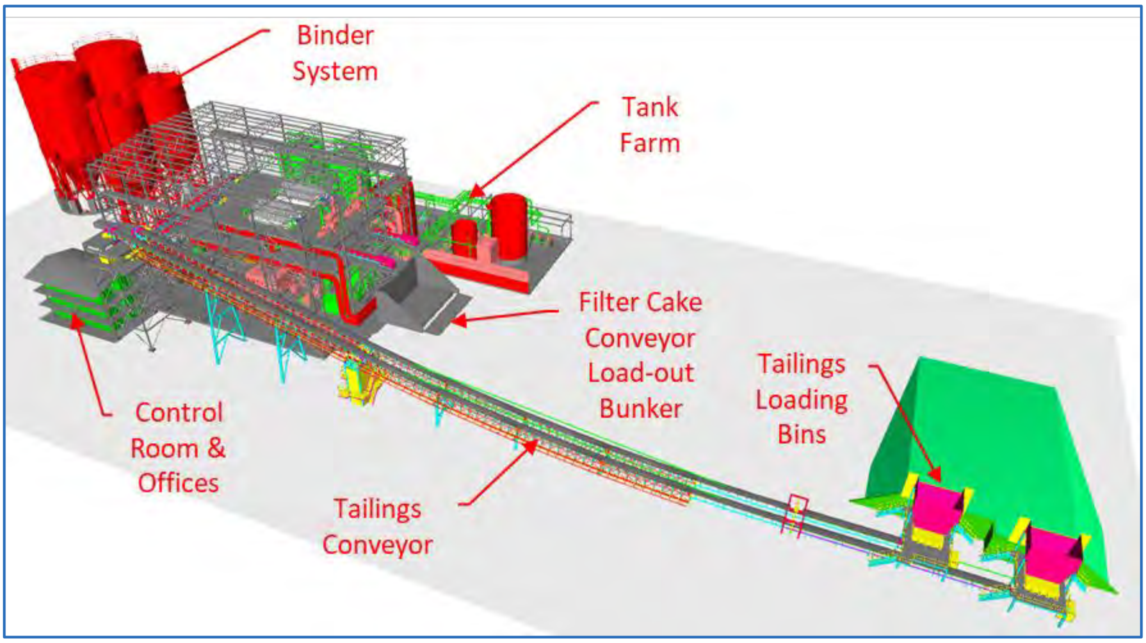


Figure 16.42 View for the Paste Plant from the North



16.2.9.6 Paste Underground Distribution System (UDS)

The cemented backfill will be transported to the stopes using hydraulic piston pumps at the CPF plant. Two high-pressure pipelines will exit the paste plant, one dedicated pipeline per module. Following a short surface section (maximum 50 m) to access the boreholes, the paste piping will drop down the paste boreholes (depth of 890) m to the underground borehole cuddy.

From the borehole cuddy the main trunk piping will run along the main access routes and tap off to the respective stopes.

Based on the layout of the Platreef UDS and system hydraulic modelling, the pipe specifications for the surface, underground and in-stope piping are detailed in Table 16.20. The pressure ratings for each pipe type are calculated according to ASME B31.3 which takes into consideration the likelihood of personnel being near the piping.

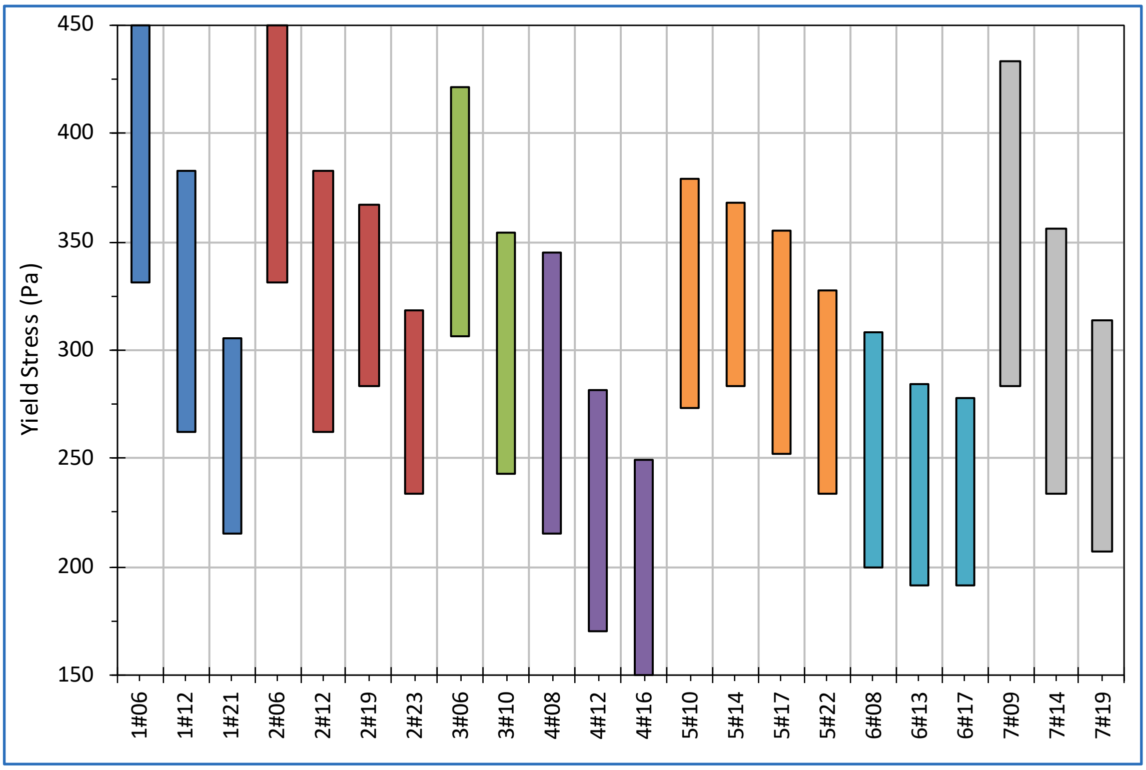
Table 16.20 Pipe Specifications

Piping	A92 (80 bar) UG only	A93 (150 bar) UG & Surface	A94 (200 bar) BH & UG	K3 (16 bar) In-Stope Piping
Pipe Description	DN 200, Sch40	DN 200, Sch80	DN 200, Sch120	DN250, SDR11
Outside Diameter	219.1 mm	219.1 mm	219.1 mm	250 mm
Wall Thickness	8.2 mm	12.7 mm	18.3 mm	20.3 mm
Wear Allowance	2.0 mm	2.0 mm	4.5 mm	2 mm
Worn Pressure Rating	8 MPa	15 MPa	20 MPa	1.4 MPa
Material	API 5L, X65, Seamless			ISO 4427, PE100
Flange Rating	Class 600 ASME	Class 900 ASME	Class 1500 ASME	SANS 1123, 16bar
Coupling Rating	25 MPa	25 MPa	25 MPa	2.0 MPa

For more details on each pipe specification and installation conditions refer to the CPF System report.

The operating ranges for the CPF system was evaluated for several discharge points representing the boundary stopes for the orebody. The operating range for each stope is shown in Figure 16.27 and is indicated by the yield stress.

Figure 16.43 CPF System UDS Operating Range



This assumes that the viscosity and yield stress correlation remain constant. The wider the yield stress operating range is, the more variation in the backfill preparation process and tailings received from the plant can be accommodated while filling.

The lower limit is determined by the minimum yield stress required to prevent slack-flow conditions while the higher limit is determined by the maximum pumping pressure. A minimum yield stress of 150 Pa is used at this stage to determine the point at which the paste starts to segregate but will need to be confirmed with test work during the next phase.

The UDS system components are described in detail in the CPF System report.

16.2.9.7 Further Backfill Work and Studies

Previous paste fill test work based on previous pilot plant tailings was available. As the paste plant will only be constructed once the process plant has been commissioned, it was determined that more accurate paste test work can be developed when plant tailings are available. Any further paste test work would have the same drawbacks as the original paste test work, mainly sample size and orebody representation.

The main items affected by the paste test is the sizing of the de-watering system components and the cement consumption required to achieve the required paste strength.

The design and use of paste backfill barricades varies between conventional shotcrete and waste rock. The latter provides significant cost savings and have been in successful operation, for example at the Zinkgruven Mine in Sweden for longhole open stope mining. Consideration for their use at the Platreef Project should be investigated to reduce operating costs. The waste rock barricades may be most applicable to the "short cycle time" Drift-and-Fill mining sequence.

The binder requirements for CRF have been estimated based on UCS results obtained during the pre-feasibility stage of study. It is recommended that UCS testing be conducted at various binder quantities to verify the assumptions.

The placement of CRF is based on 100% CRF in the primary stopes and 50% CRF and 50% Waste Rock in the secondary stopes for longhole mining. At the next stage of engineering, the amount of CRF and Waste Rock for secondary stopes should be evaluated more accurately based on the mining sequence. This presents an opportunity for Platreef to reduce the cost of backfilling.

The binder requirements for CPF have been estimated based on the UCS results measured in the Sudbury Golder's laboratory during Platreef 2017 FS. At the next stage of engineering, binder requirements for primary and secondary stopes, along with binder requirements for the different stages of pouring (plug, body, cap, etc.) should be defined more accurately for the opportunity to reduce the cost of backfill.

Although it is not believed to be currently available in the mine's immediate area, initial indications are that a binder consisting of a blend of 90% ground iron blast furnace slag and 10% normal Portland cement (90/10) could be beneficial in reducing binder consumption, and associated cost, over the life of the mine. The economic merits for the use of this binder should be evaluated in the next phase of laboratory testing. In a previous paste plant design, the availability of this binder was also thought to be non-existent until the economic merits of its supply were evaluated, and its use was implemented.

The pipeline friction losses applied in the flow model evaluation have been estimated based on the results of the lab testing programme on the Platreef tailings and Golder's database of tailings with similar properties. The conclusion was that booster stations are required to backfill the extremities of the orebody as well as Zone 3 since it is above the 750 m level. It is recommended that flow loop testing be conducted in the next phase of the Platreef project, being when the pilot plant has been commissioned in order to obtain a new tailings sample. This will allow a better quantification of the anticipated pipeline friction losses and provide a better indication of if, when, and where booster stations for paste distribution are required.

In some ore zones, long paste delivery times are observed in the flow models. It is recommended to investigate the addition of a binder hydration retardant be added to the paste product on surface to prevent hydration during transport. It is also recommended that testing be done with a retardant to determine the yield stress over time, validating the maximum time that the paste can be in transit from the plant to the stope.

In the North, part of Zone 3 is above elevation (up-grade) from the main access. Golder is recommending that a booster station be added to deliver paste to these upper stopes. However, it is recommended that Platreef examine the possibility of drilling a separate borehole from surface to the top part of Zone 3 to determine if there is an economic benefit.

In most areas of the mine, a 254 mm (10") slump is required to deliver the paste fill, along with booster stations in some of the areas. At the next stage of engineering, the location of the surface boreholes should be re-examined. Locating the boreholes in separate areas above both the North and South sections of the orebody should be considered.

The UDS is based on the installation of 200 mm (8") Sch 80 carbon steel (CS) pipe throughout the distribution system, with the exception of branch lines going into the stopes. In the next stage of engineering, the use of 200 mm (8") Sch 40 CS pipe should be considered for the installation on the levels (i.e. Level Piping). The Sch 40 pipe, due to its slightly larger internal diameter (ID), will reduce the friction losses and pressure in the system and will also present an opportunity for cost savings. This can be examined once the UDS routing is further defined.

16.2.10 Mine Equipment Requirements

16.2.10.1 Introduction

A separate study was undertaken for the Mine Mobile Quantities Estimation by OreWin. The mobile equipment for the mine design focused on primary development and production equipment. The secondary fleet to support the production and development was simulated and evaluated as a separate exercise prepared by DRA.

The equipment was selected to meet mine design and production requirements. The identification of the specific manufacture(s) and the costing of the equipment was determined through a tendering process. Equipment quantities were determined based on operating hours required and location. Swing units were then added to account for overall availability.

Primary equipment is best described as equipment that is needed to either develop drifts or Longhole Stopes. The absence of any of this equipment would stop the advancement of the mine. Secondary equipment is best described as equipment that supports the crews on primary equipment. The absence of this equipment would not stop mine advancement but would affect the performance of the primary equipment.

16.2.10.2 Mobile Equipment

The primary mobile equipment determined for the mine design covers both development and production. The secondary equipment to support these activities are covered by others.

The development equipment is based on a fleet of Jumbos, mechanised rock bolters, and explosives trucks. Also included in the development fleet are cable bolters for large spans and intersections. A complete equipment list is in Table 16.21.

The Jumbos will be two-boom units with a 6.49 m drill boom. These units will be mechanised and allow for computer automated drilling and alignment. The rock bolters will be designed to allow for the variety of bolts listed in the geotechnical portion of the mine design criteria, 2.4 m resin rebar, 3.0 m resin rebar, and other miscellaneous support types. The bolters will have carousels to load a quantity of bolts to minimise delays or worker exposure during the bolting process. The emulsion loaders are one of the units that will require operators to work most the time outside of the unit's operator's compartments. The certified explosive loaders will be required to be in a lifting basket at the work face to load the round. These lifting baskets are to be integrated into the design of the emulsion trucks.

The production equipment includes a longhole drill and an emulsion loading truck. Due to the depth of the holes, less than 20 m, a top-hammer drill is recommended. This drill will provide the required accuracy and will have a much faster penetration rate than an in-the-hole drill. The emulsion-loading trucks for the production stopes do not require lifting baskets, as the holes to be loaded are not elevated.

For each operating area, the initial equipment quantities were rounded up. The operating areas for development were by level (750 m, 850 m, 950 m) and for production were by zone (Zone 1, 2, 3, 4, and 5).

All units will have closed, air-conditioned cabs as risk mitigation for heat rest regime and to reduce work exposure to dust-borne particles while operating. Although the refrigeration design will provide for sufficient cooling for the average temperature of the hottest month, infrequent peaks may occur when the mine is operating in the warmest zones. As such, closed cabs will prevent the equipment operators for having to use a work rest regime if temperatures exceed 27.5°C WB.

Table 16.21 Mobile Equipment Types

Equipment Type	Quantity Base	Phase 1 Maximum Operating Quantity	Phase 2 Maximum Operating Quantity
Double Boom Development Rigs	All Lateral Ore (including Drift-and-Fill) and Waste Development	5	12
Longhole Production Drill Rigs	Ore Produced from Stopes	2	9
Roof Bolters	All Lateral Ore (including Drift-and-Fill) and Waste Development	5	10
Cable Bolters	Waste Development	1	1
Waste Development LHD (14T)	All Lateral Waste Development	4	8
Production LHD (17T)	All Lateral Ore Development and Ore Produced from Stopes	2	12
Mobile Rock Breaker	LHD only Hammer by others	1	2
Haulage Trucks (50T)	All Waste tonnes, CRF and Ore Tonnes	5	19
Explosives Trucks	Mine Wide	4	8
Scissor Lifts	Mine Wide	4	4
Personnel Carriers - LDV	Mine Wide	8	20
Grader	Mine Wide	-	2
Utility Equipment - UG	Mine Wide	7	8
Lube/Fuel Truck	Mine Wide	2	3
Water Spraying	Mine Wide	1	2
Scalers	Mine Wide	2	5
Multi-Purpose Vehicles (Manito)	Mine Wide	3	4
Forklifts	Mine Wide	3	3
Agicar (Concrete Mixing Truck)	Mine Wide	4	8

16.2.10.3 Fixed Equipment

A list of major fixed equipment by category for Platreef is presented in Table 16.22. This list includes fixed equipment, water handling, electrical, material handling, ventilation, and miscellaneous.

Table 16.22 Major Fixed Equipment by Category

Description	Quantity
Material Handling	
BTI MRH 16 BX30 with TB825X Hammer – Rock Breaker	9
Spillminator – Radial Gate Hydraulic Power Unit	2
Ventilation	
Spendrup 55 kW Axial Ventilation Fan	10
Howden – 1,450 kW Ventilation Fan	6
Ventilation Overhead Door Model 620	2
Pumping	
Flygt BS2201 HT	2
Flygt BS2660 HT	3
Flygt BS2075 MT	2
Flygt BS2670 HT	2
Tsurumi LH6110	9
Tsurumi LH430W	2

16.2.11 Personnel

Direct labour requirements were established to suit the selected mining method, direct support systems requirements during mine development and production. Personnel requirements are based on an operating schedule of 12 hours per shift and two shifts per day for 360 days per year. A baseline was created with all crews being owner-operator and was modified based on contractor bids and compared to the baseline. The productivities did not change when compared to the baseline. The current resourcing strategy is that initial development and production by 2025 will be performed by contractors. The contractor personnel will be transitioned into owner crews.

Training of the workforce is an extremely important requirement for the success of the project. The numbers provided for the development and production crews assume the initial contractor crews are ready and able. As the contractor crew size increases, they are providing training, so the crews are capable. The transition of this workforce to the owner provides the same trained workforce. That training process is developed to ensure that trained operators are ready to fill the positions when required.

16.2.11.1 Classification Descriptions

The following is the breakdown of labour based on equipment operation for matching skill level. The breakdowns used are presented below in Table 16.23 to Table 16.25.

Table 16.23 Development Equipment

Development Equipment	Operators per Unit
Jumbo	Drill Rig Operator
	Drill Rig Assistant
Explosives Truck	Explosives Team Operator
	Explosives Team Assistant
	Explosives Team Assistant
Bolter	Roof Bolt Operator
	Roof Bolt Assistant
Cable Bolter	Cable Bolt Operator
	Cable Bolt Assistant
LHD	LHD Operator
Truck	Truck Driver
Telehandler (nipper delivering bolts, screen, pipe, etc.)	Transport Crew
	Transport Crew
Scissor Lift (vent, pipe installation, etc.)	Transport Crew
	Transport Crew
	Transport Crew

Table 16.24 Production Equipment

Production Equipment	Operators per Equipment
Longhole Drill	Drill Rig Operator
	Drill Rig Assistant
Explosives Truck	Explosives Team Operator
	Explosives Team Assistant
	Explosives Team Assistant
LHD	LHD Operator
LHD Jammer	LHD Operator
Truck	Truck Driver
Telehandler (nipper delivering bolts, screen, pipe, etc.)	Transport Crew
	Transport Crew
Scissor Lift (vent, pipe installation, etc.)	Transport Crew
	Transport Crew
	Transport Crew

Table 16.25 Miscellaneous Equipment

Miscellaneous Support	Operators per Equipment
Fixed Rock Breaker	Rock Breaker Operator
Mobile Rock Breaker	LHD Operator
Shotcrete Jumbo	Shotcrete Jumbo Operator

No allocation to an Owner's project team is included in the mine design portion, as it is accounted for in the overall labour plan.

No allocation for a Contractor's project team is included the mine design portion, as it is accounted for in the overall labour plan.

An overall carrying complement of additional people for sick time, absenteeism, training, etc., is accounted for in other Sections of the Platreef 2022 FS. The quantities referenced above only include direct labour requirements.

16.3 Platreef 2022 FS Mining Adjustments

For the Platreef 2022 FS no change was made to the mine designs, durations of activities, development or stoping rates. The development and the production schedules have been adjusted to achieve the steady state ore production of 5.2 Mtpa.

The steps through which 5.2 Mtpa schedule has been prepared are:

- Starting point 2017 FS schedule.
- Completed tasks removed.
- Modified timing.
- 4 Mtpa schedule modified to 5.2 Mtpa.
- Key points of the Platreef 2022 FS:
- Shaft 2 sinking from start of schedule.
- Lateral development from Shaft 1 restarts 20 months after Shaft 2.
- Duration of all activities remains the same.
- Development and stoping rates remain the same.
- Development ramp up the same rate and sequence as 2017 FS.

16.4 Underground Effective Times

Underground shifts were determined to be 11.5-hour shifts, two shifts/day, seven days/week, 360 days per year; for work for underground development, construction, and production.

Fixed (Non-productive) Time

Fixed non-productive time estimates are based on prior experience with similar projects. The 11.5-hour shift is measured "collar to collar," meaning time starts when the individual enters the cage to go underground and ends when he gets off the cage on surface at the end of their shift. Fixed non-productive time includes the following:

- Pre-shift Line-up Meetings,
- Equipment Inspection,
- Lunch, Government-Mandated Breaks, and Additional Rest Periods, and
- Safety Meetings.

Table 16.26 presents the durations of fixed non-productive activities.

Table 16.26 Fixed Non-productive Time Hours

Description	Hours
Shift Change / Travel Time	1.50
Lunch Break	0.50
Safety Talks / Equipment Inspection / Reports	0.25
Subtotal Non-productive Time	2.25

Available Hours

Available time is total shift time minus fixed non-productive time. Table 16.27 summarises the available hours per shift that have been calculated for the project.

Table 16.27 Available Hours per Shift

Description	Hours
Shift Length	11.50
Subtotal Non-productive Time	2.25
Available Work Time per Shift	9.25

Efficiencies

An average of 8.13 effective hours per shift has been applied across all mining functions. Efficiency factors are applied to performance during available hours to account for unexpected underground mining delays and environmental conditions. The basis for efficiency includes, but is not limited to, the following factors:

- No Services (process water, power, compressed air, ventilation, etc.).

- Logistical Issues.
- Underground Housekeeping.
- Bathroom and Other Unscheduled Breaks.
- Delays Caused by Supervision, Engineering, Tours, and Senior Management.

The overall efficiency applied is 87.9%. Table 16.28 presents the estimated effective work time per shift.

Table 16.28 Effective Work Time

Description	Unit
Work Time per Shift	9.25 h
Efficiency Factor	88%
Effective Work Time per Shift	8.13 h

16.5 Productivity Rates

16.5.1 Waste Development

Waste development will be done through various types of ground conditions with different support requirements. The SRK recommended support requirements for waste development are as follows:

- S1A – Tunnel support for main haulages and ramps in normal conditions (shotcrete).
- S1B – Tunnel support for other haulages and ramps in normal conditions (mesh).
- S2 – Tunnel support for high stress, dynamic conditions.
- S3 – Tunnel support for extreme high stress, dynamic conditions, and rehabilitation.

The waste development support design was applied accordingly in the mine design based on the RMR value of the area in which the development will be positioned. The development advance rates are then based on the designed support type. The waste development rates are summarised in Table 16.29.

Table 16.29 Waste Development Rates Summary

5 x 5 Arched Development		Support Type			
		S1a	S1b	S2	S3
Effective Working Hours per Day	Min.	16.25	16.25	16.25	16.25
Avg. Advance per Round[1]	M	4.51	4.51	4.51	2.9
Overall Efficiency	%	100	100	100	100
Performances					

Single Heading (single end availability)	m/day	5.16	5.83	4.26	1.93
Single Heading	m/mth	155	175	128	58
Double Heading – Increase from Single Heading – 35 % (twin end availability)	m/day	6.97	7.87	5.75	2.61
Double Heading – Increase from Single Heading – 35 %	m/mth	209	236	173	78
Multiple Heading – Increase from Single Heading – 70 % (multi end availability)	m/day	8.78	9.91	7.24	3.28
Multiple Heading – Increase from Single Heading – 70 %	m/mth	263	297	217	99

[1] 150 mm Bootlegs (Sockets)

16.5.2 Ore Development

The first principles rate calculation for ore development applies to 6 m W x 5 m H arched ore access drifts. Ore development comprises loading drift development, drill drift development, and bottom drift slashing. The support requirement for all ore access development is S2 tunnel support for high stress, dynamic conditions (SRK). The ore development rates are summarised in Table 16.30.

Table 16.30 Ore Development Performance Data

6 x 5 Arched Development	Units	Quantity
Effective working hours per day	Min	16.25
Average advance per round[1]	m	4.51
Overall efficiency	%	100
Performances		
Single Heading (single end availability)	m/day	4.72
Single heading	m/mth	143
Double Heading – Increase from Single Heading – 35 % (twin end availability)	m/day	6.37
Double Heading – Increase from Single Heading – 35 %	m/mth	194
Multiple Heading – Increase from Single Heading – 70 % (multi end availability)	m/day	8.02
Multiple Heading – Increase from Single Heading – 70 %	m/mth	241

[1] 150 mm Bootlegs (Sockets)

16.5.2.1 Longhole Stoping

Longhole stoping is the predominant source of ore production for the operation. This method is used in areas where the ore thickness exceeds 15 m. The Transverse Longhole Stoping sequence is as follows:

- Drilling will be done by the production drill in one pass, starting with the drop raise and slot sequence followed by stope blast hole drilling.

- Loading and blasting will follow a set sequence that is designed to systematically increase the open area of the stope, named the slot sequence.
- Mucking will be done after every blast to load the stope area empty to create the necessary open area for the following blasts.
- The bottom drift will be slashed wider to 15 m by the ore development crew and mucked, before the slot sequence is started and will be blasted in-time to ensure that each following bench from the longhole stoping can break into the available area below.
- Cemented rock fill or paste fill will be used as post extraction support.
- Three longhole stoping production rates were calculated that will be applied in the mine design to the appropriate stope lengths:
- 15 m Long Stopes (applied to all stopes between 10–20 m long)
- 30 m Long Stopes (applied to all stopes between 21–35 m long)
- 45 m Long Stopes (applied to all stopes between 36–60 m long)

The LHS rate calculation summary is illustrated in Table 16.31.

Table 16.31 Longhole Stoping Rates

Stope Length	Cycle Components (Days)								Total Tonnes (avg.)	Performance (t/d)
	Drill	Load	Muck	Fence Constr.	Paste Plug	Fill Stope	Set	Total Days		
10–20 m	5.9	4.4	9.5	3.0	3.6	3.6	24.2	54.1	16,250	300
21–35 m	7.8	5.3	15.9	3.0	4.0	6.0	22.3	64.4	26,850	417
36–60 m	10.7	6.8	25.3	3.0	5.3	8.6	19.8	79.6	46,400	538

All rates in Table 16.31 encompass the paste fill rate listed in Table 16.32. The difference in the performance is due to the concurrent drilling activity occurring in an adjacent stope. Larger stopes require a longer drilling time therefore that delay associated to filling of the stope is reduced giving the larger stopes an overall higher performance.

Table 16.32 represents the key data used to determine the paste delay factor for each stope length in Table 16.31. Paste fill rates are not affected by the mine operating hours. The paste fill lines will operate continuously 24-hours a day with one scheduled day per month for preventative maintenance. This provides an overall utilisation of 96.4% for the system. Using the pour performance from Golder, 291 paste t/h (85% pump efficiency), the stopes will receive 6,730 paste t/d. The overall utilisation accounts for downtime due to changing pour locations, stopes not being ready to be poured, line issues, and water quantities in stopes preventing pouring. This reduction results in the stopes being filled from the plug to the top in 3.6, 6.0, or 8.6 days based on stope lengths shown in Table 16.31.

Table 16.32 Paste Fill Parameters

Stope Length	Backfill Barricade Construction Delay	Pour Hours/Day	Pour Performance (paste t/h)
10–60 m	3 Days	23.125	291

Several mining zones are established based on geological structures, dip of the ore body, and overall value, while considering the location of main infrastructure at the 750 m, 850 m, and 950m Levels. The main zones were then divided into subzones.

The number of stopes and levels per subzone were counted to calculate the number of stopes per sublevel. Based on the number of stopes per sublevel, the number of active stopes was determined based on the criteria illustrated in Table 16.33.

Table 16.33 Active Stope Criteria per Subzone

Average Stopes per Level	Maximum Active
0–7	x/4
8–15	4
16–23	6
>24	8

A production factor was assigned based on the number of sub-levels that are within a subzone based on the criteria illustrated in Table 16.34 to account for the improved efficiency of mining over multiple sublevels.

Table 16.34 Level Production Factor Criteria

Number of Levels	Production Factor
1–4	1
5–8	1.25
>9	1.5

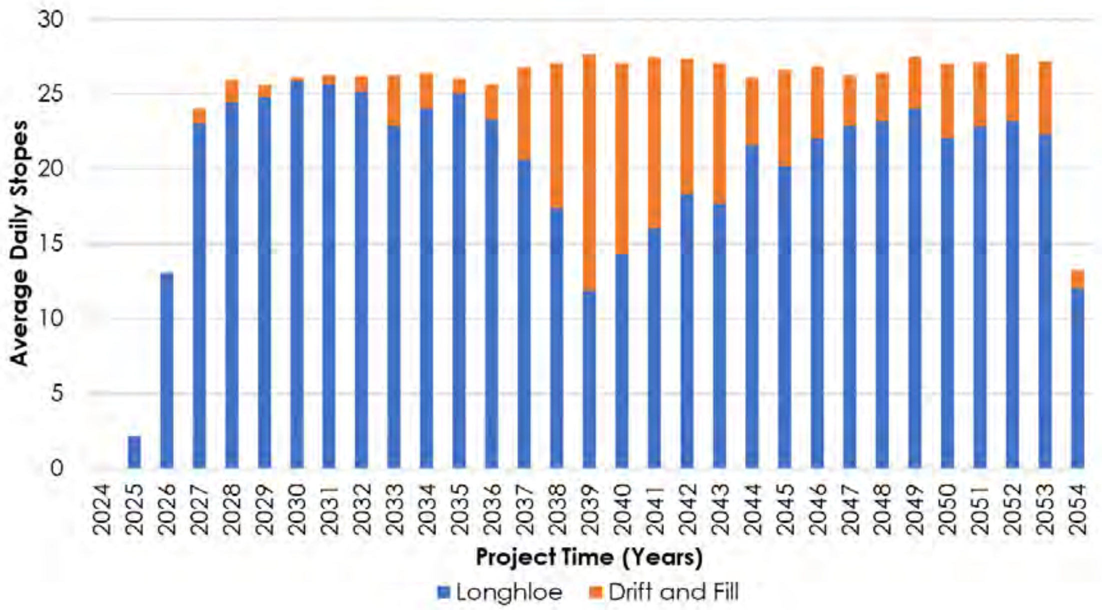
An average production rate per stope of 417 t/d was used to calculate the maximum production per zone. As a result of these factors, the maximum production rate for each zone in the Platreef Project was determined, as illustrated in Table 16.35.

Table 16.35 Production Rate Estimation per Mining Subzone

Subzone	No. of Levels	Total Stopes	Average Stopes per Level	Active Stopes per Level	Active Stopes per Zone	Number of Active Levels	Production Factor	Max. (t/d)
12	12	444	37	8	12	3	1.50	5,004
14	17	465	27	8	12	3	1.50	5,004
15	10	117	12	4	6	3	1.50	2,502
21	12	320	27	8	12	3	1.50	5,004
22	6	131	22	6	8	2	1.25	3,128
23	4	45	11	4	4	1	1.00	1,668
24	4	43	11	4	4	1	1.00	1,668
32	6	131	22	6	8	2	1.25	3,128
33	11	257	23	8	12	3	1.50	5,004
35	3	46	15	8	8	1	1.00	3,336
41	5	109	22	6	8	2	1.25	3,128
43	4	155	39	8	8	1	1.00	3,336
45	6	136	23	6	8	2	1.25	3,128
51	10	186	19	6	9	3	1.50	3,753
52	12	295	25	8	12	3	1.50	5,004
53	13	338	26	8	12	3	1.50	5,004
70	6	157	26	8	10	2	1.25	4,170
80	13	145	11	4	6	3	1.50	2,502

Platreef is a large mine with a high mining rate. Maintaining the high mining rate requires having several stopes in cycle at once. The stope cycle includes longhole drilling, loading, blasting, mucking, and backfilling; it does not include the development of the top cuts or bottom cuts. Figure 16.44 shows the average daily number of Longhole Stopes and Drift and Fill accesses in production by year.

Figure 16.44 Average Daily Stopes



OreWin, 2021

16.5.2.2 Drift-and-Fill Mining

Drift-and-Fill mining is a flexible mining method that allows near complete recovery of the ore zones. This method is development intense, resulting in lower productivity rate. This is due to the small blast size and small tonnages per cycle. The other impact is a higher mining cost per tonne, but the method allows for less dilution and more selective, higher grade mining. Good control of drilling and blasting is also necessary to minimise dilution from backfill. Mining is completed with the same equipment used for mine development.

Drift-and-Fill mining is a variation of the Cut-and-Fill mining method. In Drift-and-Fill mining, the ore zone is divided into horizontal slices, or lifts, and ore drifts are mined and backfilled adjacent to one another in a repeating fashion. Upon completion of each drift, a bulkhead is constructed, and the void is backfilled with cemented paste fill. After the backfill sets sufficiently to achieve the required strength, another drift is driven next to the fill.

For the Platreef ore body, Stantec chose two methods of Drift-and-Fill mining: transverse and chevron. The size and shape of the ore body will dictate the method used. For larger wide shapes (pancake-style shapes), the chevron method will be incorporated. For narrower stopes, the transverse method will be the applied to determine the production rates.

16.5.3 Mass Excavations

First principles rate calculations for lateral development drifting for 5 m W x 5 m H arched drifts in S1A ground were the basis for determining an aggregate development rate for mass excavations. To determine the performance rate, the number of days required to develop these excavations was determined. There are two types of mass excavations:

- Unique Excavations, and
- General Mass Excavations.

Unique excavations require slashing or additional cuts to complete the development. The list of unique excavations for the Platreef Project is listed in Table 16.36. Full-face top and bottom cuts were used in high profile or rectangular designs such the crusher rooms and crane bays. The performance of irregular shapes such as the grizzly stations and tool cribs consists of full-face advance, with slashing to the extents of design. The aggregate number of days with the aforementioned development methodologies applied against the centreline distance results in the performance rates listed in Table 16.36.

Table 16.36 Unique Mass Excavation Rates

Excavation Name	W (m)	H (m)	No. of Passes	Rate	Unit	Delay for Cable Bolts	Unit
Shop Large Profile	7.0	5.0	1	3.87	m/day	3.00	days
Shop Tool Crib	2.0	5.0	1	1.80	m/day	0.25	days
Grizzly Acc Standard	6.4	5.0	1	4.06	m/day	0.50	days
Grizzly Acc Wide	9.0	5.0	1	3.39	m/day	0.50	days
Ore Pass Temp	5.0	7.3	2	2.29 (shaft crew)	m/day	0.50	days
Transverse Stope Drift	6.0	5.0	1	4.72	m/day	–	–
Crane Bay Large	7.6	8.8	4	1.43	m/day	0.05	days/m
Crane Bay Medium	7.6	7.0	4	1.60	m/day	0.05	days/m

Schedule durations assigned to individual heading components were derived by dividing length or volume by the unit advance rate.

16.5.4 Mine Development Plan and Scheduling

Mine development has been broken down into four main stages. Stage 1 involves lateral development off Shaft 1 during shaft sinking. Stage 2 and 3 involve lateral development after Shaft 1 sinking is complete and prior to Shaft 2 commissioning. In Stage 2, lateral development is limited until Ventilation Raise 1 is completed; therefore, the priority is to commission Ventilation Raise 1 so that ventilation can be increased, and additional development crews can be added. In Stage 3, the development rate off Shaft 1 is increased after Ventilation Raise 1 is commissioned. Phase 3 is no longer limited by ventilation but is limited by the 2,500 t/d hoisting capacity of Shaft 1. Stage 4 occurs in Q3'27 when Shaft 2 is commissioned, and hoisting is no longer a bottleneck.

16.5.4.1 Preproduction Development Schedule

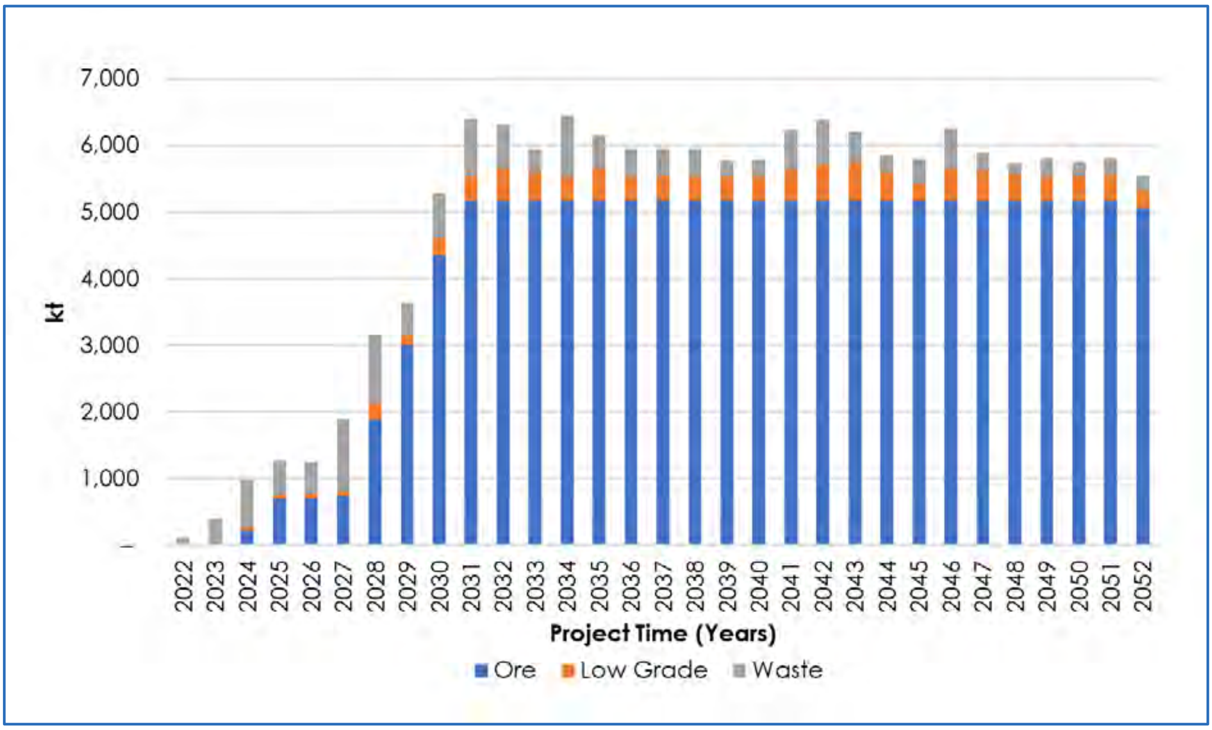
The initial refinement of the mining schedule focused on the period up to the end of the preproduction period. This period is restricted in advance rates due to the following:

- Minimal equipment availability since there will be limited development off Shaft 1.
- Limited ventilation in the overall mine design. Advance rates increase when Ventilation Raise 1 comes into operation as more mining crews can be added.

This stage of development focuses on getting the infrastructure required for production ramp-up in place and commissioned. Development would also be required to reach the ore zones that will be mined during the initial ramp-up and the initial full production years. Noncritical developments were reviewed and delayed ensuring that mainly critical development was completed due to the limited hoisting capacity.

Figure 16.45 shows the annual rock production for the LOM.

Figure 16.45 Annual Rock Production



OreWin, 2021

16.5.4.2 Life-of-Mine Development Schedule

Once the preproduction development was defined, the overall LOM development was scheduled, (see Table 16.37 and Table 16.38). At that point, the remaining LOM development was analysed for advance priorities, production requirements, total waste production, access to other infrastructure (such as ventilation raises), and an overview of access to ore zones. The schedule was optimised based on the parameters listed above.

During mine development and production, a total of approximately 195,000 m of waste development, or 15 Mt of waste material, is mined. The overall waste development is summarised in Table 16.39.

Table 16.37 Life-of-Mine Development Schedule to 2035

Description	Units	Total	Up to 2022	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Shafts																	
Shaft Development	m	2,096	996	–	16	100	733	252	–	–	–	–	–	–	–	–	–
	kt	408	134	–	4	25	182	63	–	–	–	–	–	–	–	–	–
Vertical Development																	
Vertical Development	m	10,263	–	194	879	1,051	333	21	1,643	1,415	199	671	460	189	121	738	168
	kt	432	–	3	72	48	11	0	124	73	4	14	10	4	3	16	3
Lateral Waste Development																	
Lateral Development	m	182,963	268	1,242	3,669	7,962	4,155	4,990	12,126	12,687	6,463	8,421	11,025	8,614	4,535	11,981	6,476
	kt	14,109	44	103	307	638	313	406	962	963	491	642	842	654	344	913	493
Waste Development																	
Waste Development	m	195,089	1,264	1,437	4,564	8,998	5,217	5,241	13,749	14,102	6,662	9,092	11,444	8,787	4,656	12,719	6,644
	kt	14,944	178	106	383	709	506	469	1,086	1,036	495	657	851	657	346	929	496
Low Grade Development																	
Low Grade Development	m	120,177	–	–	90	693	795	914	875	3,100	1,820	3,342	4,672	5,931	5,216	4,436	5,949
	kt	9,956	–	–	7	58	63	76	71	242	146	267	385	491	432	352	488
Waste and Low-Grade Development																	
Waste and Low Grade	m	315,266	1,264	1,437	4,655	9,691	6,012	6,156	14,625	17,202	8,482	12,434	16,116	14,718	9,873	17,155	12,593
	kt	24,900	178	106	390	767	570	545	1,156	1,279	641	924	1,235	1,148	778	1,281	984
Ore Development																	
Ore Development	m	131,774	–	–	25	996	1,285	758	1,802	5,547	6,723	7,362	7,564	6,254	7,455	8,164	5,690
	kt	10,976	–	–	2	74	100	60	147	445	559	618	631	527	622	677	471
Total Development																	
Total Development	m	447,041	1,264	1,437	4,680	10,687	7,297	6,914	16,426	22,749	15,205	19,796	23,680	20,972	17,328	25,319	18,284
	kt	35,876	178	106	392	841	670	605	1,304	1,724	1,200	1,541	1,866	1,675	1,400	1,958	1,455

Table 16.38 Life-of-Mine Development Schedule to 2052

Description	Units	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052
Shafts																		
Shaft Development	m	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
	kt	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Vertical Development																		
Vertical Development	m	122	52	239	–	–	415	639	50	174	–	208	123	–	64	–	95	–
	kt	2	1	5	–	–	9	14	1	4	–	5	2	–	1	–	2	–
Lateral Waste Development																		
Lateral Development	m	5,437	5,244	5,481	2,908	3,162	7,673	8,635	6,001	3,330	4,743	7,820	3,635	2,200	3,685	2,598	3,171	2,624
	kt	414	398	422	221	240	585	657	460	253	365	596	279	170	282	198	244	210
Waste Development																		
Waste Development	m	5,559	5,296	5,720	2,908	3,162	8,087	9,274	6,051	3,503	4,743	8,013	3,758	2,200	3,750	2,598	3,266	2,624
	kt	416	399	427	221	240	594	671	461	257	365	600	282	170	283	198	246	210
Low Grade Development																		
Low Grade Development	m	4,412	4,709	4,373	4,691	4,478	5,555	6,352	6,900	5,074	3,161	5,821	5,256	4,712	4,276	4,578	4,595	3,397
	kt	363	387	353	383	372	467	534	579	425	264	485	441	401	360	386	391	289
Waste and Low-Grade Development																		
Waste and Low Grade	m	9,972	10,005	10,093	7,600	7,641	13,643	15,627	12,951	8,578	7,903	13,834	9,014	6,912	8,026	7,177	7,861	6,021
	kt	780	786	780	604	611	1,061	1,205	1,040	682	629	1,084	722	570	643	584	637	499
Ore Development																		
Ore Development	m	7,175	6,148	5,040	3,571	2,908	2,526	2,634	3,215	5,130	4,881	4,855	5,371	3,623	4,428	3,962	3,725	2,957
	kt	601	520	422	299	241	204	218	264	424	408	410	457	308	376	335	312	242
Total Development																		
Total Development	m	17,147	16,154	15,133	11,171	10,548	16,169	18,261	16,166	13,708	12,784	18,689	14,385	10,535	12,454	11,139	11,585	8,979
	kt	1,381	1,306	1,202	903	852	1,265	1,423	1,304	1,106	1,037	1,494	1,180	878	1,019	919	949	742

Table 16.39 Life-of-Mine Waste Development Summary

Description	Units	Total
Shafts		
Advance	km	2
Waste	kt	408
Lateral Development		
Advance	km	183
Waste	kt	14,109
Vertical Development		
Advance	km	10
Waste	kt	432
Low Grade Development		
Low Grade Advance	km	120
Low Grade Tonnage	kt	9,956
Total Waste Development (Waste+LG)		
Total Advance	km	315
Total Waste	kt	24,900

16.5.4.3 Low-Grade Development

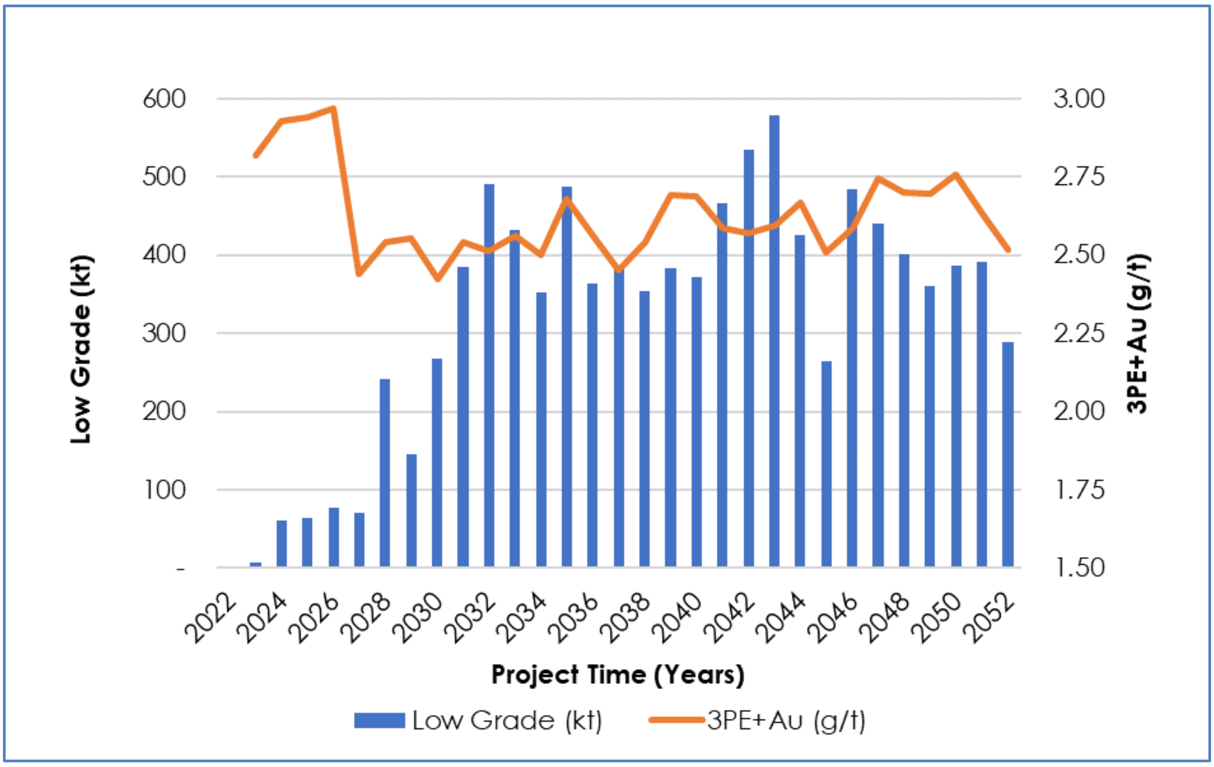
Low-grade development has been separated in the schedule and is defined as any development with 3PE+Au greater than 1.8 g/t but less than 4.0 g/t in Phase 1 (700 ktpa) and 3.2 g/t in Phase 2. This material will be identified and stockpiled separately on surface. For the Platreef 2022 FS, low grade development is not considered reserve and has no economic value. However, during the life of the mine, metal prices may result in NSR values providing an opportunity to process this material.

For this to occur, it is necessary to separate the material after development. With the large number of active headings underground, separate low-grade storages will be available during the development cycle. Once the material is stored underground, batch hoisting of the low-grade stockpile will be required.

With the excess capacity in the ore handling system and the availability of the waste pass as a backup, this separation is possible. With a maximum production of 1,250 t/d of low-grade development, only one hour per day of hoisting is required. See Figure 16.46 for the annual low-grade development tonnages. Batch hoisting of the low-grade material is therefore possible once or twice a week. This batching could occur and still allow the required time for ore hoisting, since Shaft 2 has a capacity of over 6.19 Mtpa.

While low-grade material is being batched through one line of the ore handling system, ore movement can continue in the other line through the pass, crusher, and into the fine ore bin. Then, once the low-grade hoisting is completed, the system should have a significant amount of ore in the other pass and bin to resume ore hoisting.

Figure 16.46 Annual Low-Grade Production with 3PE+Au



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16.5.5 Production Planning Criteria

Although the resource has approximately 330 Mt above an $80/t NSR cut-off, the reserve focused on maximising the grade profile for 125 Mt in the life of mine. As such, higher NSR cut-offs were used to develop the reserve statement. This strategy provides opportunity for either a longer mine life or ramping up to higher production rates to utilise more of the resource.

The LOM production plan focuses on maximising higher-grade areas. The ore body was targeted to get approximately 130 Mt at the highest NSR. This resulted in using declining cut-off grades, decreasing the NSR from $155/t to $100/t.

A further focus on optimising NPV targeted the higher-grade stopes in the LOM plan for mining in the early years. An optimisation was performed based on stope locations, stope grades, mining method, and subzone productivities. Table 16.40 shows production, NSR and grade in the first 10 years and in the remaining LOM.

Table 16.40 3PE+Au in Early Production

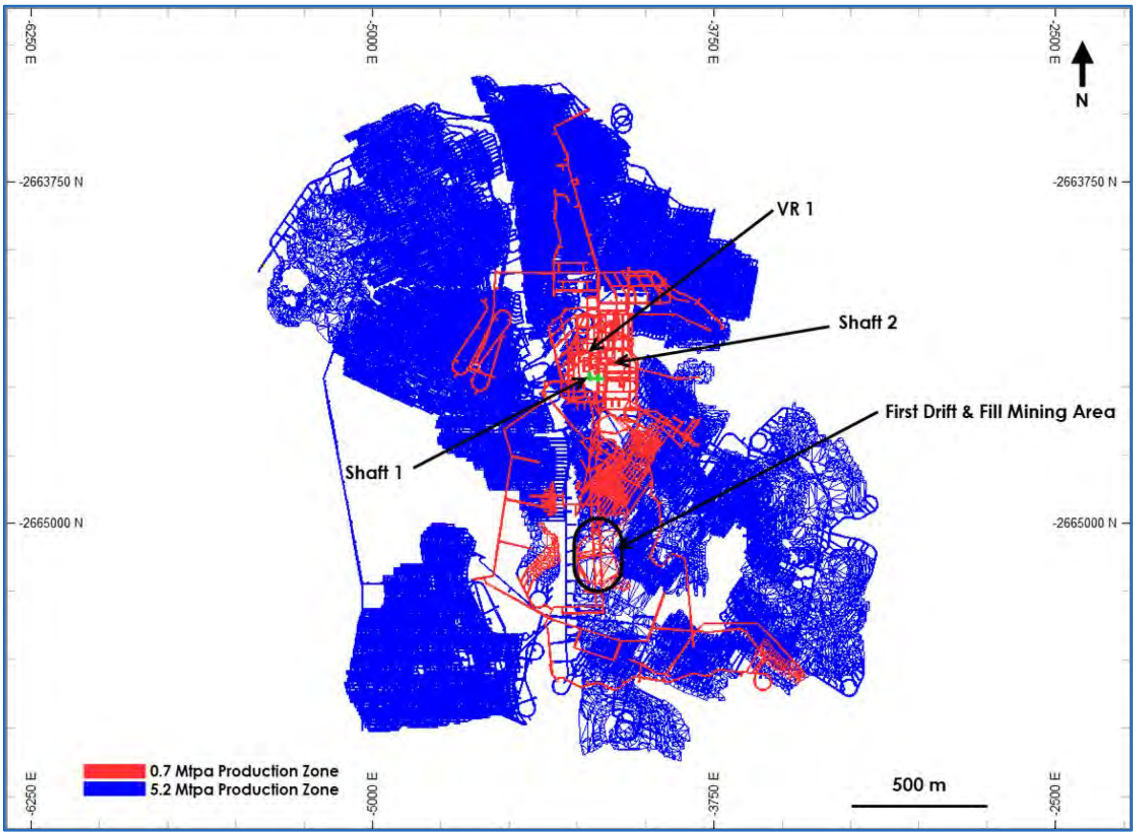
Item	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034–2052
Ore (kt)	209	700	700	735	1,885	3,000	4,356	5,170	5,170	5,170	98,114
NSR ($/t)	211	199	215	189	178	181	180	169	160	157	152
3PE+Au (g/t)	5.87	5.57	6.06	5.16	5.00	5.25	5.19	4.77	4.51	4.44	4.22

Four subzones (41, 46, 51, and 90) were identified and prioritised for initial and early development. The following additional criteria were applied over the mine life:

- Proximity to shafts and early development.
- High grade (3PE+Au greater than 4.5 g/t).
- Highly productive.
- Shaft pillars are not extracted until the end of the mine life.
- Low operating cost.

Development and production timeline schematic of Platreef 2022 FS is shown in Figure 16.47.

Figure 16.47 Platreef 2022 FS Phased Development Zones



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16.5.5.1 Production Ramp-Up Schedule

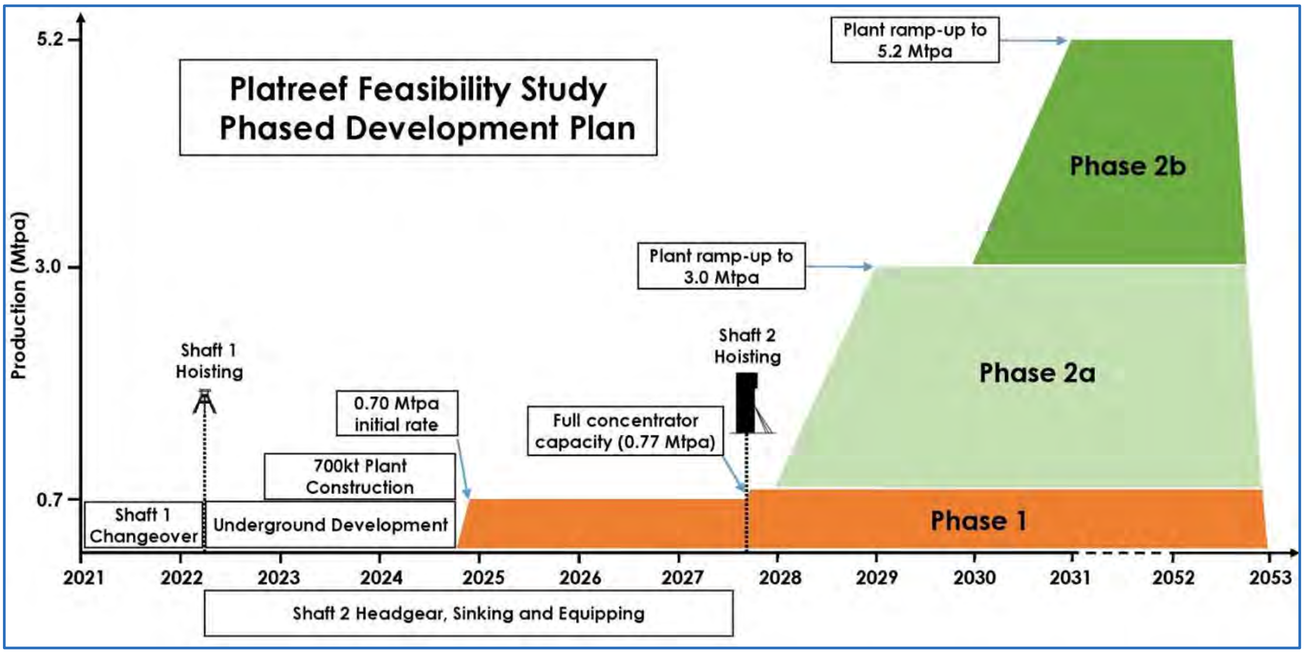
The Platreef 2022 FS begins with mining in high grade profile zones. The production starts with a 700 ktpa rate in September 2024 until 2027 when Shaft 2 commences and mining ramps up for the first 2.2 Mtpa concentrator to reach a steady state production of 3 Mtpa in 2029. Then, with the start of the second 2.2 Mtpa concentrator, mining ramps up to reach the steady state production of 5.2 Mtpa to the end of LOM. The key dates for the Platreef 2022 FS are summarised in Table 16.41.

Table 16.41 Platreef 2022 FS Key Dates

Activity Name	Start	Finish
Shaft 1 commissioning		Q1'22
Restart Development from Shaft 1	Q2'22	
Ventilation Raise 1 (750 m Level to Surface)	Q1'23	Q4'23
Ventilation Raise 1 (950 m Level to 750 m Level)	Q4'23	Q1'24
Shaft 2 Sinking to –60 m Level	Q3'23	Q4'24
First Concentrator	Q3'24	
Shaft 2 Sinking to –114 m Level	Q4'24	Q4'24
Shaft 2 Sinking to –750 m Level	Q4'24	Q4'25
Shaft 2 Sinking to –850 m Level	Q4'25	Q1'26
Shaft 2 Sinking to –950 m Level	Q1'26	Q1'26
Shaft 2 Sinking to –1,050 m Level	Q1'26	Q2'26
Shaft 2 Sinking to –1,100 m Level	Q2'26	Q3'26
Shaft 2 Equipping Complete		Q3'27
Start of mining ramp up for first 2.2 Mtpa concentrator	Q1'28	
Start of mining ramp up for second 2.2 Mtpa concentrator	Q1'30	
Mine Production Steady State (5.2 Mtpa)	Q4'30	

Figure 16.48 shows the production ramp-up and key milestones for the Platreef 2022 FS.

Figure 16.48 Platreef 2022 FS Phased Development



16.5.5.2 Life-of-Mine Production Schedule

The first 10 years of production was analysed to maximise the ore grade profile. During this period, constraints such as stope availability, ventilation requirements, and hoisting capacity were taken into consideration. Figure 16.49 and Figure 16.50 show the NSR and 3PE+Au grade ranges of the stopes. These values were taken into consideration when developing the LOM production schedule.

Figure 16.49 NSR Values by Stope

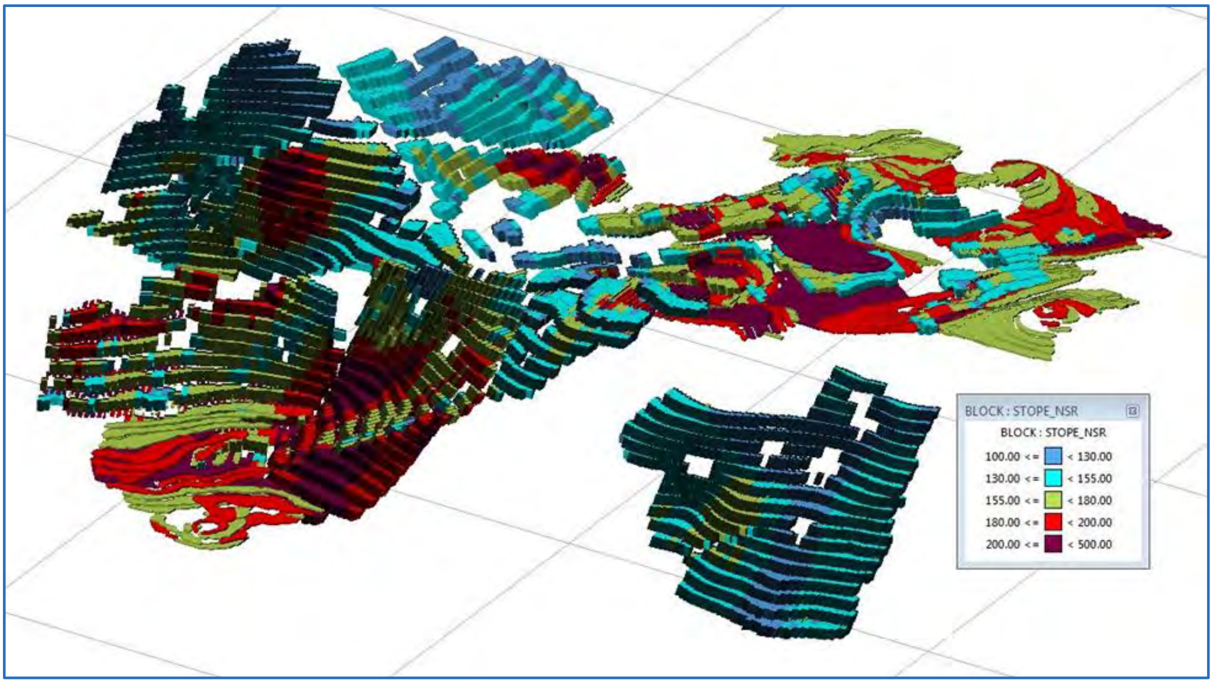
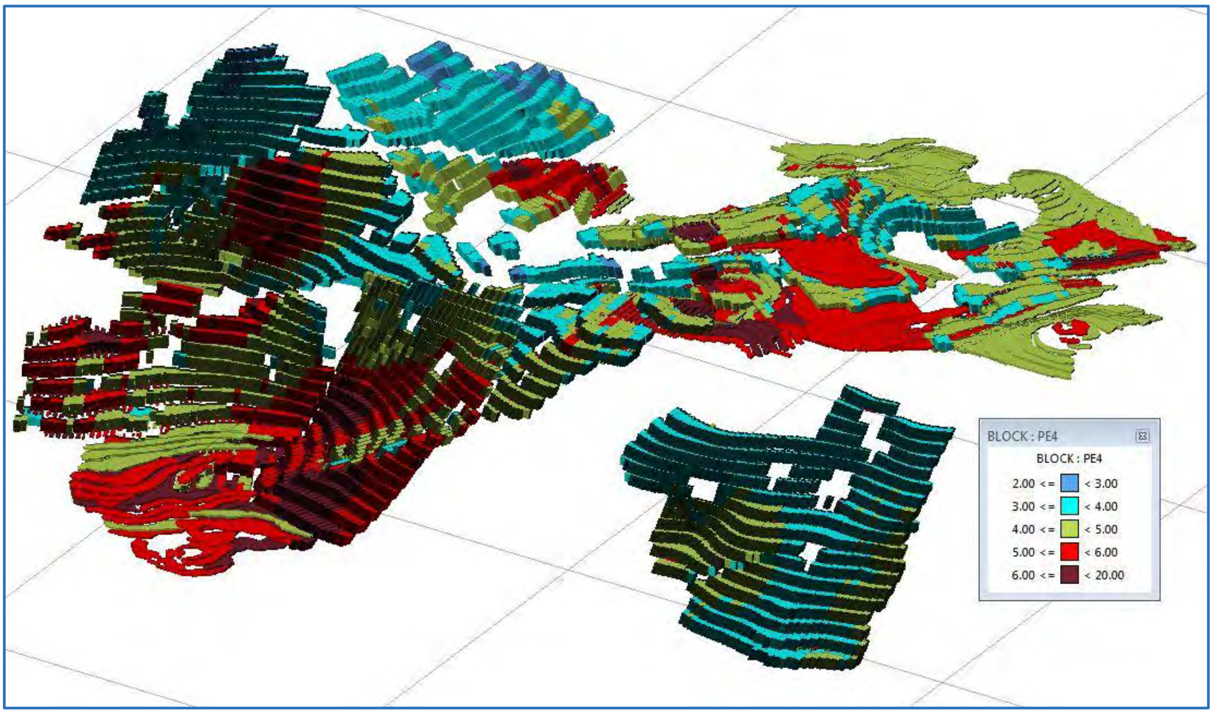


Figure 16.50 3PE+Au Value by Mining Shapes



Production Summary

Production is summarised in the following table and six figures. Table 16.42 shows the overall LOM production summary. Figure 16.51 and Figure 16.52 show the annual production rate along with the associated NSR based on BDT20 and 3PE+Au, respectively.

Figure 16.53, Figure 16.54, Table 16.43 and Table 16.44 show the annual production rate by mining method and the total production along with the associated NSR based on BDT20 and 3PE+Au, respectively. Year -2 is 2022. Figure 16.55 and Figure 16.56 show the annual production rate by rock type along with the associated NSR and 3PE+Au, respectively. When referencing the rock type, some of the mined material is backfill (most which is CPF over the LOM), which is not included in the block model. Therefore, this material is referenced as "Paste and Other Tonnes" in Figure 16.55 and Figure 16.56.

Table 16.42 Life-of-Mine-Production Summary

Description	Units	Total
Drift-and-Fill	kt	20,317
Longhole Stopes	kt	93,920
Ore Development	kt	10,976
Total Ore	kt	125,212
Diluted Grades	–	–
NSR	$/t	156
Cu	%	0.16
Ni	%	0.34
Pt	g/t	1.94
Pd	g/t	1.99
Au	g/t	0.30
Rh	g/t	0.13
S	%	0.82
3PE+Au	g/t	4.37
Max Daily Production Rate	t/d	14,164
Low-Grade Stockpile	kt	9,961
Low Grade Stockpile NSR	$/t	95
Waste (excl Low Grade)	kt	14,766

Figure 16.51 Annual Production with NSR

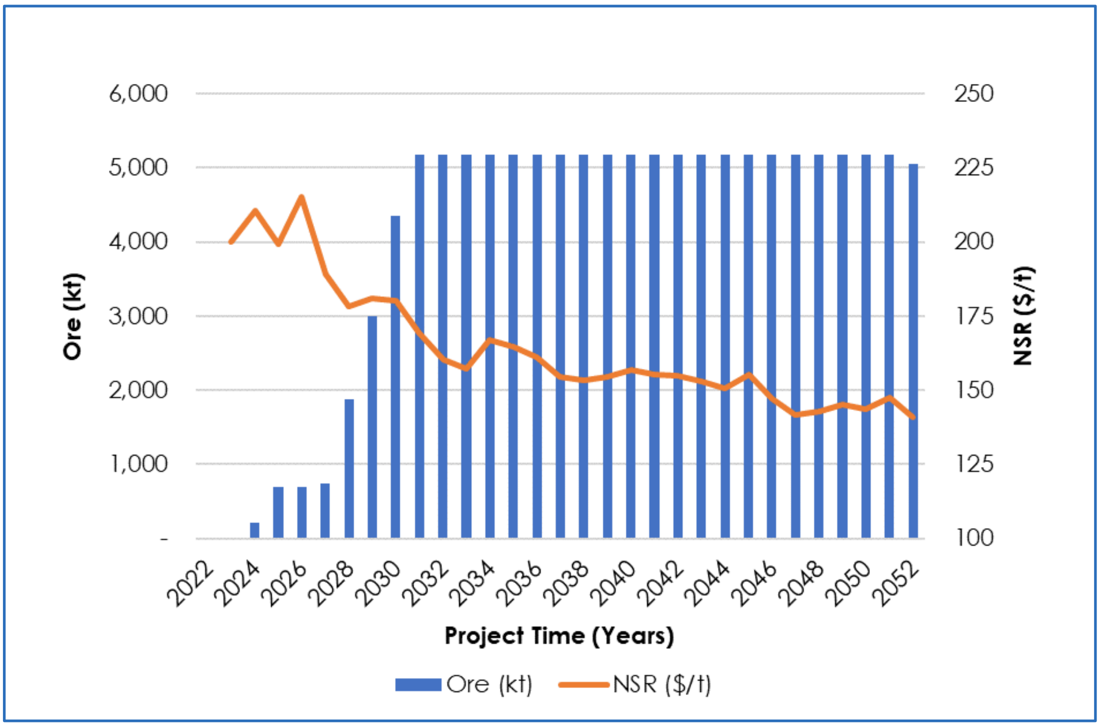


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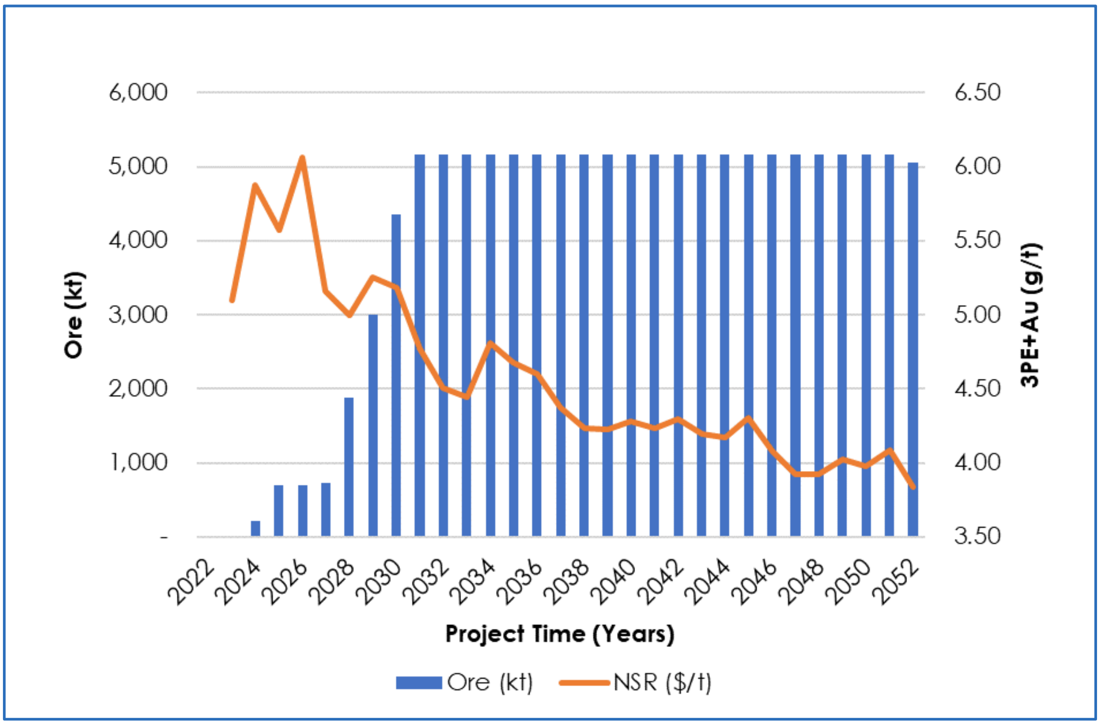
Figure 16.52 Annual Production with 3PE+Au



OreWin, 2021

Figure 16.53 Annual Production by Mining Method with NSR

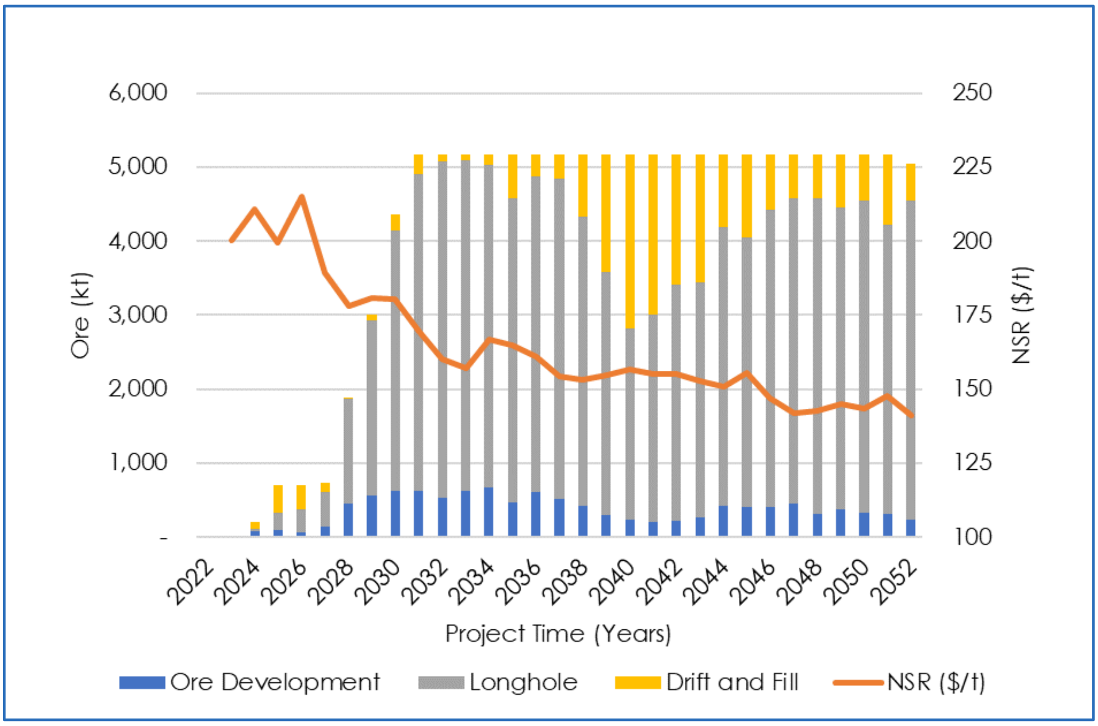


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Figure 16.54 Annual Production by Mining Method with 3PE+Au

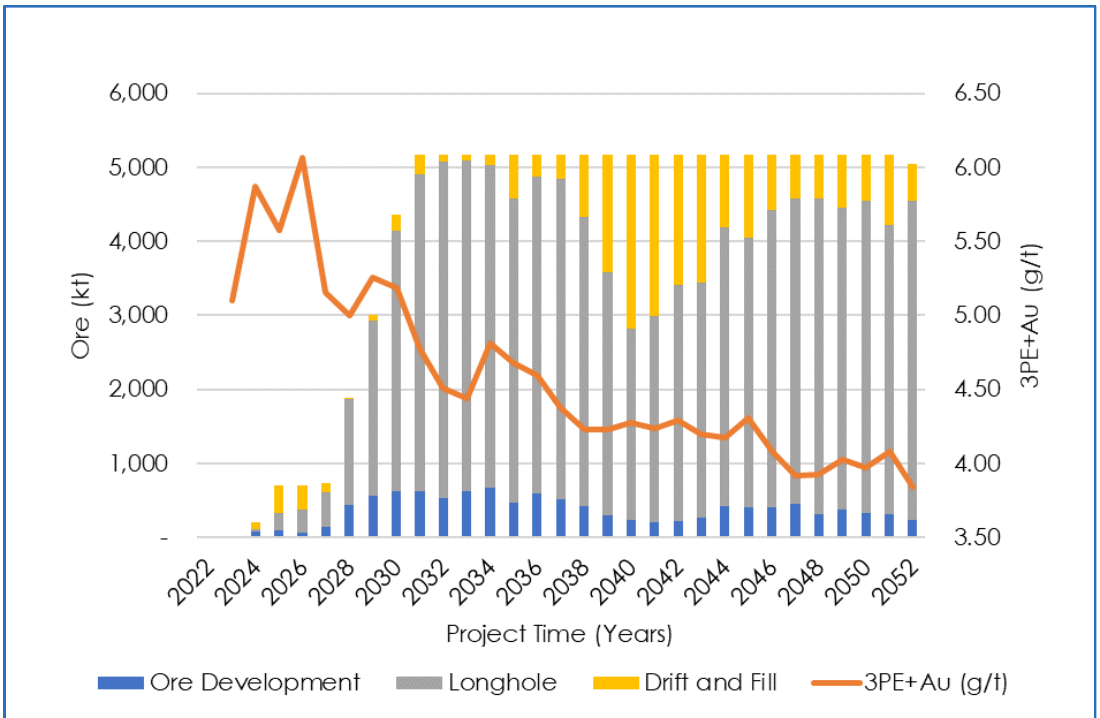

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Figure 16.55 Annual Production by Rock Type with NSR

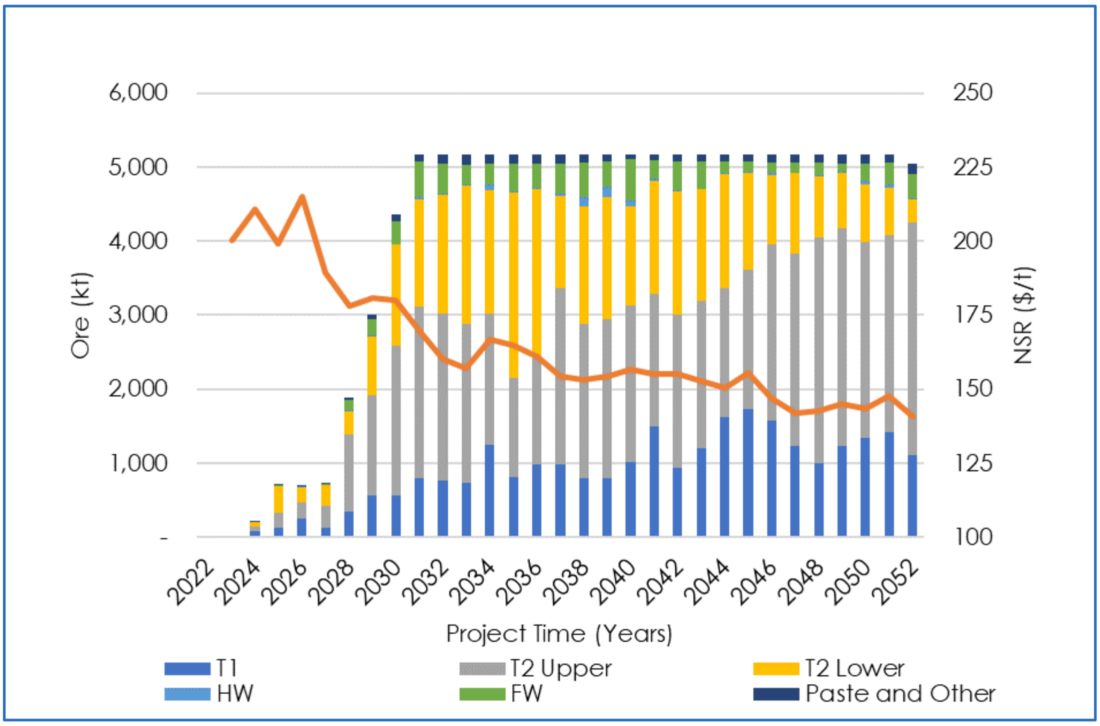

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Figure 16.56 Annual Production by Rock Type with 3PE+Au

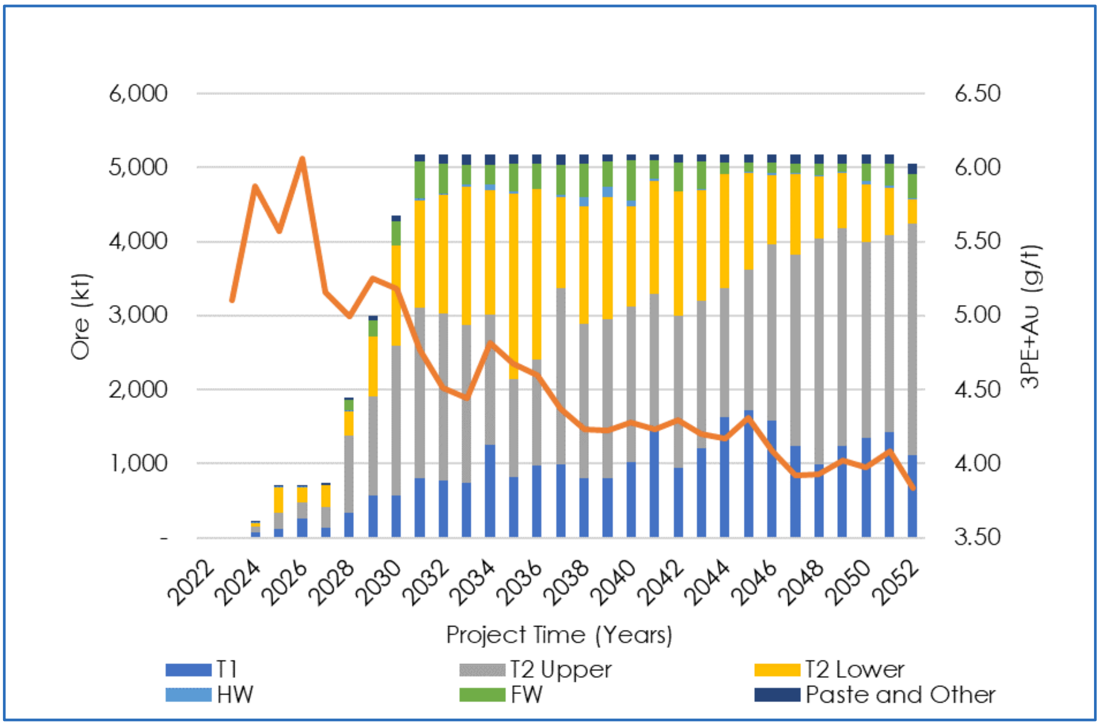

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Table 16.43 Mine Production by Method

	Year Number	Total	-2	-1	1	2	3	4	5	6	11	21
	Year To									10	20	29
Ore Development	kt	10,976	–	2	74	100	60	147	445	2,956	3,918	3,272
NSR	$/t	142	–	200	212	182	178	162	156	149	140	133
Platinum	g/t	1.79	–	2.23	2.67	2.26	2.27	1.99	1.97	1.89	1.76	1.65
Palladium	g/t	1.85	–	2.37	2.48	2.19	2.23	1.97	2.01	1.99	1.84	1.68
Gold	g/t	0.27	–	0.36	0.47	0.39	0.34	0.33	0.29	0.27	0.26	0.27
Rhodium	g/t	0.12	–	0.14	0.16	0.14	0.15	0.12	0.13	0.14	0.12	0.11
Copper	% Cu	0.15	–	0.23	0.21	0.19	0.17	0.17	0.16	0.15	0.15	0.15
Nickel	% Ni	0.31	–	0.49	0.42	0.38	0.35	0.36	0.33	0.31	0.31	0.31
Sulfur	% S	0.75	–	1.21	1.06	0.96	0.85	0.90	0.83	0.77	0.72	0.74
3PE+Au	g/t	4.03	–	5.10	5.79	4.98	4.99	4.42	4.40	4.29	3.98	3.71
Longhole	kt	93,920	–	–	37	224	312	462	1,426	19,182	35,980	36,296
NSR	$/t	152	–	–	220	184	212	194	185	171	152	140
Platinum	g/t	1.88	–	–	2.83	2.25	2.63	2.30	2.35	2.14	1.88	1.71
Palladium	g/t	1.95	–	–	2.63	2.32	2.50	2.32	2.32	2.26	1.97	1.74
Gold	g/t	0.29	–	–	0.49	0.38	0.47	0.42	0.35	0.30	0.28	0.29
Rhodium	g/t	0.13	–	–	0.16	0.14	0.16	0.14	0.15	0.15	0.13	0.12
Copper	% Cu	0.16	–	–	0.19	0.18	0.21	0.20	0.18	0.17	0.16	0.16
Nickel	% Ni	0.33	–	–	0.41	0.38	0.43	0.42	0.37	0.35	0.33	0.32
Sulfur	% S	0.81	–	–	1.01	0.94	1.04	1.02	0.89	0.85	0.79	0.79
3PE+Au	g/t	4.25	–	–	6.12	5.09	5.76	5.19	5.17	4.85	4.26	3.86
Drift-and-Fill	kt	20,317	–	–	98	376	328	126	14	728	11,805	6,843
NSR	$/t	184	–	–	206	213	225	205	208	177	180	187
Platinum	g/t	2.30	–	–	2.55	2.58	3.06	2.89	2.90	2.20	2.23	2.37
Palladium	g/t	2.25	–	–	2.79	2.93	2.91	2.50	2.65	2.24	2.20	2.24
Gold	g/t	0.37	–	–	0.32	0.31	0.36	0.34	0.34	0.40	0.37	0.38
Rhodium	g/t	0.15	–	–	0.19	0.21	0.24	0.21	0.20	0.13	0.15	0.16
Copper	% Cu	0.18	–	–	0.18	0.19	0.17	0.16	0.16	0.18	0.18	0.18
Nickel	% Ni	0.37	–	–	0.39	0.42	0.36	0.33	0.34	0.35	0.37	0.37
Sulfur	% S	0.91	–	–	0.98	1.04	0.89	0.80	0.81	0.86	0.92	0.91
3PE+Au	g/t	5.07	–	–	5.85	6.03	6.57	5.94	6.09	4.97	4.95	5.15

Table 16.44 Total Mine Production

	Year Number	Total	-2	-1	1	2	3	4	5	6	11	21
	Year To									10	20	29
Total Ore Mined	kt	125,212	–	2	209	700	700	735	1,885	22,867	51,703	46,411
NSR BDT20	$/t	156	–	200	211	199	215	189	178	168	158	146
Platinum	g/t	1.94	–	2.23	2.64	2.43	2.80	2.34	2.26	2.11	1.95	1.81
Palladium	g/t	1.99	–	2.37	2.65	2.63	2.67	2.28	2.25	2.22	2.01	1.81
Gold	g/t	0.30	–	0.36	0.41	0.35	0.41	0.39	0.34	0.30	0.30	0.30
Rhodium	g/t	0.13	–	0.14	0.17	0.18	0.20	0.15	0.15	0.15	0.13	0.12
Copper	%Cu	0.16	–	0.23	0.19	0.19	0.19	0.19	0.18	0.17	0.17	0.16
Nickel	%Ni	0.34	–	0.49	0.41	0.40	0.39	0.39	0.36	0.34	0.34	0.33
Sulfur	%S	0.82	–	1.21	1.01	1.00	0.95	0.96	0.88	0.84	0.82	0.80
3PE+Au	g/t	4.37	–	5.10	5.88	5.58	6.07	5.16	5.00	4.78	4.39	4.04

Note: NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

16.6 Development and Production Compared to the Platreef 2017 FS

The development and production schedules of the Platreef 2022 FS were compared to the Platreef 2017 FS and the results are shown in Figure 16.57 to Figure 16.61.

Figure 16.57 Ore Tonnes and 3PE+Au Grades

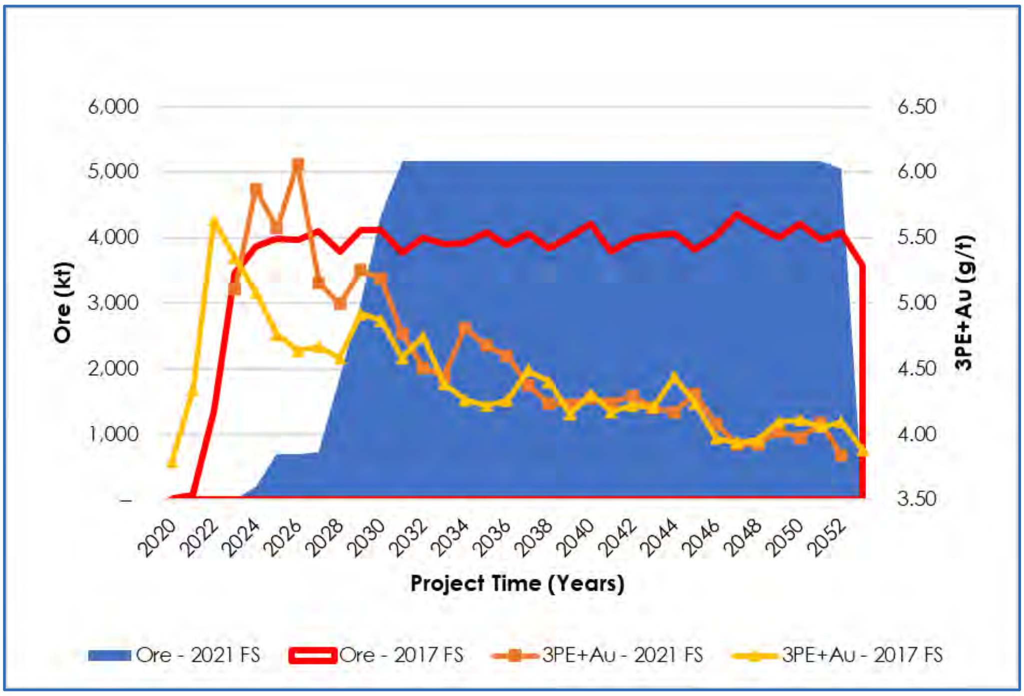

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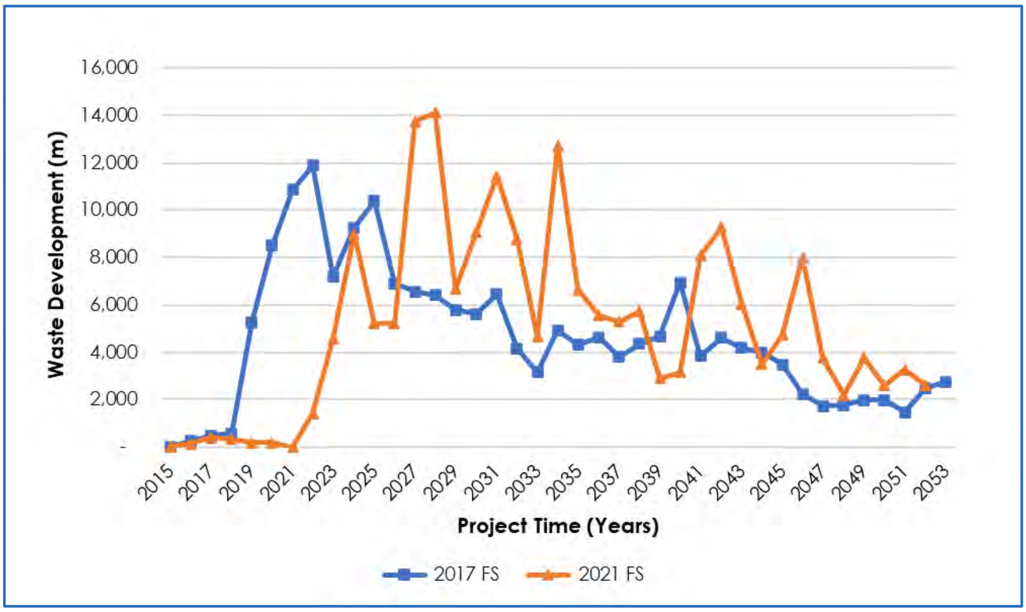
Figure 16.58 Waste Development



OreWin, 2021

Figure 16.59 Waste and Low-Grade Development

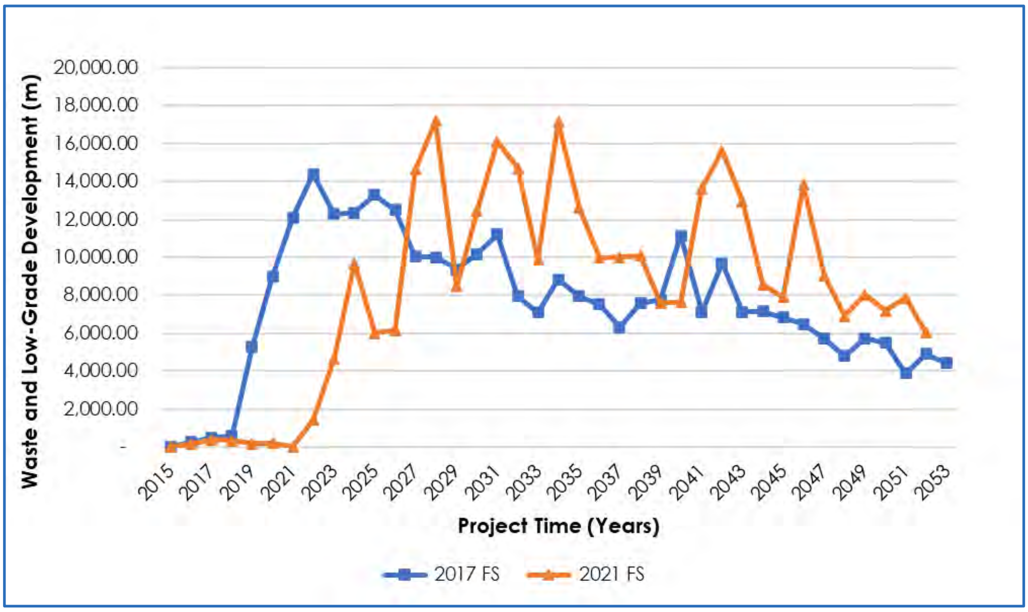


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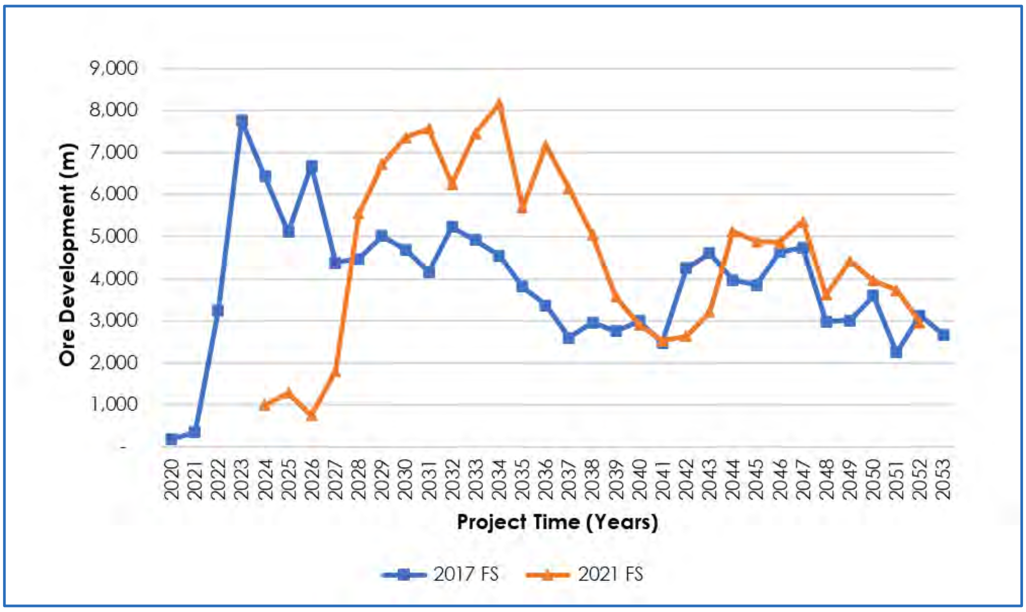
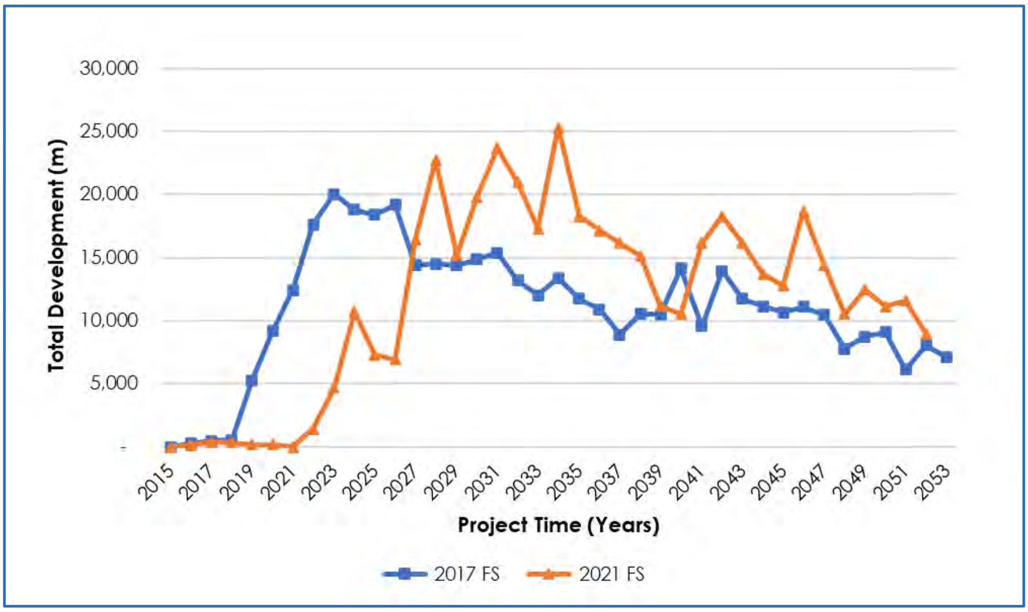
Figure 16.60 Ore Development



OreWin, 2021

Figure 16.61 Total Development



OreWin, 2021

16.7 Underground Infrastructure

Underground infrastructure involves several components, such as ore and waste handling systems, dewatering, maintenance shops, fuelling, ventilation, etc. OreWin worked with DRA and others on the underground infrastructure to ensure that the functional specifications were aligned with the mine design. This Section includes ore and waste handling down to the crushers, ventilation and refrigeration, and production return water. The additional facilities were reviewed and agreed upon, and OreWin completed the excavation designs to accommodate them.

16.7.1 Mine Dewatering

The dewatering system for Platreef can be separated into the following components, starting from active development ends back to the shaft:

- Face dewatering
- Tertiary dewatering
- Secondary dewatering; and
- Primary dewatering.

Each system feds the water back to the next. Each system will be described in more detail.

16.7.1.1 Face Dewatering and Tertiary Dewatering

Mine production return water will include drill water, mine service water, fissure water, backfill flush water, and backfill seepage. All development drives on the main levels will be driven on a positive gradient and will include a ditch system to allow mine production water to flow back to a series of collection sumps spaced every 400 m throughout the mine workings. Headings will use electric face pumps powered by the drill rigs to transfer water from the face to the collector sumps.

16.7.1.2 Secondary Dewatering

The mine water inflow is estimated to be 35 L/s during maximum production. The pumping system is designed for 150 L/s to account for spikes from initial groundwater inflows and paste backfill flushing. Each sump will not operate on a continuous basis but will have an overall utilization dependent upon actual mine water inflows and mine water usage. This system will allow for improved settling in the main sumps and less fines reaching the primary dewatering system.

The secondary dewatering system comprises several different types of sumps depending on their location in the mine. The higher laying sumps are generally gravity transfer sumps and lower sumps typically below 950 m Level are equipped with pumps. The sumps are described in more detail per level below.

750 m Level Sumps

The two sumps on the 750 m Level are gravity transfer sumps and are provided to collect water from the development and mining areas above the 750 m Level. These sumps are designed to transfer mine water and solids to boreholes which report to the collection sumps on the 850 m Level. Depending on ground conditions, casing is advisable to keep the boreholes open.

850 m Level Sumps

There are three gravity transfer sumps on the 850 m Level identical to the 750 m Level. These sumps are intended to collect water from the development and mining areas above the 850 m Level and from the gravity transfer sumps on 750 m Level. Boreholes will transfer mine production water from the 850 m Level sumps to the 950 m Level sumps or in some case directly to the 950 m Level Primary dewatering system.

There also three slurry transfer sumps on this level, which are designed to collect water and solids from the level and pump it to the gravity transfer sumps. The Slurry transfer sumps are a small cavity-style sump proven to work extremely effectively in the transfer of mine water, while alleviating the clean-up requirements of solids that is common with most transfer sumps.

One single-bay, settler-style sump is provided at 850 m Level for shotcrete borehole flushing. It reports directly into the primary dewatering system on 950 m Level.

950 m Level Sumps

The sumps on the 950 m Level will collect water from the development and mining areas above the 950 m Level, sumps on the 850 m Level, and the sumps below the 950 m Level. They collect mine water that has not already been directed to the primary dewatering system from 850 m Level.

There are two dual-bay, settler-style sumps on the 950 m Level. Both settle out the solids prior to pumping the mine water to the primary dewatering system on 950 m Level. For settling capacity of all the sumps of this nature, refer to Table 16.45.

Table 16.45 Sump Settling Capacity

Settler	Capacity per Side (L)	Inflow (L/s)	Percent Solids (w/w)	Solids (L/s)	Sump Retention Time (days)
North Sump – 950 m Level	154,688	24	1.0	0.080	22.4
North Sump A – 1,050 m Level	103,000	24	0.5	0.040	29.8
North Sump – 1,170 m Level	103,000	16	1.0	0.053	22.5
North Sump – 1,050 m Level	103,000	16	1.0	0.053	22.5
South Sump – 950 m Level	103,000	24	0.5	0.040	29.8

There are three sumps on the 1,050 m Level. Two are dual-bay, settler-style sumps, while the other is a slurry transfer sump. All these sumps either directly or indirectly report the mine water to the 950 m Level main pump clarification system.

1,200 m Level Bottom of Ramp Sumps

There are two bottom of ramp sumps on the 1,200 m Level. One is a dual-bay, settler-style sump, while the other is a slurry transfer sump. Both of these sumps report to their respective settler sumps at the 1,050 m Level.

16.7.1.3 Primary Dewatering

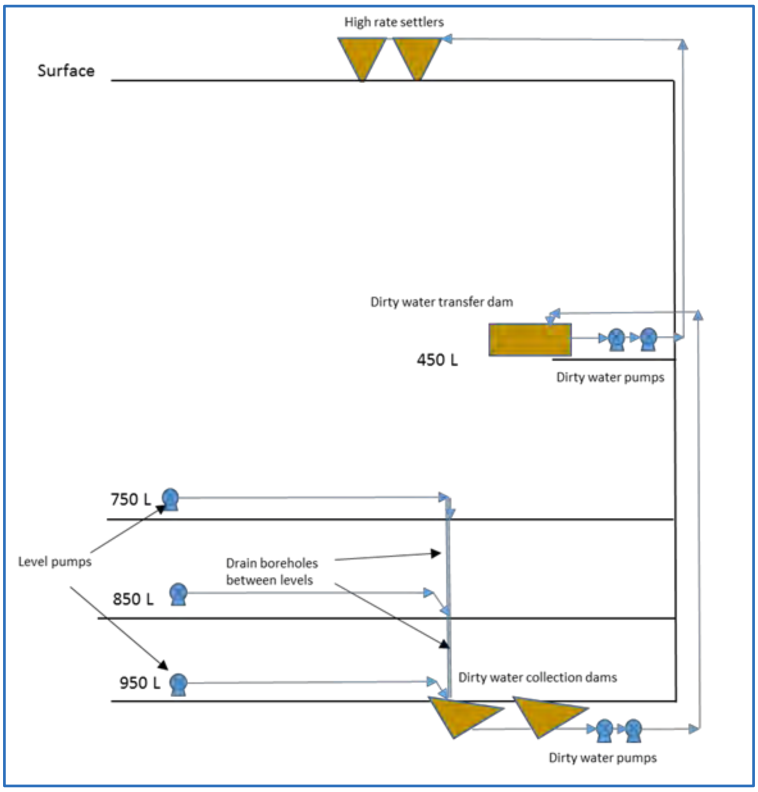
The primary pumping system will be a dirty water pumping system for Phase 1, located at Shaft 1, that will cascade water from 950 m Level to 450 m Level and then to surface where settling and clarifying will be done. During Phase 2, an underground settling and clear water pumping directly from 1,050 m Level to surface via Shaft 2 will be done.

Phase 1 - Primary Dewatering System

Mine return water will collect in two dirty water collection dams on 950 m Level. These dams are designed to remove a substantial amount of grit prior to being pumped to the 450 m Level relay pump station. The 450 m Level station will pump directly to surface into settlers.

The two dams on 950 m Level will be converted into collection dams that will drain through boreholes to a settling system that will be installed on 1,000 m Level after Shaft 2 has been commissioned (shown in Figure 16.62).

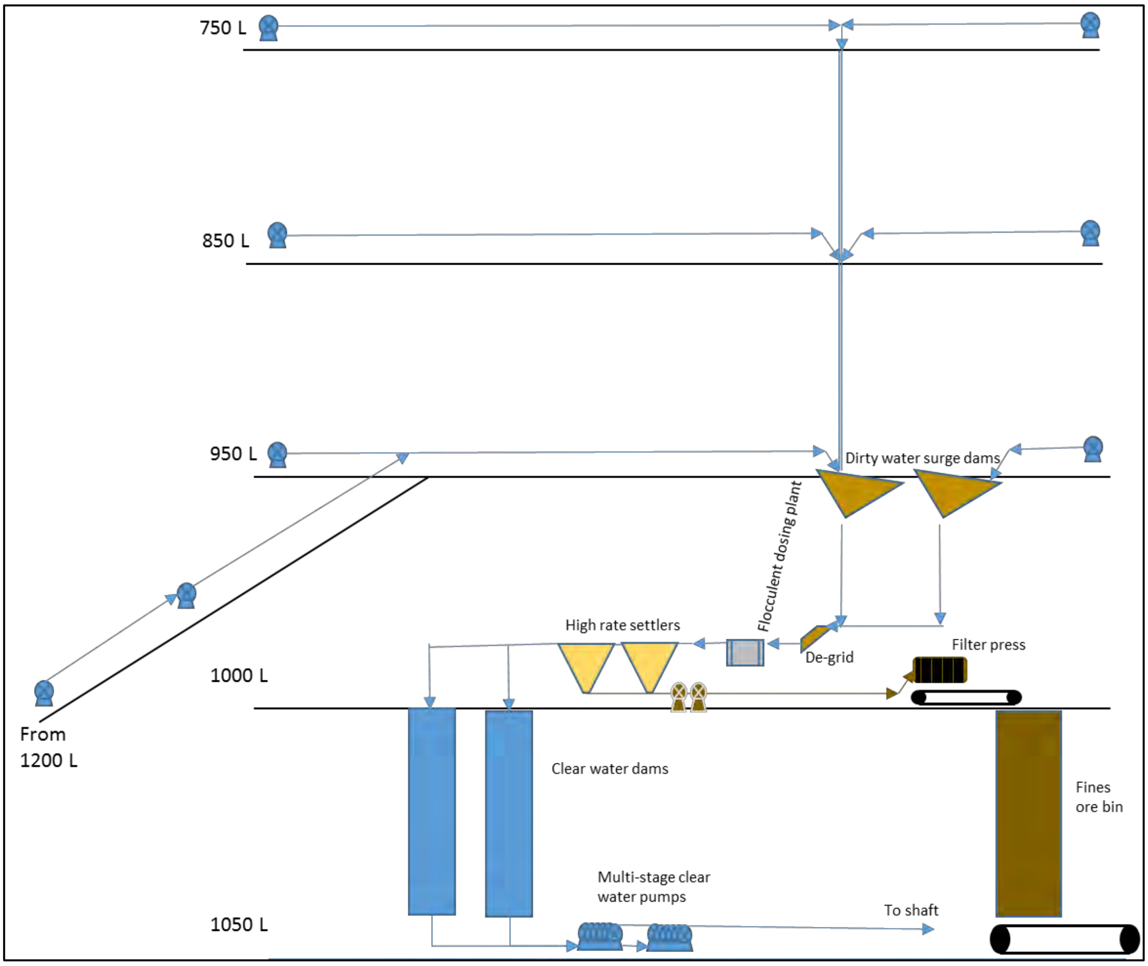
Figure 16.62 Phase 1 Primary Dewatering System



Phase 2 - Primary Dewatering System

The Phase 2 pumping system will be installed during the sinking phase of Shaft 2 and commissioned in 2028. This will be a clean water pumping system with underground settling and removal of grit and mud. The Secondary dewatering system will feed this system with dirty water into the collection dam on 950 m Level, refer to Figure 16.63.

Figure 16.63 Phase 2 Permanent Pumping System



Dirty water is pumped to drain holes located near Shaft 2 and collectively drains all the water from the levels above 950 m Level into surge dams. Water below 950 m Level will be pumped via the ramps into the same surge dams designed to collect 45% of grit, prior to entering drain holes leading to the settlers on 1,000 m Level.

At the settlers, a degritter removes the remaining grit (t larger than >1 mm). Flocculent is added by means of an automated system prior to entering the high-rate settlers. Suspended solids are settled out and clarified water overflows to the clear water dams, while the underflow is pumped to a filter press where the solids are discharged as filter cakes onto a conveyor, and fed into the ore handling system.

Filtrate from the filter press is pumped back to the water handling system. Clear water from the clear water dams is pumped to surface with two 1.8 MW multi-stage pumps through a shaft pump column located in Shaft 2.

16.7.2 Ore and Waste Handling Systems

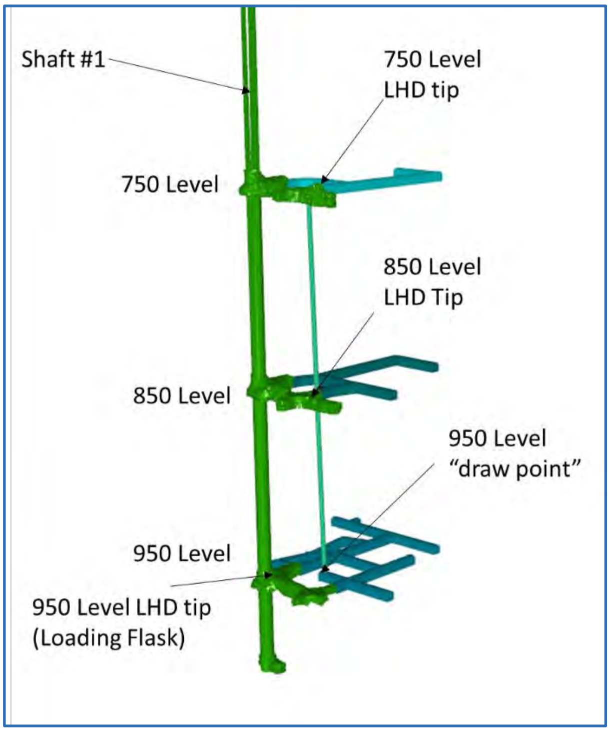
The Platreef rock handling system will be described in this section per Phase. Initially a temporary rock pass system will be used, then the Phase 1 permanent rock handling system will be installed which includes underground crushing for longhole stoping tonnes. All tonnes during Phase 1 will be hoisted via Shaft 1.

Phase 2 will use Shaft 2 as the hoisting shaft and a suitable rock handling system that includes truck tips, surge bins and crushers will be installed with a designed capacity of 6 Mtpa ore.

16.7.2.1 Phase 1 Temporary Rock Handling Pass

The temporary rock handling pass is located offset from Shaft 1. The pass will be installed making use of existing development established during shaft sinking. As soon as Shaft 1 is commissioned, a raisebore machine is set up on 750 m Level to pilot the hole down to 950 m Level. No development can occur on 750 m Level or 850 m Level until the piloting, reaming, and commissioning of the temporary rock pass is completed, as this temporary rock pass on the 950 m Level is the only method of removing rock on these two levels (shown in Figure 16.64).

Figure 16.64 Phase 1 Temporary Rock Pass

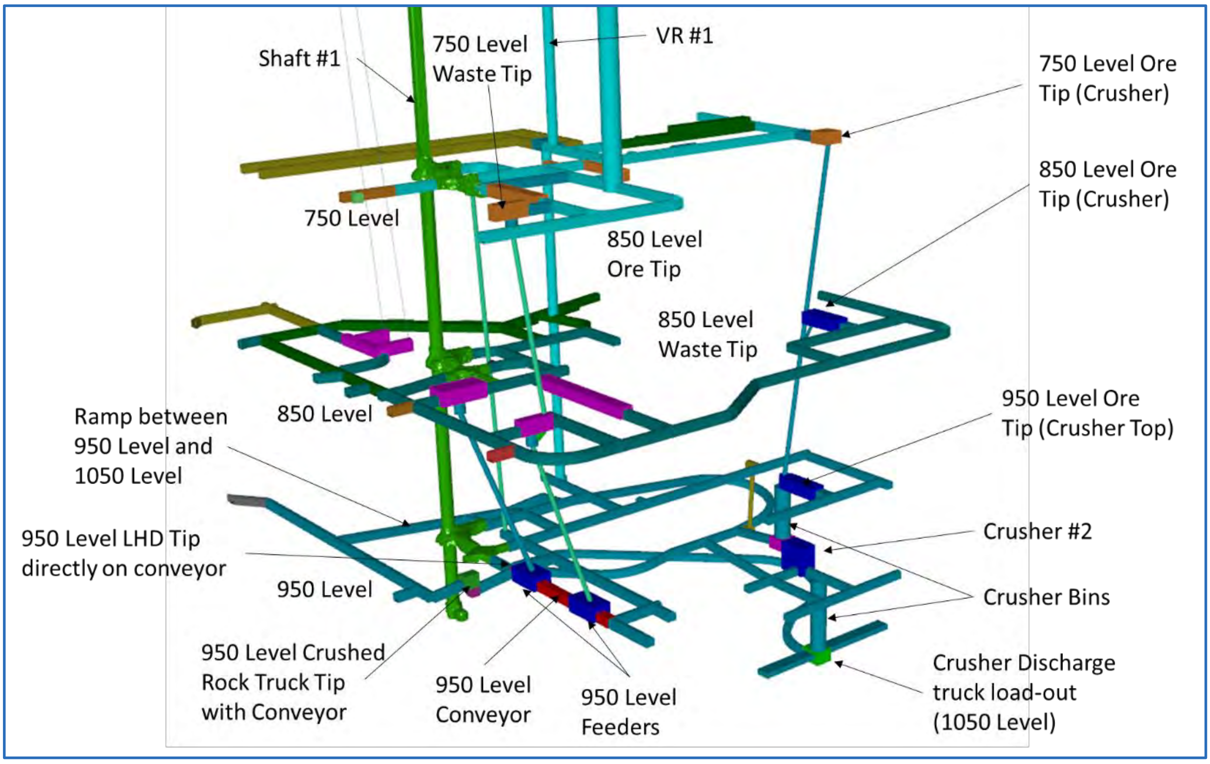


On 850 m Level, a holing into this raise will be drilled, which will allow for the removal of rock from 850 m Level. All rock will be loaded with an LHD at the bottom of the temporary waste pass on the 950 m Level and dumped into the skipping arrangement at Shaft 1.

16.7.2.2 Phase 1 Permanent Rock Handling

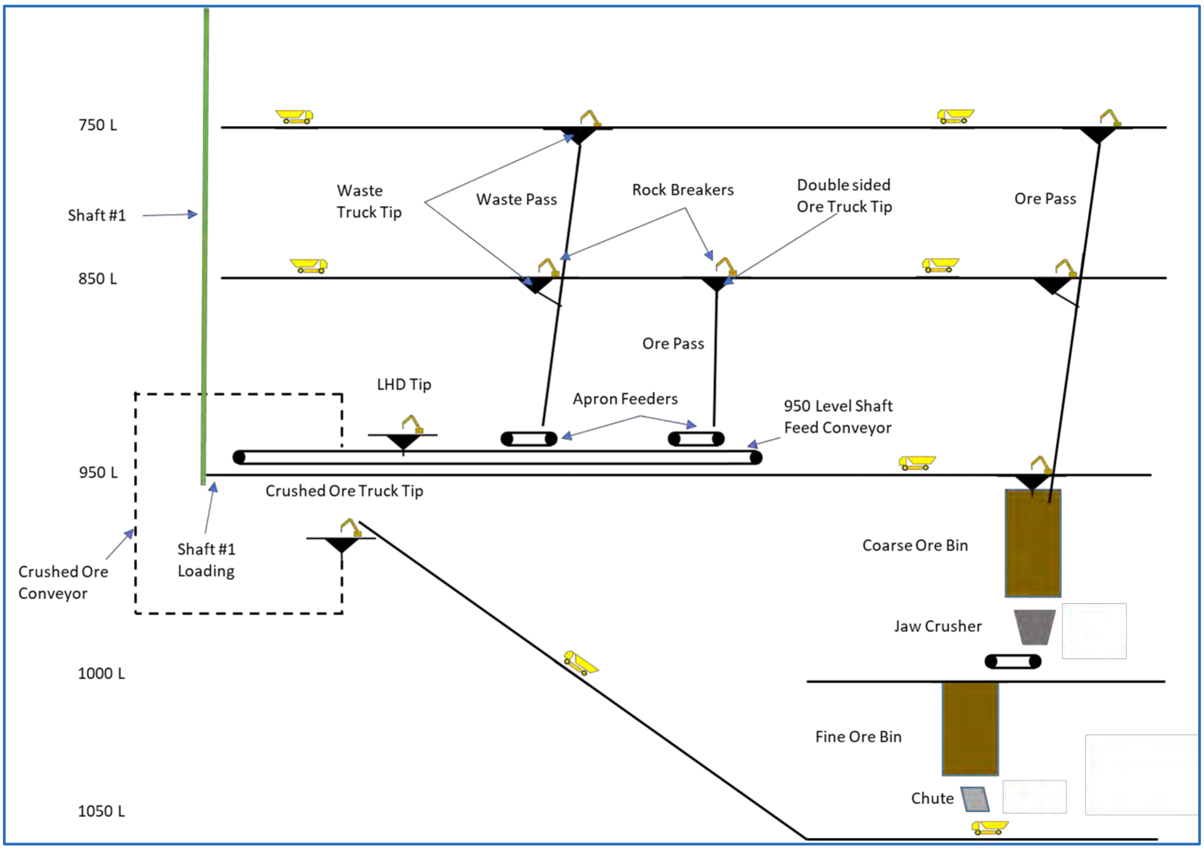
As the mine starts with ore production a Phase 1 permanent rock handling system will be constructed. The Phase 1 permanent rock handling system will comprise a truck tip for ore and waste on 850 m Level and a waste only truck tip on 750 m Level that will feed onto a conveyor on 950 m level (Shaft Feed conveyor). This conveyor will feed directly into the loading flask with a designed capacity to match the hoisting cycle of the shaft. Material generated on 950 m Level will be loaded onto the same belt directly by installing a LHD tip over the belt (shown in Figure 16.65).

Figure 16.65 Phase 1 Permanent Rock Handling Infrastructure – Isometric View



The indicated Crusher #2 will be required for the larger rocks produced by longhole stoping. The complete system, including the ore truck tips on 750 m, 850 m and 950 m Levels will be installed during Phase 1. Trucks will be used to tip blasted material on each main level and again to load the crushed material from the bottom of the crusher discharge bin on 1,050 m Level and transport it up a ramp to 950 m Level. On 950 m Level a truck tip with conveyor will be installed to load the material on the shaft feed conveyor. A diagram overview of the system is illustrated in Figure 16.66.

Figure 16.66 Phase 1 Permanent Rock Handling System Process Flow



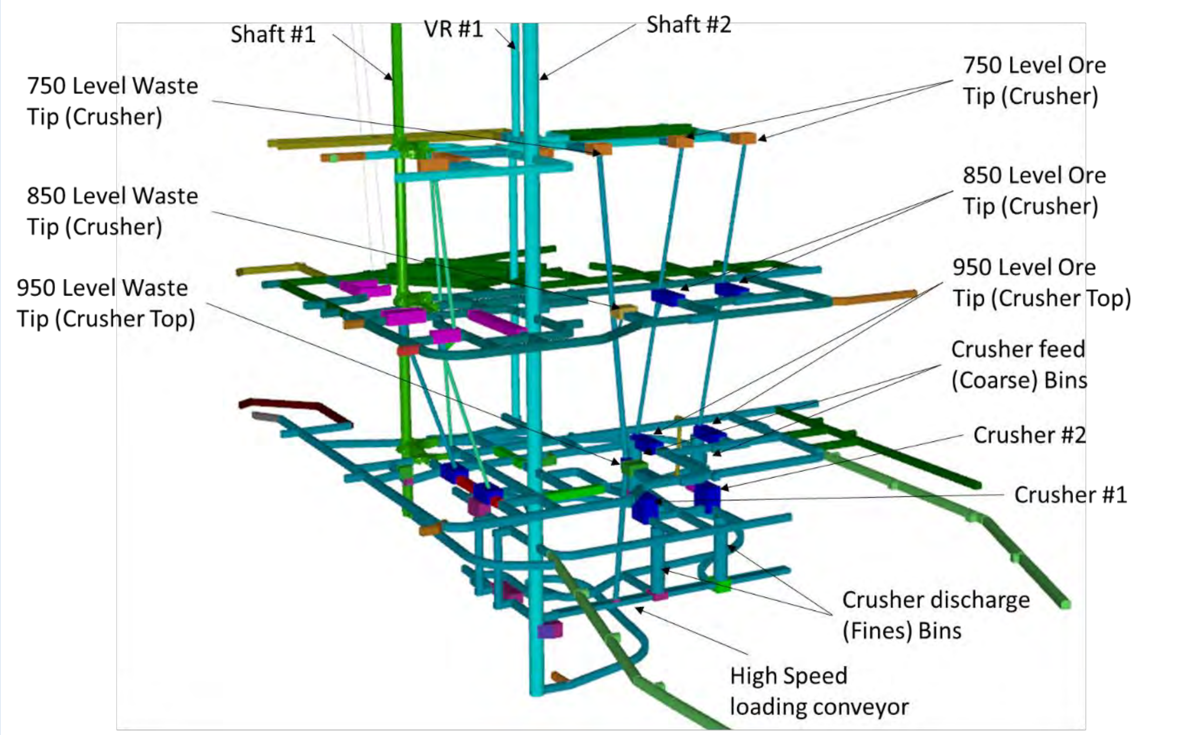
16.7.2.3 Phase 2 Rock Handling

During Phase 2, mining will increase and activities will be on various levels across the mine. For mining areas located above the 950 m Level, ore will be hauled by LHD from the stopes to grizzly stations on each sublevel. Finger raises into the ore passes will transfer the ore to the 850 m or 950 m main haulage Levels. At these haulage levels, the ore will be chute loaded into trucks for haulage to the truck ore dumps near Shaft 2.

For mining areas located below the 950 m Level, ore will be loaded directly into trucks by LHD. The trucks will then haul the ore up the access ramps to the 950 m Level and then to the truck ore dumps near Shaft 2.

On each of the three main haulage levels, two ore and one waste truck dump will be located near Shaft 2. The truck dumps will be equipped with grizzlies and fixed hydraulic rock breakers. One main waste pass will extend from the 750 m Level to the 1,050 m Level loading station. Two main ore passes will extend from the 750 m Level to the 950 m Level. At the 950 m Level, the ore will be diverted to the coarse ore bins. The 950 m Level truck dump grizzly will directly feed the coarse ore bin.

Figure 16.67 Phase 2 Rock Handling – Isometric View



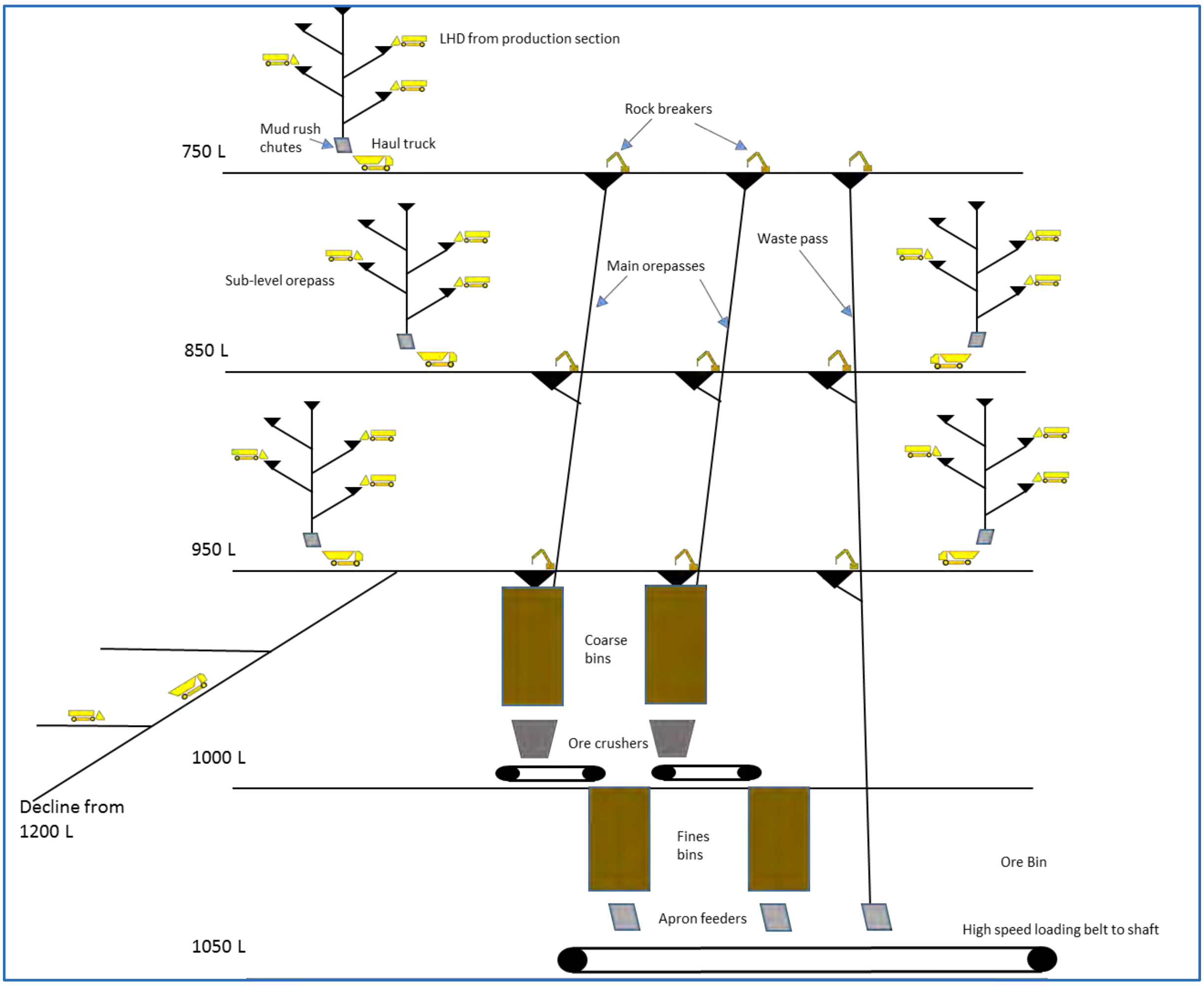
There will be two crusher stations, one below each coarse ore bin, midway between the 950 m Level and the 1,050 m Level loading station. Rock will be reduced in size by the jaw crusher and fed into the crushed ore bin. Each crusher will have a capacity of 3 Mtpa for an overall capacity of 6 Mtpa.

Apron feeders will load the crushed ore or sized waste onto the high-speed weigh conveyor belt on the 1,050 m Level. A skip-load of ore or waste will be loaded onto the high-speed conveyor while the skips are in transit in the Shaft. Upon arrival at the loading station, the load will be discharged directly from the conveyor into the skips via diverter chutes.

On surface, ore and waste will be discharged from the skips into the headframe bin. Discharge conveyors will transport it to either the mill or waste storage area.

A single waste pass system is provided with finger raises on various levels that report the waste directly to the load-out conveyor, bypassing the crushing plants.

Figure 16.68 Phase 2 Permanent Rock Handling System - Process View



Sublevel Ore Passes

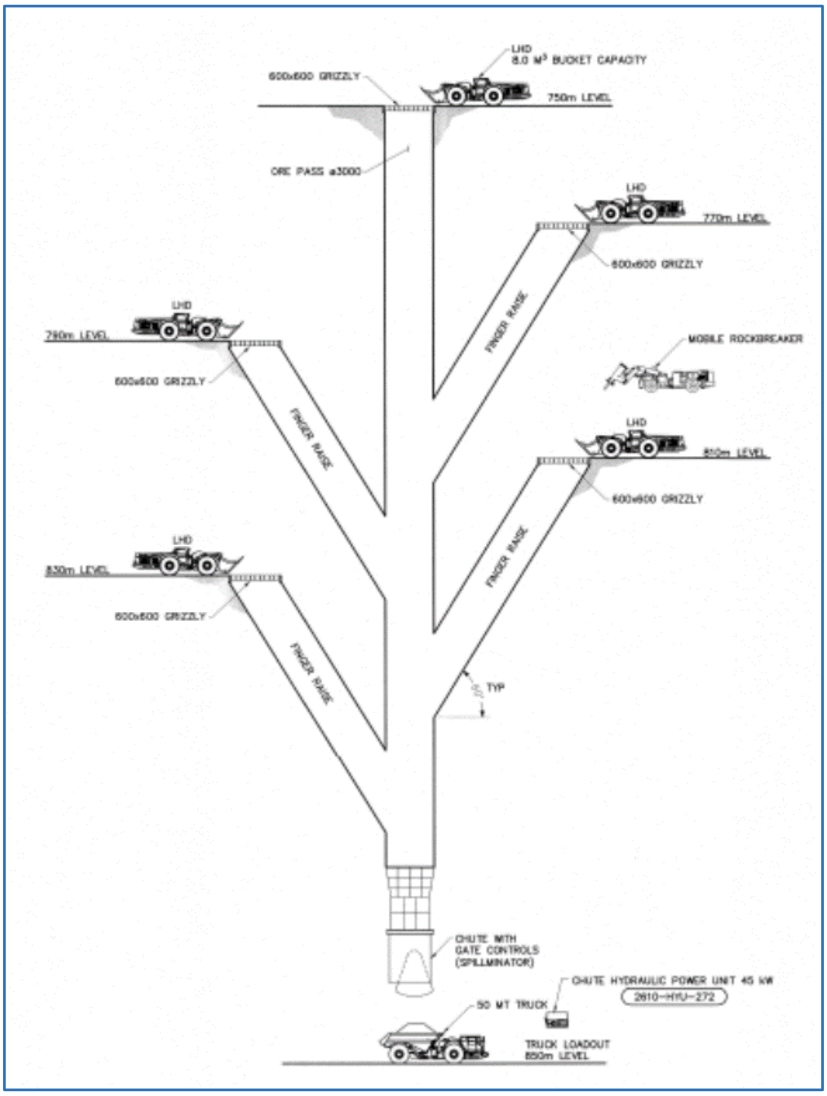
Rock production above 950 m Level will use sublevel ore passes for the movement of material down to the main levels. Rock from the development headings, longhole stopes, and Drift-and-Fill will be mucked from point of origin directly into the sublevel ore passes. The sublevel ore pass design incorporates the following:

- Four metre diameter raise bore holes will be used for the ore passes.
- Ore passes will go from above 750 m Level to 750 m Level, from 750 m Level to 850 m Level, and from 850 m Level to 950 m Level.
- Finger raises will be provided at each sub level into the ore pass.
- The top of the pass and its fingers raises will have LHD–style grizzlies sized at 600 mm x 600 mm.

- Mobile rock breakers will be available in the mining zones or at the grizzly stations to handle oversized rock.
- Grizzlies will be constructed at sill level to allow oversized rock to be swept off by the LHD and later resized by the mobile rock breakers.
- Truck chutes at the bottom of the ore pass system will be Spillminator-style chutes per South African standards depicted by Platreef.
- Sublevel ore pass locations are designed to optimise LHD tramming versus development costs. The ore passes are spaced 400 m apart on the sublevels, which makes the average haul distance for all stopes 100 m on the footwall drift and the longest haul distance 200 m, excluding top cut distances. To accommodate for the top-cut distances, an average of 175 m was used as the one-way haul distance in the productivity calculations.

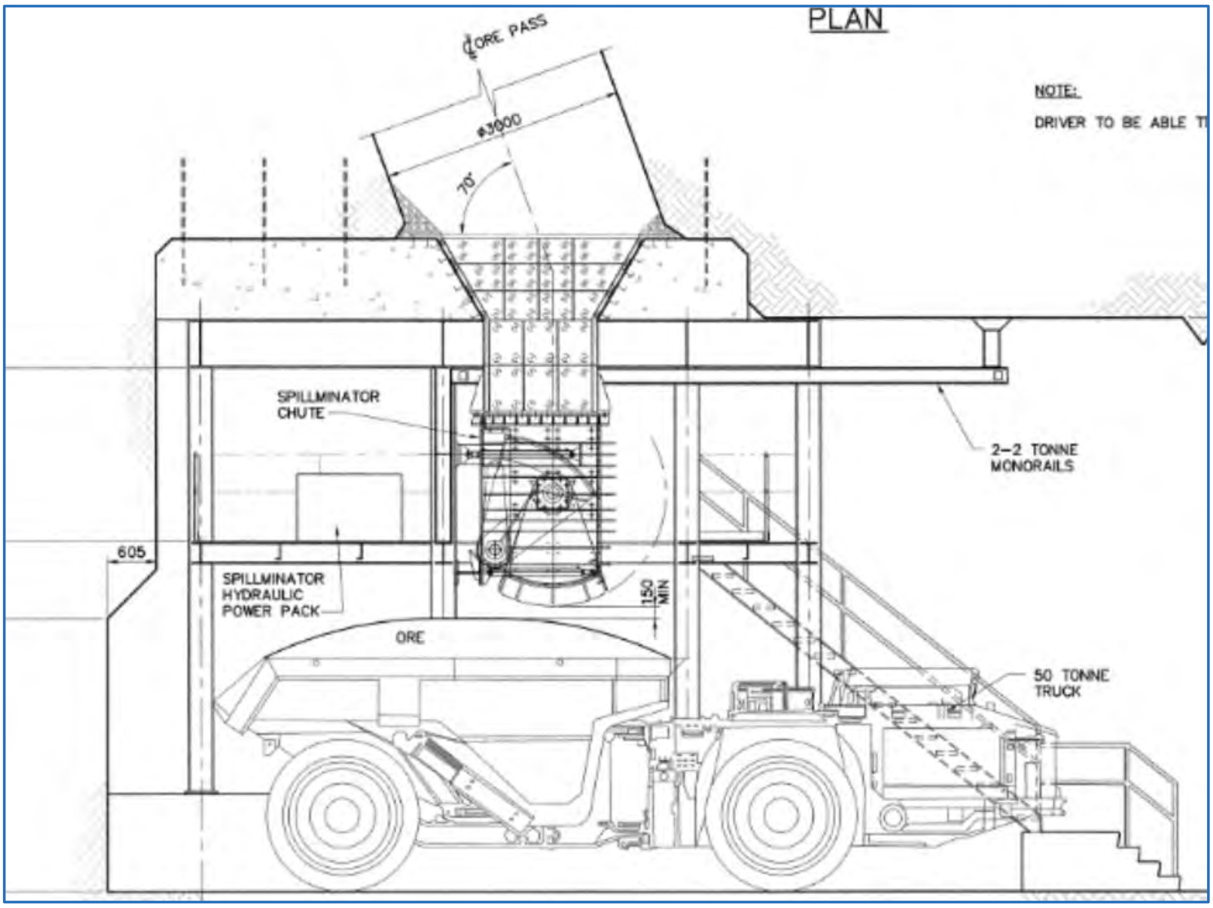
Refer to Figure 16.69 for the sublevel loading and transfer arrangement.

Figure 16.69 Sublevel Loading and Transfer Arrangement



The 50 t haulage trucks are loaded by Spillminator-style chutes at the bottom of the ore passes and will haul across the main levels (750 m, 850 m, and 950 m) to the main ore pass. At the main ore pass, the material will be sized to 450 mm x 450 mm. Refer to Figure 16.70 for a typical truck loading arrangement.

Figure 16.70 Typical Truck Loading Arrangement



Rock Handling Below 950 m Level

Rock production below 950 m Level will use haulage trucks to truck the material to the 950 m Level ore or waste passes. Rock is removed from the headings or stopes and moved to a remuck area on the level. From the remuck areas a dedicated LHD will load the haulage trucks and the material will be trammed up to 950 m Level.

The current design accounts for 33% of the fill quantity in secondary Longhole Stopes and 25% of the Drift-and-Fill stopes to be direct waste rock. During the CRF period of filling, the secondary stopes are filled with 50% waste rock. Ideally, the waste rock will be moved from the closest generation point to the stopes being filled.

Main Ore Passes

There will be two 4 m diameter vertical raise bore ore passes above the 950 m Level. Refer to Figure 16.71 for a schematic of one of the pass arrangements. The design of the ore passes will incorporate the following:

- Truck dump ore pass grizzlies sized 450 mm x 450 mm on the 750 m, 850 m and 950 m Levels are provided complete with stationary rock breakers with remote operational capability.
- At the 950 m Level truck dump, a transfer chute is provided to control feed to the coarse ore bin. This feed bypasses the truck dump grizzly. The transfer chute is a Kiruna-style chute with low level detection. See Figure 16.72 for a comparison of Spillminator-style and Kiruna-style chutes.
- Capability to control chutes automatically from a remote location will be provided.

Figure 16.71 Main Ore Pass Arrangement and Truck Chutes

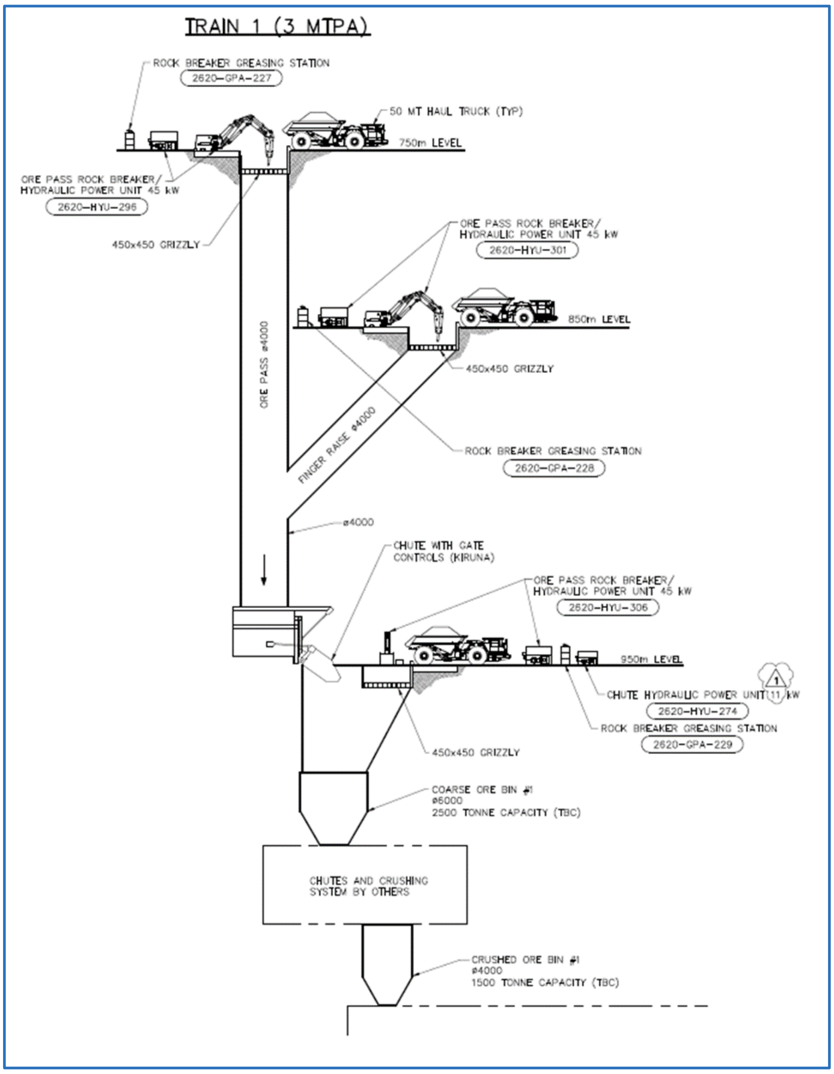


Figure 16.72 Chute Comparison

The Spillminator-style chute was selected for ore pass arrangements feeding haul trucks due to its unique safety capability of controlling a run of wet muck. The Kiruna style chute was selected for the transfer of ore from one ore pass to another due to its side-loading feature and capability to control ore flow well. It also facilitates the safe blasting of hang-ups better than the Spillminator-style design.

Below the 950 m Leve station is the coarse ore bin and crushing plants that feed into the crushed ore bin and the high-speed load-out conveying system. These systems were designed by DRA and Murray & Roberts.

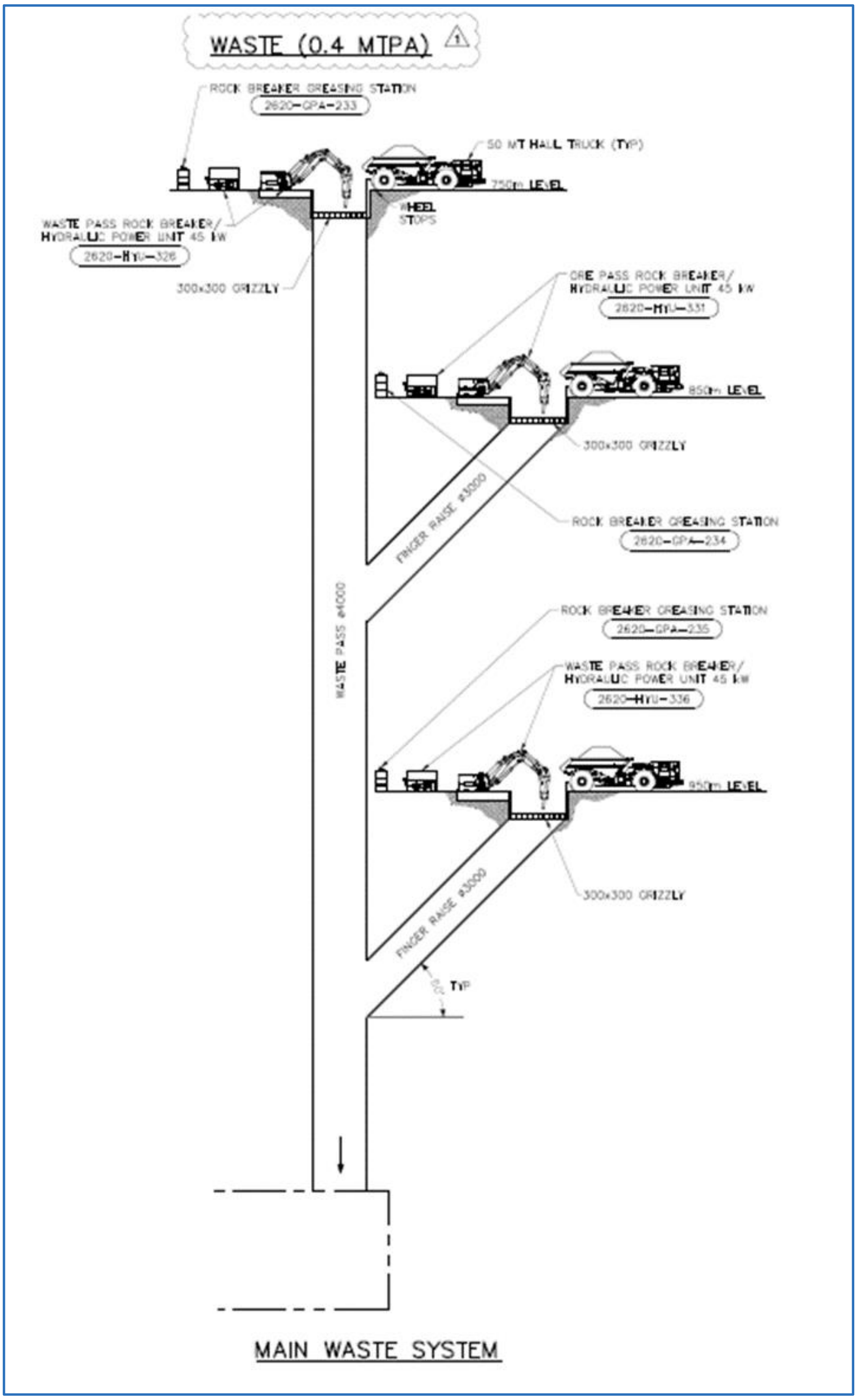
Waste Pass System

The waste pass system from the 750 m Level to the 1,050 m Level will be a pass 4 m diameter raise bore hole developed in two legs: one leg from 750 m Level to 950 m Level, and the other from 950 m Level to 1,050 m Level. The waste pass design will incorporate the following:

- Truck dumps will be located on the 750 m, 850 m, and 950 m Levels, with finger raise access into the main waste pass.
- A 300 mm x 300 mm grizzly will be located at each dump point.
- A stationary rock breaker with remote capabilities will be located at each grizzly.
- Ventilation control will be designed to prevent recirculation of air or excessive dust at grizzlies.

Figure 16.73 illustrates the waste pass system.

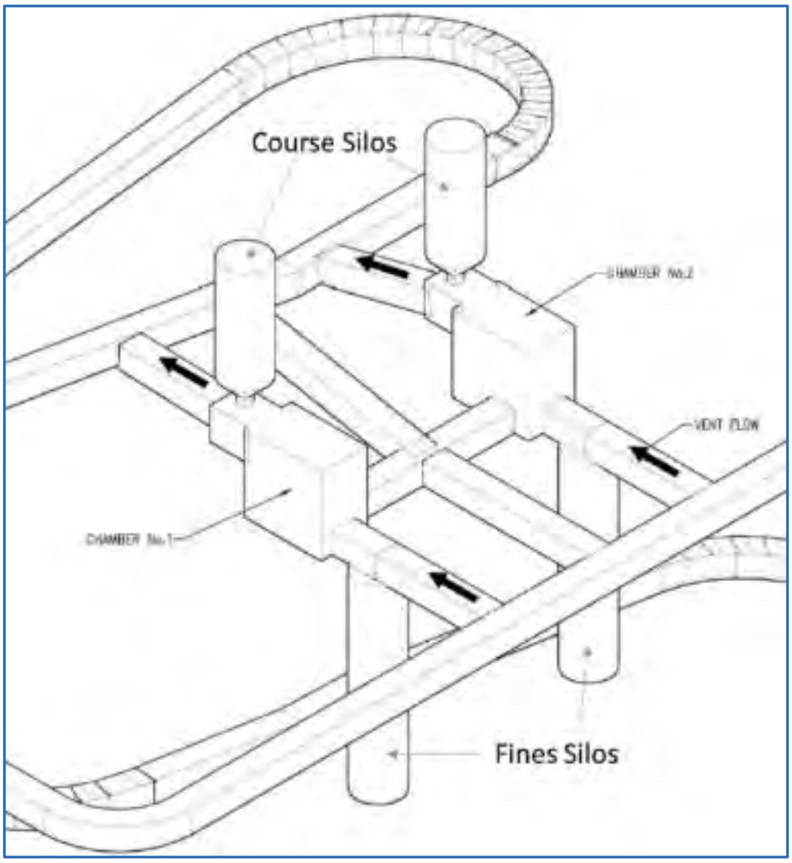
Figure 16.73 Waste Pass System



Crushing Facilities

The crusher chambers are accessible from the main shaft on 1,000 m level and also from the decline to shaft bottom. This enables ease of installation and maintenance, assists with ventilation through these chambers and allows for early construction. Figure 16.74 indicates the position of the crusher chambers along with the coarse and fines silos.

Figure 16.74 Position of Underground Crusher Chambers



The main ore tips are equipped with 450 x 450 mm static grizzlies, along with impact rock breakers to break oversize rocks. Waste tips are equipped with 300 x 300 mm static grizzlies as well as impact breakers.

After discharging the ore through the grizzlies, coarse ore is stored in the coarse silo. The ore is drawn from this silo through a chute with a controlled radial gate by means of a vibrating feeder. This vibrating feeder controls feed onto a short conveyor belt fitted with a self-cleaning belt magnet, to remove any tramp iron. The tramp iron is discharged into a chute that diverts into a cassette that can easily be removed and transported by a general cassette carrier.

The belt feeds ore onto a scalping grizzly feeder. The fines fall through into a chute feeding the crusher discharge belt and oversize rocks feed into the jaw crusher. The crusher discharge belt feeds into the fines silo.

Figure 16.75 and Figure 16.76 show the layout of a crusher chamber.

Figure 16.75 Crusher Chamber – Section View

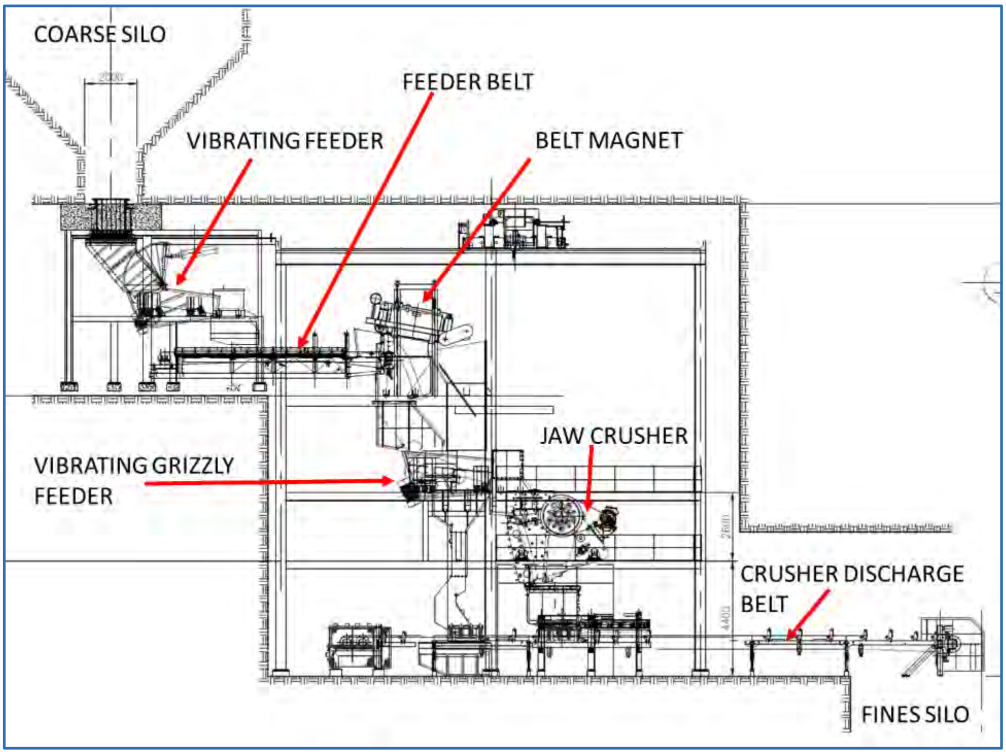


Figure 16.76 Crusher Chamber – Plan View

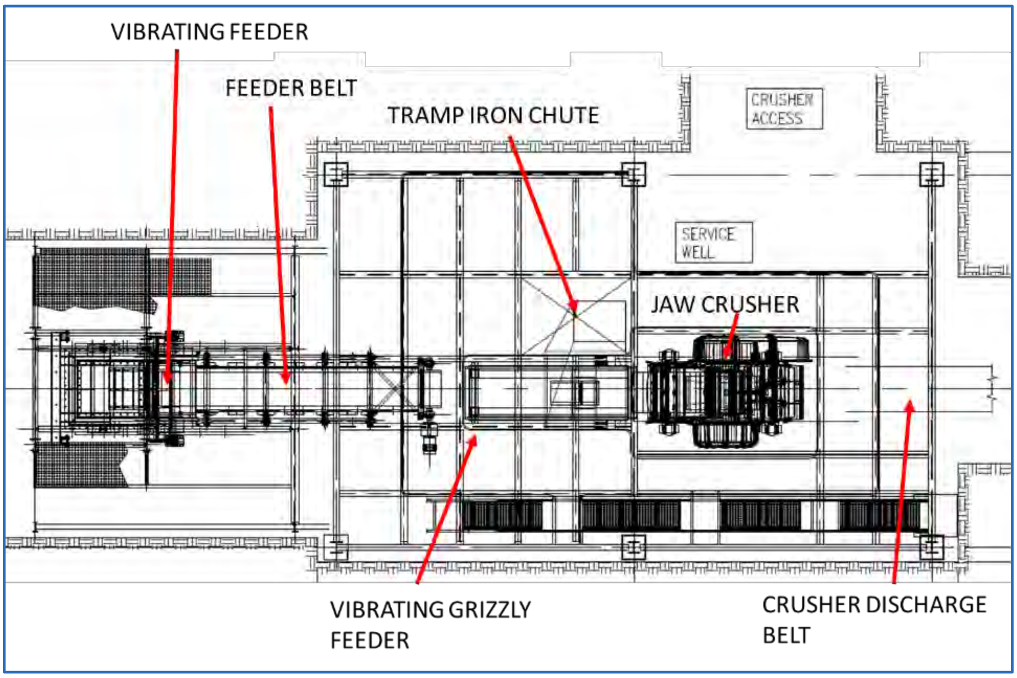


Figure 16.77 indicates the process of removing tramp iron. The belt magnet will lift the tramp iron off the belt and discharge into a chute then into a bin. This bin will be collected by a cassette carrier and transported to the shaft. The belt magnet is a self-cleaning suspended electromagnet.

Figure 16.77 Tramp Iron Handling

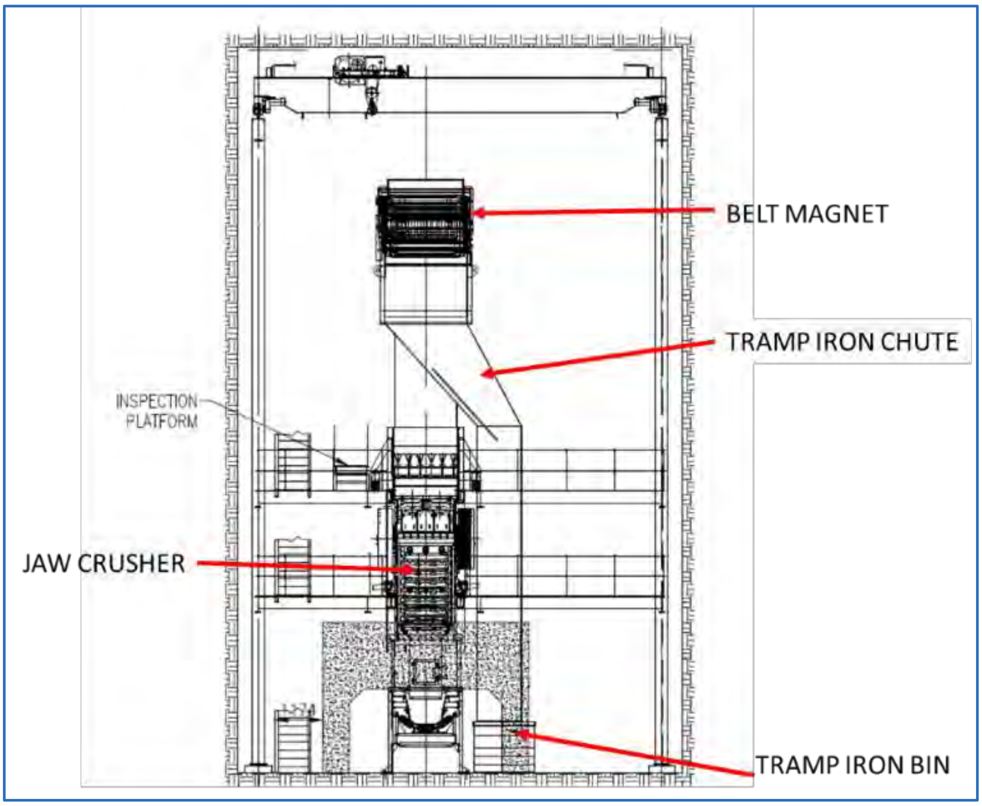


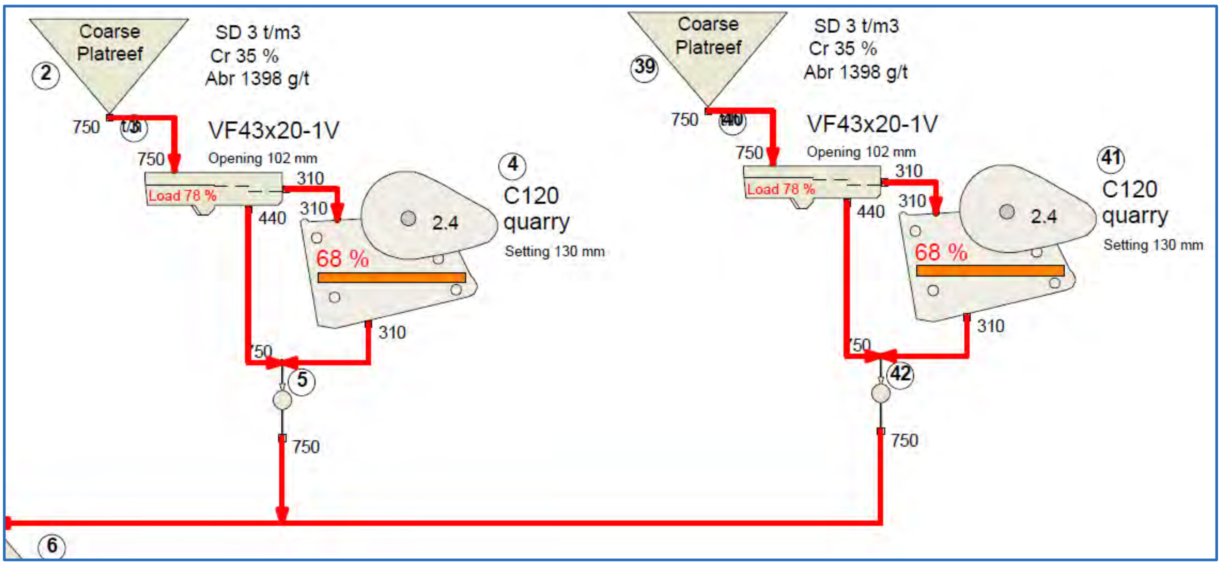
Table 16 44 summarises the operating criteria of each of the two crushers.

Table 16.46 Crusher Operating Criteria

Criteria Description	Criteria	Unit of Measure
Material Type	Platinum Ore	
Operating Hours/Shifts	18 hours	
Crusher Loading	68	> %
Feed Solids Density	3.04-3.19	t/m³
Bulk Density	1.902	t/m³
Moisture Content	5-10	%
Maximum Feed Rate per Crusher Circuit	750	t/h
Average Feed Rate per Crusher Circuit	600	t/h
Maximum Feed Size	550	mm
Compressive Strength	142 - 200	MPa
Crushability Work Index	14.4- 22.1	kWh/t
Abrasion Index	0.4 - 0.41	g
BBWi @ 106µm Screen	21.6-23.5	kWh/t
Product Size 100% Passing	270	mm
Product Size 80% Passing	150	mm
Maximum Feed Rate per Crusher	310	t/h
Average Feed Rate per Crusher	250	t/h

A Bruno process simulation was undertaken on the maximum feed rate, to determine the crusher duty and size, as shown in Figure 16.78.

Figure 16.78 Bruno Process Simulation



16.7.3 Material Handling Logistics

Material will be delivered to, stored at and distributed from one central main store on surface. Specific consumables, such as fuel, lube, explosives and emulsion, will be stored in separated areas within the mine area and will also be controlled by the main store control system.

Any delivery vehicles will report to the main store first from where they will be directed to the correct off-loading area.

On request, through the material controls system, spares, consumables or any other material will be loaded into material cassettes according to the area and section ready to be delivered to underground sections.

Cassette carrier vehicles, also referred to as utility vehicles (UV), will be utilised for material transport. The cassettes that were filled at the main stores are then transported to the shaft and offloaded at the material transfer area. This area is divided into the “full cassettes” area and “empty cassettes” area close to the shaft bank.

The full cassettes will be loaded onto a trailer, two cassettes per trailer, whereafter a tractor tows the trailer into the cage. The driver disembarks and the tractor is guided by means of beacons and remote control through the cage. The trailer is then unhooked from the tractor and locked into position. This tractor then waits on the other side of the shaft for a trailer with empty cassettes to be brought to surface. While this tractor is waiting for empty cassettes, another tractor is in the process of loading full cassettes to follow the same path as described above.

The same process is followed on 750 m, 850 m and 950 m Level stations with the exception of the loading and offloading of the cassettes in the material transfer area. An overhead crane is installed to fulfil this function.

16.7.3.1 Underground Material Handling System

Phase 1:

Shaft 1 will be equipped with an interchangeable cage/skip known a bridal system. During every shift the skip will be swopped out with a cage for men and material hoisting. Material can also be slung down the shaft without the cage if the weight of material exceeds the maximum allowable design weight shown in Table 16.47.

Table 16.47 Capacity and Slinging

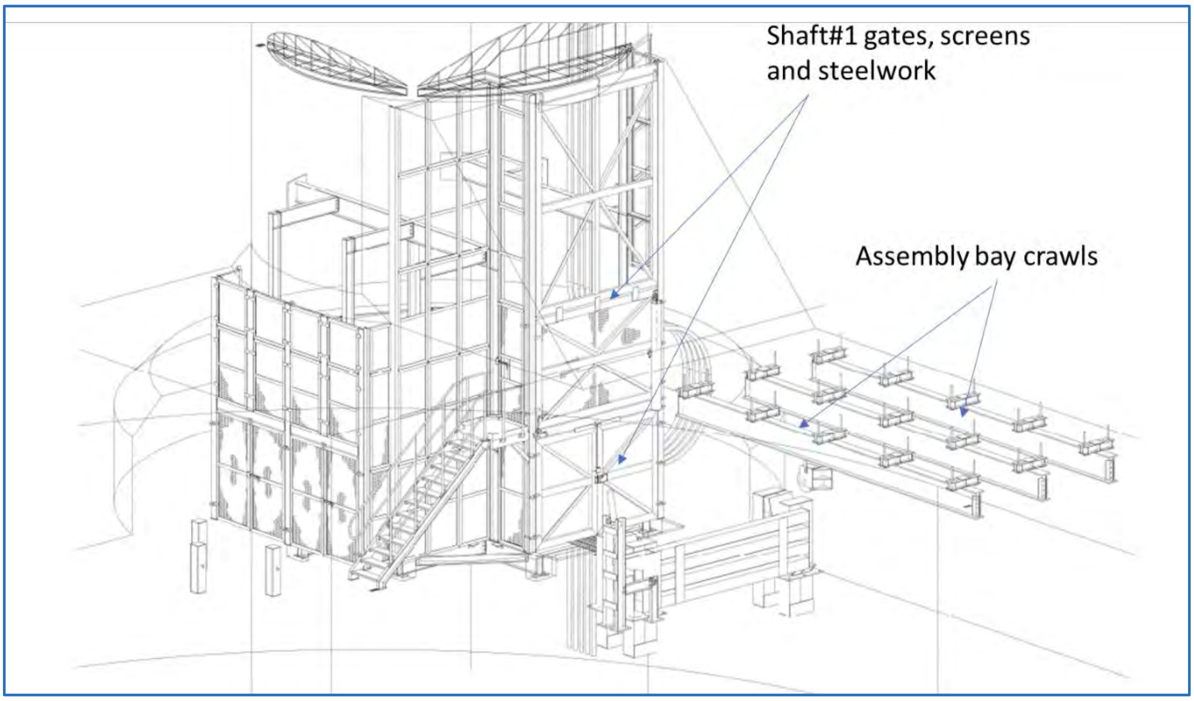
	Capacity (tons)
Maximum allowable mass attached to rope	21.25
Mass of attachments	0.65
Mass of bridle & attachments	2.95
Mass of slinging crosshead & attachments	1.95
Maximum slinging mass in bridle	18.3
Maximum mass in slinging crosshead	19.3
Maximum slinging mass with cage & skip	12.5

Material will either be slung, palletised, loaded in tankers or containers. Large mobile equipment will be split apart and slung down the shaft and reassembled on the stations in provided assembly bays located on 750 m Level, 850 m Level and 950 m Level.

Each assembly bay will be equipped with three overhead cranes with 20 ton manual chain hoists on the Northern side of the shaft. The crawls will be 9 m long with 1,750 mm spacing between centres and will be used for the handling of large equipment during the operation of the shaft. The typical shaft station layout is shown in Figure 16.79.

Material will be palletised and loaded into the cage by a multi-purpose vehicle equipped with a forklift attachment on an extendable boom. The same vehicle will be used to load the pallets of material onto caste carriers for further transport throughout the mine.

Figure 16.79 Typical Shaft Station with the Assembly Area



Phase 2:

Shaft 2 will have the option of driving in and out of the cage with mobile equipment and full sizes cassettes that can be parked in the cage. The cassettes will be loaded onto a specially designed trailer that can be hooked to a tractor that can pull these trailers into the cage and off-hook them remotely. There will be two tractors allocated to the levels and two on surface. The two tractors located underground will be transported between the different levels. As the cage reaches surface, one of the two tractors will remove the empty trailer in the cage towards the one side and the other tractor will pull in with fully loaded trailer from the other side. On the underground levels the full trailers will be removed to the one side and the second tractor, with the empty cassette, will pull into the cage from the other side.

The tractor with the full cassette will transport it to the material transfer area while the other tractor waits for the next cage to arrive. The underground material transfer areas are equipped with overhead cranes to load and offload the utility vehicle cassettes. A “telehandler” will fulfil that function on surface.

Figure 16.80 shows the material handling arrangement on each main level, i.e. Levels 750 m, 850 m and 950 m.

Figure 16.80 Station Material Handling Layout

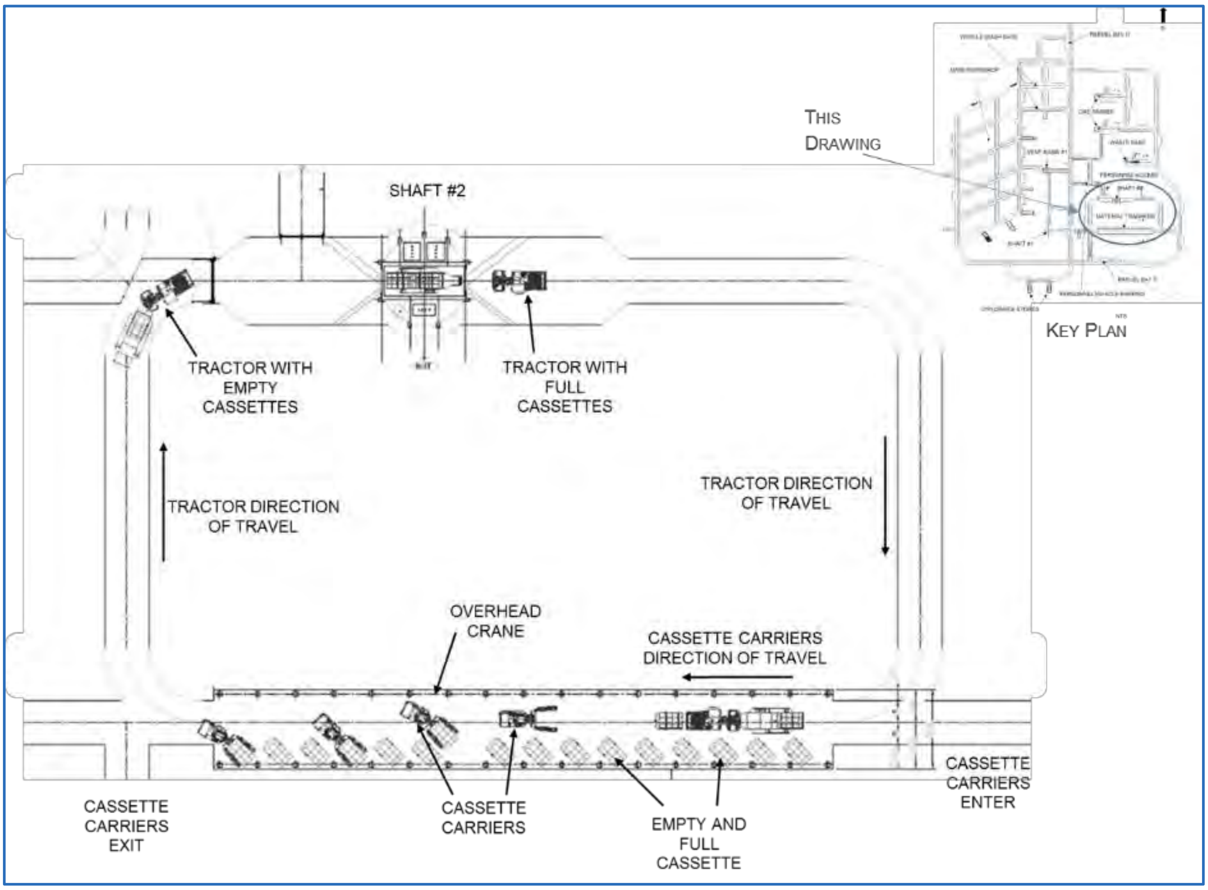
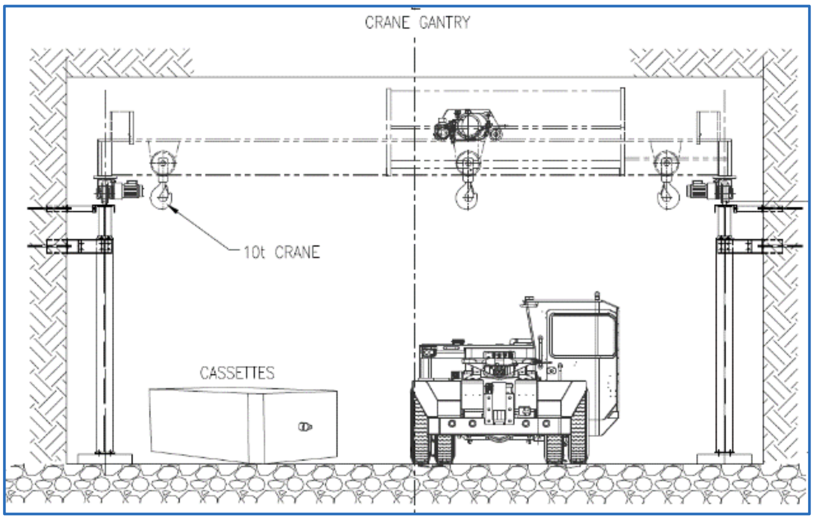


Figure 16.81 shows the material transfer area with the 10t overhead crane.

Figure 16.81 Material Transfer Overhead Crane



16.7.3.2 Explosives Handling and Distribution

The bulk explosive product selected for Platreef is emulsion, offloaded and stored on surface in tanks. Explosives accessories will be offloaded at a designated explosives offloading area and transferred into explosives cassettes, to be transported underground.

During Phase 1 emulsion will be transported from surface in dedicated cassettes directly to the working areas. Explosives accessories will be collected from the applicable underground stores and distributed by a dedicated explosives team.

During the later part of Phase 1 an emulsion vertical drop system will be constructed that will drop emulsion down directly to 850 m and 950 m Levels via a borehole. Sensitizer will be transported separately in tanks via the shaft.

16.7.4 Mine Services

Compressed Air

No compressed air will be installed during Phase 1. During Phase 2 compressed air will be supplied to applicable facilities around the shaft areas, workshops, main ore passes and shaft station infrastructure. Compressed air will not be distributed throughout the mine due to prone to leakages and general wastage.

Service Water

Service water will be recirculated from surface back underground. During Phase 1 the water will be settled on surface and stored in service water tank before distributing back underground, through a 200NB (ASTM A106 Grade B, Schedule 120) shaft pipe column, to the respective production levels. During Phase 2 clear water will be pumped to surface UG settling will be done.

Potable Water

Potable water consumption is calculated at five litres per person per day for underground consumption. For the purpose of calculating pipe sizes and volumes, it is assumed that a maximum number of 1,200 people will be underground at any stage in the life of mine. This amounts to 6 kL of potable water per day plus an additional 1.2 kL per day for the settler's flocculent plant, totalling 7.2 kL per day.

A 50 mm diameter pipe column will be installed to supply potable water to the underground workings.

Fire Water

Fire water will be stored on surface and distributed to underground through by means of a 200NB column.

Fire water will be drawn from a dedicated source on surface and gravity fed to the underground reticulation system, ensuring the availability of 500 m³ at all times. The tanks will be fitted with a dual suction.

Each tank section will have an infill, overflow, drain, suction, test return and diesel engine cooling water return line nozzles.

The tank has been designed to supply dedicated firewater via fire water mains to sustain firefighting from two hydrants operating simultaneously for 120 minutes.

Firewater reticulation pipe work will be SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be class 16. All isolation / section valves will be UL listed / FM approved.

Fire hydrants will be fed off the fire water column and will be placed no further than 60 m apart in the required areas. The maximum permissible velocity is 6 m/sec in the hydrant reticulation pipework. Cognisance has been taken to ensure compliance with these limitations when sizing the ring main

Due to the variety of risks associated with this project, a vast number of suppression system types have been designed and catered for. In all instances, the systems comply strictly with the applicable codes of practice, both locally and internationally. Each system has been designed as a fit for purpose solution which protects the equipment and personnel as well as does not restrict the operation. Table 16.48 is a list of the typical systems used.

Table 16.48 Types of Suppression Systems

System	Typical Location
Medium velocity spray system	Lubrication rooms, lube packs, underground conveyors, and hydraulic power packs
High velocity spray system	Transformers
Free agent gas suppression system	Sub-Stations and MCCs
Foam/water deluge system	Fuel and lube storage on surface and underground
Hose reels and extinguishers	Site-wide, on all structures, in all buildings and are located both on surface and underground

16.7.5 Workshops

Mobile equipment will remain underground for the duration of the machine's life cycle and will be serviced/maintained in applicable underground workshops. Machines will only come out of the mine for a complete OEM refurbishment, or to be scrapped and replaced.

There are three main workshops, one per production level; 750 m, 850 m and 950 m.

Workshops on 850 m and 950 m levels are similar in size and layout. The workshop on 750 m Level is smaller due to lower production and fleet size.

Production fleet vehicles operating mainly at the production face (drill rigs, bolters and LHDs) will be serviced and maintained (minor repairs) at satellite workshops where practical, to be constructed as close to the working areas as possible. All vehicles will revert to the main workshop for major services/repairs.

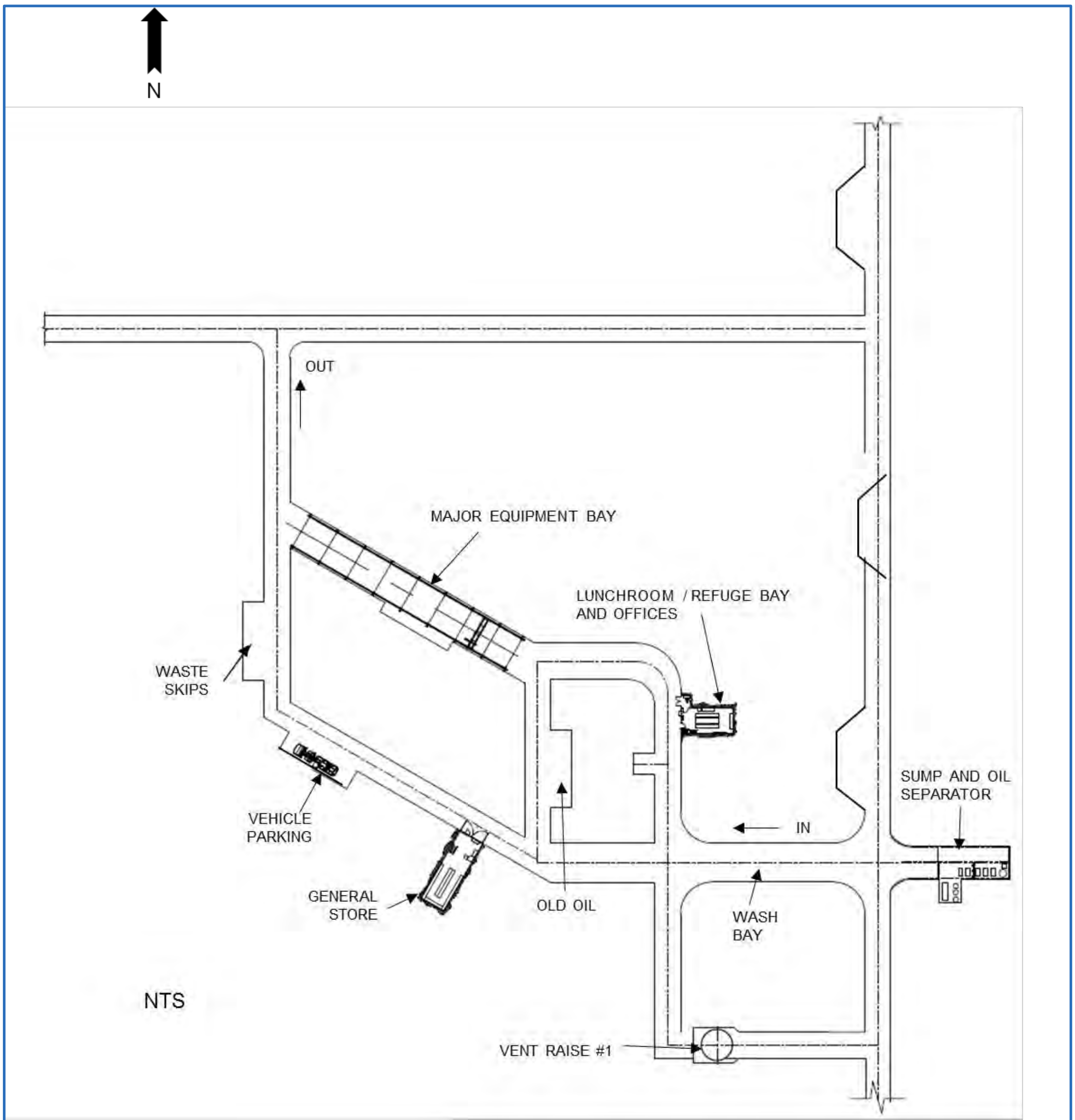
All other vehicles will report to the applicable main workshop for minor/major services and repairs.

750 m Level Workshop

The workshop indicated at Figure 16.82 on 750 m Level comprises:

- A vehicle wash bay;
- An office area with lunch room to double up as a refuge chamber;
- Toilet facilities;
- An OEM store;
- A waste oil storage area;
- Waste skips storage area; and
- A single large equipment service and repair bay, equipped with a ramp and 25t overhead crane.

Figure 16.82 750 m Level Workshop



850 m and 950 m Level Workshops

The workshops on 850 m and 950 m Levels illustrated in Figure 16.83 and Figure 16.84 are similar in design and layout comprising:

- Two vehicle wash bays for large equipment;
- One vehicle wash bay for small equipment;
- An office area with lunch room to double up as a refuge chamber;
- Toilet facilities;

- Two OEM stores;
- Lifting equipment store;
- Hose store with repair facilities;
- Sub-assembly store with separate areas allocated for new and for old equipment;
- Welding bay;
- Waste oil storage area;
- Waste skips storage area;
- Two large equipment service and repair bays, each equipped with a ramp and a 25t overhead crane;
- Two large equipment repair bays with no ramps, each equipped with a 25t overhead crane (extension of the crane mentioned above);
- One medium equipment service and repair bay equipped with a ramp and a 10t overhead crane;
- One medium equipment service and repair bay equipped a 10t overhead crane (extension of the crane mentioned above);
- Tyre bay and storage area; and
- LDV service and repair bay equipped with a LDV lift.

Figure 16.83 850 m Level Workshop

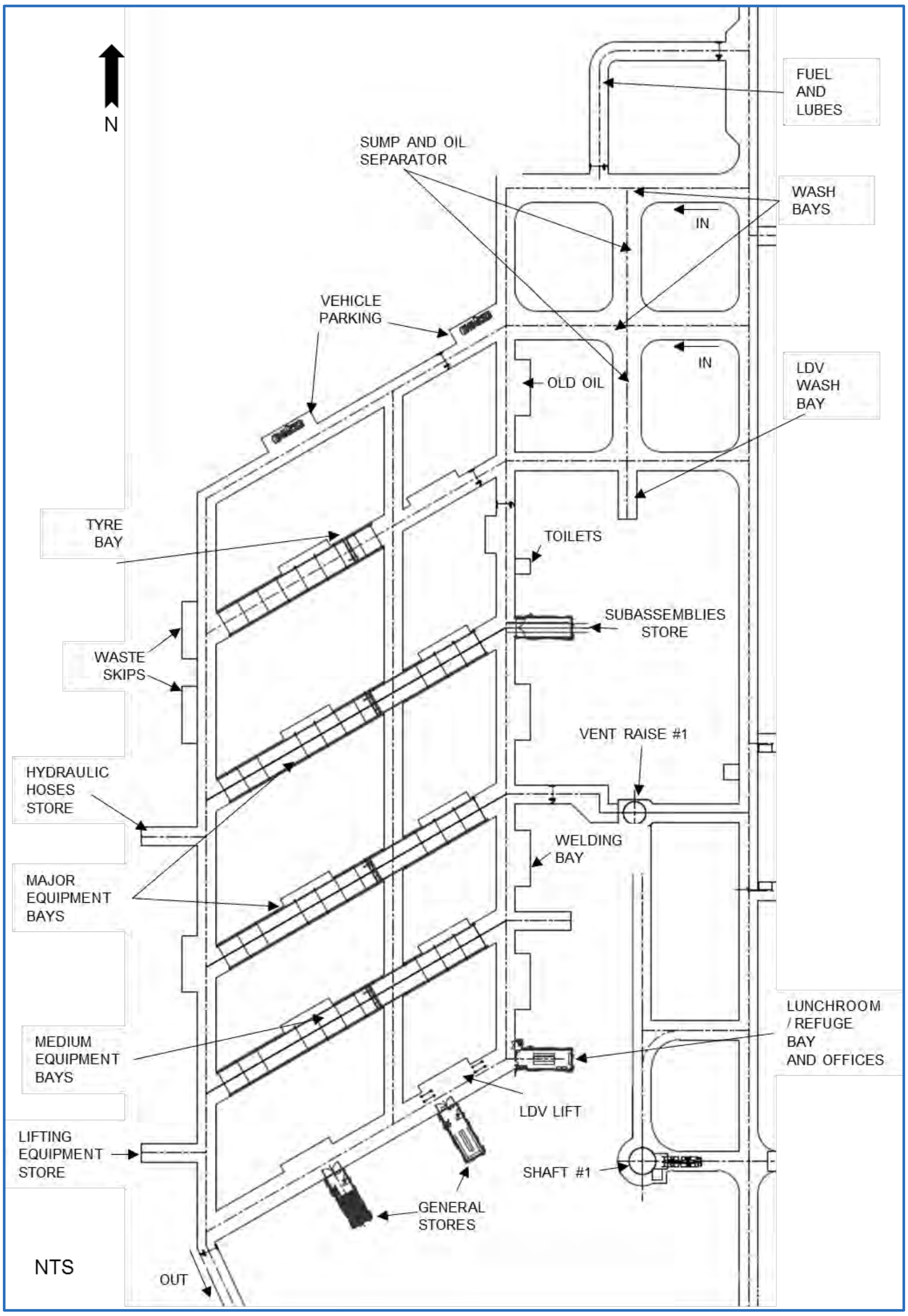
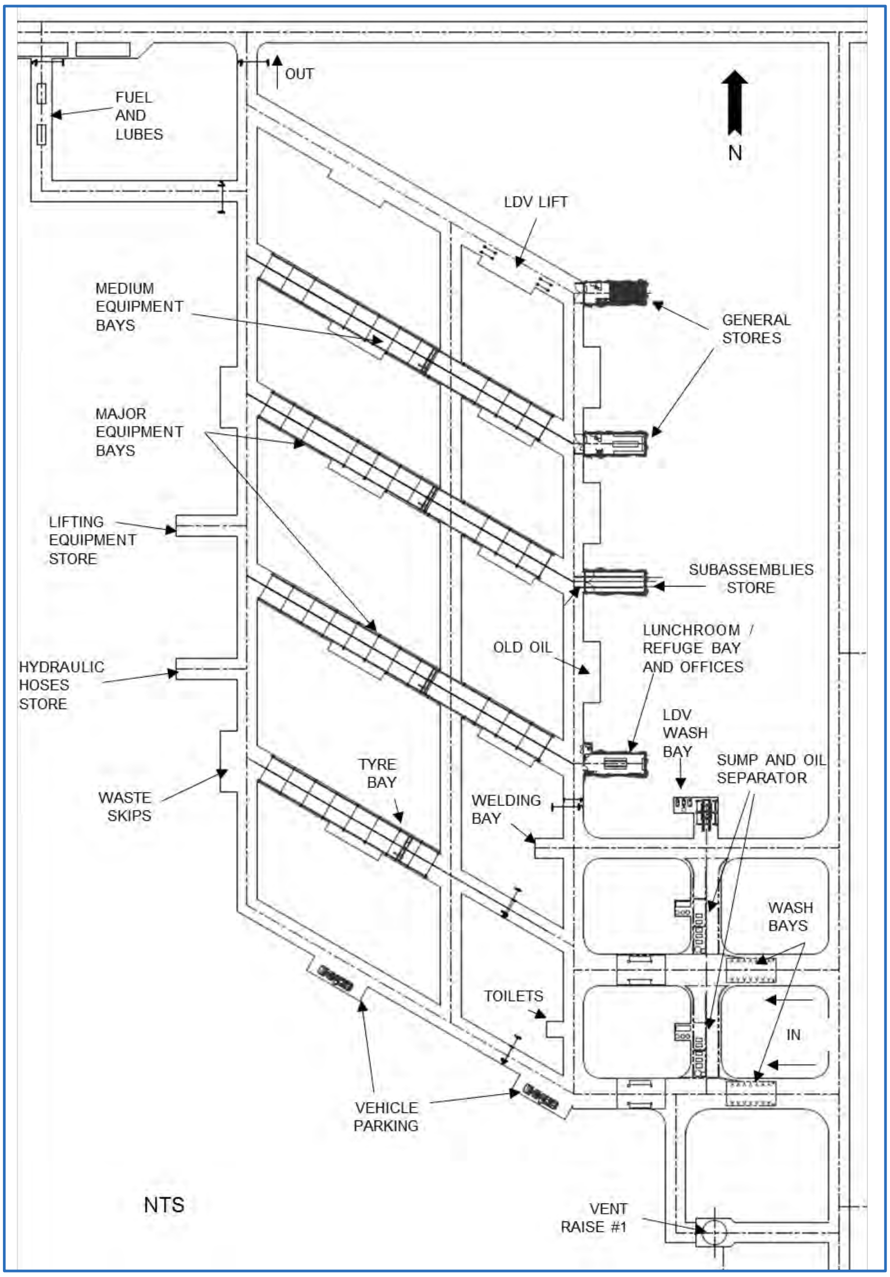


Figure 16.84 950 m Level Workshop



16.7.5.2 Refuge Stations and Emergency Egress

Refuge chambers will be required <750 m from any workplace, in accordance with South African regulations. Due to the absence of compressed air and the high degree of moveability, self sustaining type refuge bays will be used. A typical refuge chamber shown in Figure 16.85).

Figure 16.85 Typical Mobile Refuge Chamber

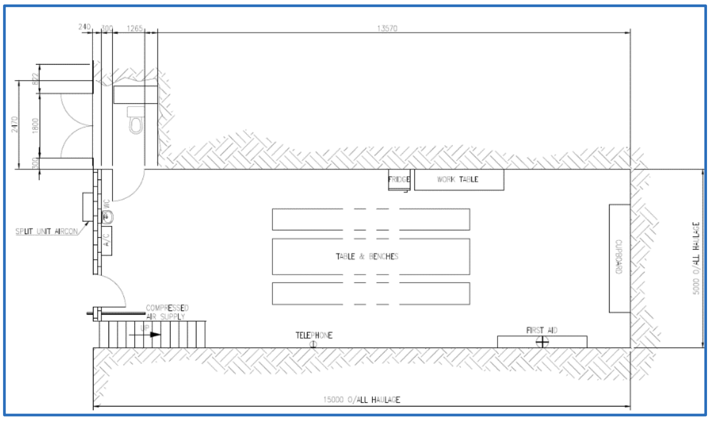
Mobile refuge chambers are completely self-sustaining and provide all basic life support systems required. Inside a standard refuge chamber a number of vital life support systems combine to create a safe and secure ongoing environment for occupants, including; oxygen supply, carbon dioxide (CO_2) and carbon monoxide (CO) scrubbing, cooling and gas monitoring.

These units require a weekly and monthly inspection from mine personnel and an annual inspection from the supplier. Mine personnel can also be trained to conduct the annual inspections and maintenance if required.

Workshop Refuge Chambers

The main workshops will have a lunchroom area that will be constructed and equipped as a refuge chamber. Figure 16.86 shows the layout of this refuge chamber. These chambers will be equipped with compressed air during Phase 2.

Figure 16.86 Workshop Refuge Chamber Layout



General emergency escape plans follow South African and United States codes. Mobile personnel hoisting is considered a safe and efficient means of escape for circumstances that may be encountered underground. Proven rubber tire hoisting systems using bullet style conveyances, which can be lowered into ventilation raise boreholes, can effectively extract otherwise trapped personnel.

The following South African regulations are followed:

- The two separate and independent shafts or outlets to surface required in terms of regulation 6.1.1.
- Shall not at any point be nearer to each other than 9.2 m.
- Shall be provided with proper arrangements, which will be kept constantly available for use, to enable persons to travel to and from the surface.
- Shall be maintained in a safe condition and at a sufficient cross sectional area throughout to allow for the free passage of persons.

Procedures for defining, evaluating, and reviewing the emergency escape system are part of the emergency escape strategy. Simulated emergency exercises (training) is conducted at the mine at regular intervals.

16.7.6 Toilet System

Portable toilets are located in strategic locations underground. A purpose built toilet facility is connected to each permanent refuge station at the underground workshops and sealable to the outside environment in the event of emergency. The toilets are serviced by a mobile effluent removal cassette and transported to surface for discharge.

16.7.7 Power Distribution

The bulk power supply is detailed in Section 18.7 of this report.

16.7.8 Power and Communication Systems

The main power consumers in area 2000 is set out in Section 18.7.

16.7.8.1 Emergency Generator Plant

Emergency power is provided by a 20 MVA 11 kV 50 Hz generator plant located on surface near the consumer substation.

To facilitate orderly evacuation of the mine; emergency loads are as follows:

- Ventilation fans located on surface at Ventilation raise 1, Ventilation raise 2, and Ventilation raise 3. One fan per ventilation raise station.
- Auxiliary Hoist
- Critical Communications

- Emergency lighting

16.7.9 Communications and Control Systems

16.7.9.1 Communications

- Primary: Leaky feeder with radios
- Secondary: Telephones
- Fibre optic cables installed in Shaft 1 and Shaft 2 and extended to all underground working areas.
- Communication and remote control to all process automation equipment.
- Personnel emergency dispatch system (PEDS) allows one way, mine wide emergency communication from surface to cap lamps equipped with the PEDS pager.
- Collision Avoidance System, (CAS), on all vehicles. This should include an asset tracking system that allows position monitoring of all personnel and vehicles in the mine. Communications to vehicles, trucks, LHDs, etc. for dispatch control (leaky feeder).
- Mine dispatch system for all vehicles, trucks, LHDs, etc. will be provided.
- Environmental monitoring system
- Geotechnical monitoring system

16.7.9.2 Control Philosophy

- Control of the mine process automation systems should be done via a centralised control room. HMIs will be located in the field for special cases and troubleshooting but it is intended that all control be coordinated from the control room.
- Control of all mining vehicles will occur in-situ within the vehicle or via tele-remote. Where available, vehicle telemetry should be communicated to the control room on surface

16.7.9.3 Centralised Blasting system

- A centralised blasting system will be used to remotely detonate blasts from surface once the shaft is classified as clear to blast.
- The system should be capable of initiating electronic detonators and shock tubes.
- The system will be a standalone system.
- Development ends will be blasted during the shift, from a safe location underground (Multi-blast conditions).

16.7.10 Materials and Personnel Logistics

16.7.10.1 Personnel logistics

- General

- Personnel will be batched on surface before entering the cage for the respective levels.
- UG the personnel will have designated travelling ways to keep them apart from other mobile equipment.
- Personnel will walk from the cage to a designated LDV and bus parking area from where they will be transported to their respective working areas.
- Workshop personnel will walk along designated walkways to the workshops.
- After dropping off personnel at their respective working places, the busses will be parked at the last drop off point until the end of shift.
- Parking for other equipment around the shaft will be limited to available passing bays around the area. The philosophy would require that drivers should leave their equipment on the levels near their working areas and wait for the busses to pick them up at end of shift.
- Busses and LDV's will transport personnel back to the shaft waiting areas where they will be batched for hoisting to surface at the correct time.

- Phase 1:
  - Skip will be swapped out with men and material cage at start and end of shift to hoist the bulk of labour force.

16.7.10.2 Material logistics

The personnel and material handling system design is based on the following criteria and assumptions:

- General
  - Materials are packaged into cassettes.
  - Bulk materials are packaged in hoppers or tote bags at their point of origin.
- Shaft 1 - Phase 1 operation:
  - Material will be palletised and loaded into cassettes that can be moved around by Multipurpose vehicle (Forklift attachment) into the cage.
  - A crawl structure will be installed to lift equipment and material as required in assembly bays located on each level
  - UV's will be used to distribute material to working areas
- Shaft 2 - Phase 2 operation:
  - Pallets/cassettes will be loaded onto a trailer that will be towed into the cage on surface.
  - Underground, at the different levels, a tractor will be in position to tow the full trailer out of the cage. As the full trailer is being towed out, a second tractor with an empty trailer will be used to tow in an empty trailer from behind.
  - The full trailer will be parked in the allocated material transfer bay on each level where an overhead crane will be installed to unload the trailer.

- UVs will drive into the material transfer area and load the cassettes for further distribution to the working areas underground.
- Mobile equipment and big pieces of equipment are assembled and tested on surface prior to underground transfer.
- The service cage has drive on, drive off capability.
- Large pieces of equipment may have to be broken down into smaller components for transport from the surface to the underground mine. An equipment assembly bay underground will be used to re assemble the equipment. This assembly bay will be positioned as close as possible to the shaft.
- Special arrangements are made for direct loading large pieces of equipment into the cage.
- Mine refuse and garbage from underground is brought up Shaft 2 in returning containers or cassettes.
- Mine refuse and garbage from the containers is delivered to the designated waste disposal pad for sorting and transfer to the proper Platreef project disposal area.

16.7.11 Warehouse Facilities and Laydown Areas

- There will be no centralised underground warehouse facilities. Separate underground storage facilities are located at the underground workshops, fuel and lube stations, explosives magazines, etc.
- Separate storage or laydown areas will be provided for items such as drilling supplies, ground support supplies, ventilation supplies, etc. at various locations in the mine, near points of use.
- Temporary storage of offloaded material will be within the material cassette transfer loop and the assembly bays.

16.7.12 Fuel and Lubricant Storage and Dispensing

- Fuel storage tanks are located at a central facility on the surface.
- Total fuel storage capacity underground is limited to three days' supply.
- Bulk fuel is transferred underground in 5,000 mL batches by a pipeline in Shaft 1.
  - Fuel is transferred from surface batch tank to storage tanks located on 750 m Level, 850 m Level and 950 m Level.
  - Fuel is dispensed at re fuelling stations located on 750 m Level, 850 m Level and 950 m Level.
  - Lubricants are brought down the shaft in stackable storage and dispensing tanks (totes). Used fluids are transferred out of the mine in similar totes.
  - Lubricants are dispensed at fuel and lube stations.
  - Storage and measuring tanks are enclosed in concrete containment basins to contain spillage.

- Fire suppression systems are installed at all fuel storage sites.

16.7.13 Explosives Storage Facilities

- Production: Emulsion with electronic detonation fully monitored and controlled.
- Development: Emulsion with shock tubes.
- Explosives and explosives accessories are packaged in containers on the surface and delivered to the underground explosives storage facilities on the 850 m Level and 950 m Level.
- The storage facility includes separate emulsion, cap, and powder magazines.
- Explosives trucks (LDV's) transport the detonators, and other blasting supplies from the magazines to the points of use.
- Emulsion will be stored on surface in emulsion silos that will be converted to a vertical drop system during Phase 1 steady state production.

16.7.14 Sewage and Effluent Management

Portable toilets to be provided throughout mine and a mobile service unit will be provided for cleaning toilets.

16.8 Mining Opportunities

The following is a list of potential mining opportunities for Platreef:

- Continued optimisation of the stope design and sequencing to improve the grade profile during the early years of production.
- Remote operation of the fixed rock breakers at ore and waste truck dumps.
- Placing a small percentage of waste rock into primary stopes.
- Evaluating batch design to optimise cement and admixture usage minimising fill curing rates.
- Storing and hoisting development ore during the Shaft 1 hoisting phase.
- Reducing the angle of the footwall and hanging wall in production stopes as a benefit of the definition drilling programme.
- Reviewing the use of a small raisebore or a drill similar as an alternative to drilling and blasting the initial slot raise.
- Reducing the number of individuals supporting the development crews (e.g., service crews, materials handlers, operator assistants).
- Replacing the drop raise in Longhole Stopes with an Atlas Copco Easer to increase overall production of the stope.

16.8.1 Risks

The following is a list of potential mining risks for Platreef:

- Lack of available workforce with sufficient skills to meet the specified performance rates.
- Production and schedule constraints due to shaft pillar designs.
- Possible requirement of replacement ore and waste passes; the design provides for locations, but they are not currently in the design.
- Uncertainty of raise boring to 750 m in length and deeper in this location.
- The amount of lining required in the ventilation raises once they are reamed.
- Mining through the Tshukudu Fault.
- Timeliness of the definition drilling during the preproduction and early production stages.
- Cooling capacity during the summer peaks. This can be managed with a work-rest regime and supplemental underground air conditioning units.
- Mining underneath cemented paste fill or rock fill in the sill pillars (although less than 1% of the tonnage is mined under a sill pillar).
- The amount of additional S3 support that may be required in the later years of the mine life if ground conditions deteriorate and mining impacts are more severe.
- Difficulty handling the low-grade material during development.
- The stability of the accesses in the secondary stopes during and after the mining of the primary stopes.
- The impact on the Platreef site of any social uncertainty within South Africa during the project’s life.
- The amount of additional S3 support that may be required in the later years of the mine life if ground conditions deteriorate, and mining impacts are more severe.
- Difficulty handling the low-grade material during development.
- The stability of the accesses in the secondary stopes during and after the mining of the primary stopes.
- The impact on the Platreef site of any social uncertainty within South Africa during the project's life.

16.8.2 Recommendations

Mining recommendations:

- Optimise the definition drilling programme required for the initial mining areas.
- Determine stope sizes and footwall and hanging wall angles once the block sizes in the block model are reduced due to more detailed definition drilling.
- Monitor fragmentation during the development stage to eliminate the need for secondary breaking on development rock.

- Maximise the flow of ventilation from Shaft 1 (cooled air) to the deepest and warmest mine workings.
- Monitor and optimise the first development through the Tshukudu Fault.
- Develop an operating procedure to allow waste rock to go into primary Longhole Stopes and Drift-and-Fill areas.
- Set up a programme for ore pass monitoring to ensure the longevity of the passes.
- Set up a ground control observation programme to proactively recondition the ground support as needed during the mine life.

17 RECOVERY METHODS

17.1 Summary

The 2017 Platreef FS was based on the development of a large scale, mechanized, underground mine accessed via two vertical shafts, with a processing plant and associated infrastructure. The study was based on a process plant with nominal throughput of 4.0 Mtpa and a design maximum of 4.4 Mtpa (2 off 2.2 Mtpa modules).

Based on the outcome of the 2020 PEA, the current Platreef 2022 FS is based on constructing an initial concentrator plant with a nominal throughput of 0.70 Mtpa and a design throughput of 0.77 Mtpa to treat un-crushed RoM material hoisted through Shaft 1. This will be followed by the addition of the 2 off 2.2 Mtpa concentrator modules as per the original 2017 FS to process primary crushed RoM material hoisted through shaft 2. The Platreef FS is thus based on phased production ramp up from 0.70 Mtpa to 5.2 Mtpa.

The process plant for the Platreef 2022 FS is designed to treat 0.77 Mtpa and 4.4 Mtpa Platreef ore in Phase 1 and 2 respectively. The plants include all processing requirements from the run-of-mine (ROM) storage through to final concentrate plant load out and tailings disposal.

The process design for the respective 0.77 Mtpa and 4.4 Mtpa concentrator plants have been developed based on the testwork findings and assessments, various desktop level trade-off studies and relevant DRA design information. The Platreef concentrator flow sheet is based on a conventional three-stage crushing and ball milling circuit followed by flotation. This single-stage milling and flotation process flow sheet is well known in industry, and has historically been proven as a suitable processing route for various platinum ores. The process flow sheets are summarised in Figure 17.1 and Figure 17.2.

The concentrator engineering design is based on previously constructed and proven DRA unit processes adapted for Platreef-specific requirements. Project-specific design criteria and specifications were developed to ensure conformity across the mine site and various contractors.

The engineering design has taken cognisance of the environmental and social impacts with regard to noise, dust, light and visual pollution. Dust extraction and suppression systems have been included to minimise dust. Dust suppression is present on all ROM transfer points. Silos are specified with concrete roofs and fitted with dust extraction and filtration units. Noise-generating equipment was identified, independently simulated and noise-attenuating cladding designs were conducted. The Phase 2 conveyors are sheeted to minimise noise, dust and light pollution.

The final concentrator layouts were decided taking into consideration the environmental and social impacts and aimed to minimise transfer and wear points in the ROM section, reduce building height and improve constructability and maintainability in the main concentrator area.

The concentrator plant has been designed in accordance with the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities.

Figure 17.1 Phase 1: 0.77 Mtpa Concentrator Flow Sheet

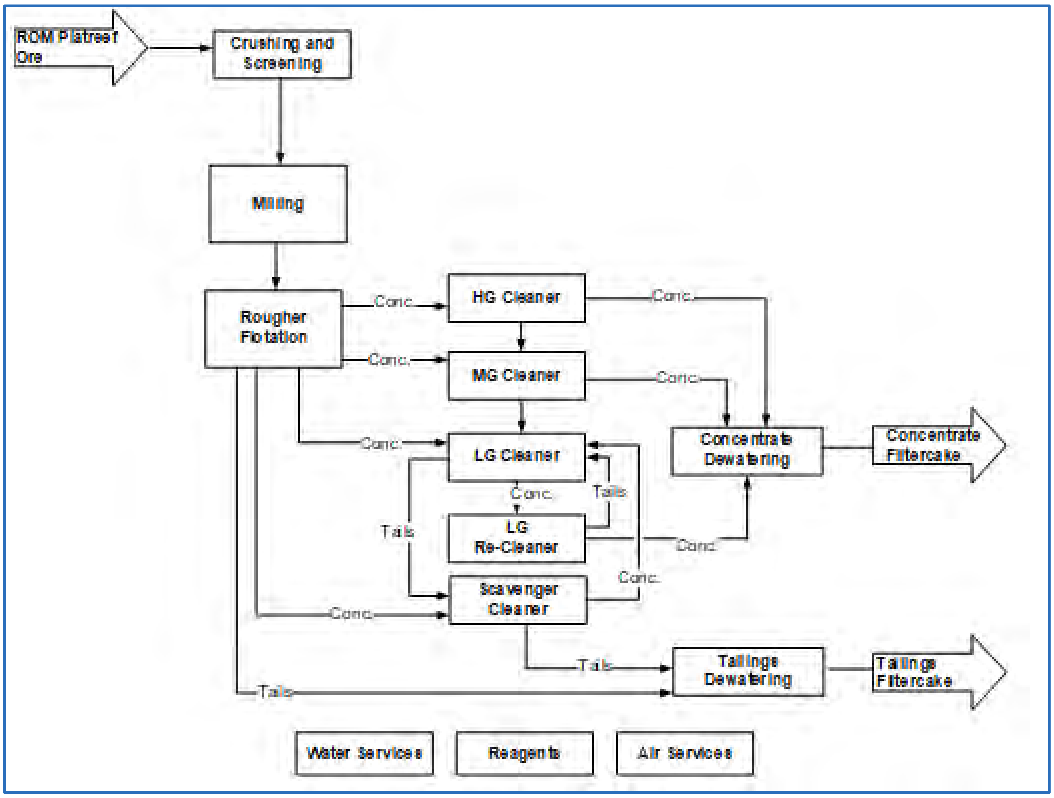


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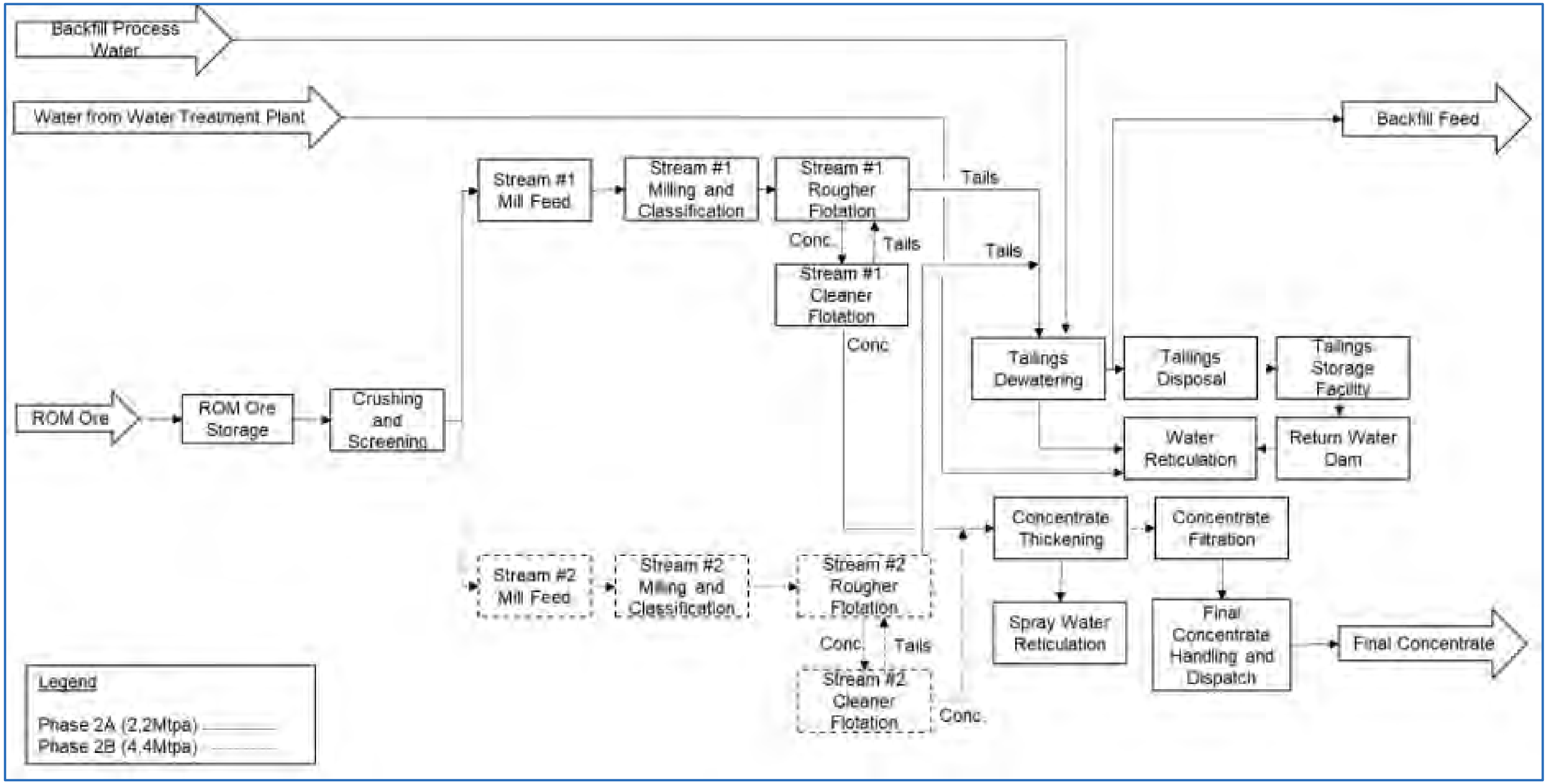
Figure 17.2 Phase 2: 4.4 Mtpa Concentrator Flow Sheet



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17.2 Introduction

The phased approach to the construction will see the construction of a 0.77 Mtpa concentrator plant in the Phase 1 of the project. This will be followed by the construction of a 4.4 Mtpa plant as 2 x 2.2 Mtpa modules in Phase 2 to match the mine ramp up profile.

The concentrator plants have been designed to treat Platreef ore and include all ore processing requirements from the Primary Crusher ROM Tip through to a final concentrate load out and tailings disposal. Each concentrator consists of the following:

- Crushing and screening circuit
- Milling circuit
- Flotation circuit
- Concentrate handling circuit
- Tailings de-watering and disposal circuit
- Reagent make-up and distribution facilities
- Utilities which include air and water distribution
- Plant infrastructure which includes a control room, various substations, workshops, and a warehouse.

This Section outlines the concentrator process design basis as well as the engineering design.

17.3 Concentrator Process Design

The previous 2017 FS had concluded that a two-phased production approach for the 4.4 Mtpa concentrator was the most suitable approach. This phased approach was retained for the current Platreef 2022 FS.

The Platreef 2017 FS was based on a processing rate of 4.0 Mtpa, aligned to the mine plan and schedule at the time. The process plant, however, was adequately sized to treat a maximum of 4.4 Mtpa (2 x 2.2 Mtpa modules) as stated in the process design criteria. This higher processing rate has been utilised in the Platreef 2022 FS. All equipment, inclusive of major process items i.e. mills, flotation cells, thickeners and filters, has been designed to accommodate a maximum throughput of 4.4 Mtpa.

The Platreef concentrator design is based on a conventional three-stage crushing and ball milling circuit followed by flotation. This single-stage milling and flotation (MF1) process flow sheet is well known in industry, and has historically been proven as a suitable processing route for various platinum ore types containing significant concentrations of base metal sulfides. The key aspects of the concentrator plant design are discussed below. The concentrator design criteria is summarised in Table 17.1.

Table 17.1 Summary of Concentrator Design Criteria

Criteria	Units	0.77 Mtpa Concentrator	4.4 Mtpa Concentrator
Production Summary			
Mining			
Plant Feed Year 1	Mt	0.20	
Plant Feed Year 2 - 4	Mt	0.70 – 0.77	
Plant Feed Year 5	Mt	1.89	
Plant Feed Year 6		2.97	
Plant Feed Year 7		4.39	
Plant Feed Year 8+		5.17	
Life of Mine	Years	29	
Plant Throughput			
Design Throughput Phase 1	Mtpa (dry)	0.77	-
Design Throughput Phase 2A	Mtpa (dry)	0.77	2.20
Design Throughput Phase 2B	Mtpa (dry)	0.77	4.40
Design Mass Pull	%	6.3	4.7
Head Grades			
Platinum	g/t	1.69 – 2.79	1.69 – 2.34
Palladium	g/t	1.72 – 2.67	1.72 – 2.44
Rhodium	g/t	0.11 – 0.20	0.11 – 0.17
Gold	g/t	0.28 – 0.41	0.28 – 0.34
3PE+Au	g/t	3.84 – 6.06	3.84 – 5.25
Copper	%	0.15 – 0.19	0.15 – 0.18
Nickel	%	0.32 – 0.41	0.32 – 0.36
Overall Recovery			
3PE+Au recovery	%	84-90	
Nickel recovery	%	69–78	
Copper recovery	%	86–90	
Concentrate Grades			
3PE+Au grade target	g/t	85	
Nickel grade target	%	4.3–6.2	
Copper grade target	%	2.5–3.9	
Crushing Operating Schedule			
Operating days per annum	days	365	365

Criteria	Units	0.77 Mtpa Concentrator	4.4 Mtpa Concentrator
Operating shifts per day	number	2	2
Hours per shift	h	12	12
Availability	%	67	74
Utilisation	%	94	94
Crushing Circuit Running Time	%	63	70
Overall Running Time	h/annum	5,131	6,115
Design Circuit Feed Rate	t/h (dry)	150	725
Milling and Flotation Operating Schedule			
Operating Days per Annum	days	365	365
Operating Shifts per Day	number	2	2
Hours per Shift	h	12	12
Availability	%	94	94
Utilisation	%	98	98
Milling Circuit Running Time	%	92	92
Overall Running Time	h/annum	8,000	8,000
Number of Modules	number	1	2
Circuit Feed Rate	t/h/module (dry)	96.3	275
Milling and flotation modules Phase 1A	number	1 x 0.77 Mtpa	-
Milling and flotation modules Phase 2A	number	1 x 0.77 Mtpa	1 x 2.2 Mtpa
Milling and flotation modules Phase 2B		1 x 0.77 Mtpa	Phase 1A + 2.2 Mtpa

17.3.1 ROM Handling

The Phase 1, 0.77 Mtpa concentrator plant crushing circuit receives ROM from underground mining at a top size of 600 mm based on the underground blast fragmentation expected particle size.

The Phase 2, 4.4 Mtpa concentrator plant crushing circuit receives primary crushed ROM from underground mining at a top size of 270 mm based on the underground crusher expected particle size.

For both the 0.77 Mtpa and 4.4 Mtpa a conventional three-stage crushing, and screening circuit was selected, operating in closed circuit to produce a mill feed size of approximately 13 mm. A 0.77 Mtpa, modular three-stage crushing, and screening circuit was selected for Phase 1. For the 4.4 Mtpa concentrator plant, the first stage of crushing (primary crushing) is included within the underground mining scope and the design includes a new secondary and tertiary crushing and screening circuit on surface.

17.3.2 Milling

The mills have been sized to cater for the 85th percentile ore hardness as determined from the comminution variability test data. Each milling circuit is fed with 13 mm tertiary crushed material and will produce a product size of 80% passing 75 µm.

The Phase 1 milling circuit is comprised of a single 0.77 Mtpa ball mill and cyclone cluster operated in closed circuit.

The recommended mill size for Phase 1 is an 18'ftØ x 24'ft effective grinding length (EGL) ball mill with a grate discharge liner arrangement and 4.5 MW motor. The Phase 1 mill motor is fixed speed with modifications to the ball charge allowing for variation in the feed material, required grind size and mill throughput.

For the 4.4 Mtpa concentrator, two separate 2.2 Mtpa milling circuits were selected based on a plant ramp-up and comminution trade-off and concentrator production ramp-up profile. Each milling circuit is comprised of a single 2.2 Mtpa ball mill and cyclone cluster operated in closed circuit.

The recommended ball mill size for each 2.2 Mtpa milling stream is 22½'ft diameter x 34½'ft Effective Grinding Length (EGL) with a grate discharge liner arrangement and 12 MW (2 x 6 MW) variable speed geared pinion drives. The design has been based on a maximum ball charge of 35%. The ball charge, together with the variable speed motor capability, allows feed material variability, required grind size and mill throughput. A summary of the expected ball mill performance when treating ore of the 85th percentile hardness, is presented in Table 17.2.

Table 17.2 Summary of Ball Mill Performance Criteria

Throughput Scenario	Ball Charge	Treatment Rate per Mill (dry tph)	Design Power Draw (kW)
Phase 1: 1 x 0.77 Mtpa	32%	96.3	4, 280
Phase 2: 2 x 2.2 Mtpa	35%	275	10,713

Testwork conducted as part of the Platreef 2022 FS indicated that the flotation circuit response was sensitive to the grinding media type used. It was noted that the flotation performance improved when using high chrome media (Cr>16%) as compared to carbon steel media. Based on the testwork findings, the design and operating cost estimates have been based on wear estimates and costs for wear-resistant high chrome (Cr>18%) grinding media. High chrome (Cr>18%) grinding media are widely available in South Africa and costs approximately 30% more than forged mild steel grinding media.

In addition to using high chrome grinding media, the mill liners made of 18% high chrome steel have been specified, and all piping in the milling circuit will be rubber lined.

The Platreef FS testwork programme included semi-pilot scale HPGR test work which was used to assess the potential inclusion of an HPGR circuit as alternative to the tertiary crushing stage.

This assessment indicated that an HPGR circuit offered no benefit at the low throughput rate for the 0.77 Mtpa Phase 1 concentrator plant.

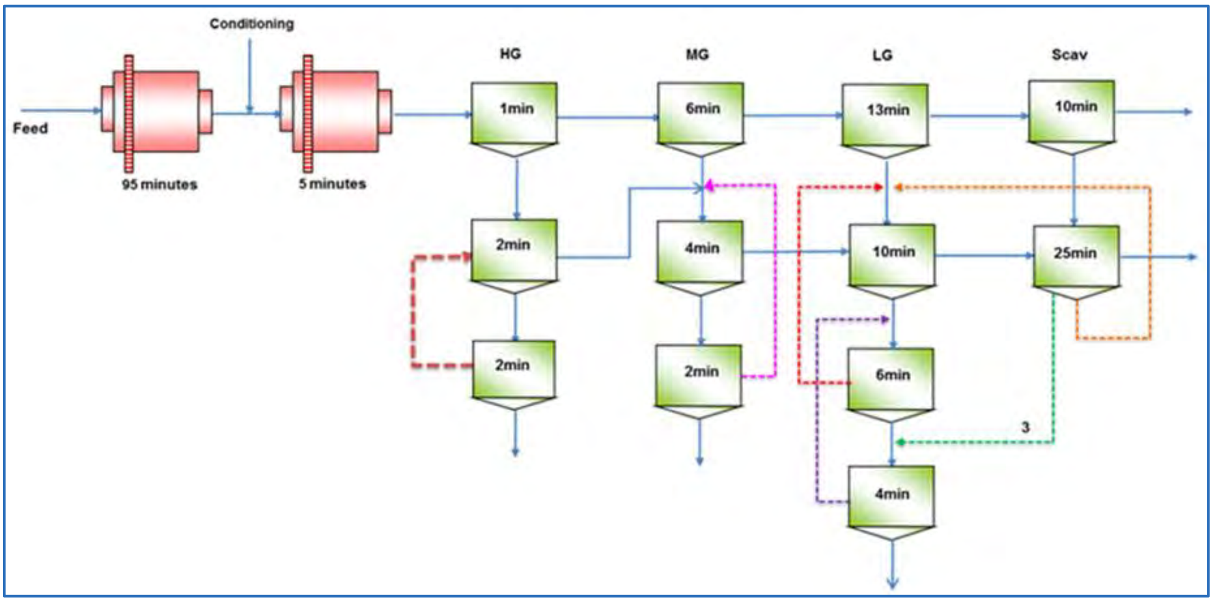
For the larger 4.4 Mtpa Phase 2 concentrator, a high-level trade-off, which considered differential capital and operating costs, was concluded. This assessment indicated that an HPGR circuit would offer the potential for an approximate 7% operating cost saving for the crushing and milling circuits.

The option of an HPGR circuit for the 4.4 Mtpa concentrator was not included in the current Platreef 2022 FS design however it is noted as a potential optimization opportunity. This option will be considered in more detail during the phased implementation programme.

17.3.3 Flotation Circuit

The optimised flotation circuit flow sheet as developed during testing was used as the basis of design. This optimised flow sheet is based on a conventional platinum ore reagent suite with the inclusion of a targeted copper collector, using a circuit configuration as presented in Figure 17.3. The cleaner flotation circuit design is based on treating the fast, medium and slow floating fractions in separate cleaner circuits. This is commonly referred to as a rate matching or split cleaner configuration in the South African platinum industry.

Figure 17.3 Optimised Flotation Flow Sheet as Derived from Testwork



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A single 0.77 Mtpa flotation circuit was selected for the Phase 1 concentrator and two separate 2.2 Mtpa flotation circuits were selected for Phase 2 based on the production ramp-up and comminution trade-off studies conducted during the 2017 FS.

Preliminary mini pilot plant commissioning runs have indicated that an SIBX reagent is a potential alternative to the copper collector reagent suite. This was also noted during the original flowsheet development phase but was not adopted due to froth stability concerns. The mini pilot plant commissioning runs did, however, not show evidence of poor froth stability highlighting that this opportunity should not be completely disregarded. The inclusion of an SIBX reagent make-up and dosing facility during the 0.77 Mtpa concentrator detailed design phase would allow for this option to be trialled during the full-scale concentrator ramp-up and optimization phase.

The flotation testwork conducted at three independent laboratories during the Platreef 2015 PFS campaign highlighted that the flotation feed Eh is negative when the sample is milled using mild steel grinding media. A positive Eh prior to flotation was achieved by using high chrome grinding media. The milling circuit design is based on high chrome grinding media and mill liners. All mild steel piping within the slurry plant is specified to be epoxy lined to minimise any slurry contact.

17.3.4 Concentrate Thickening and Filtration

The concentrate thickening and filtration circuit design for each phase is based on the installation of a single concentrate thickener and horizontal plate filter to treat the combined concentrate from the flotation circuit.

Based on the MO test work in combination with benchmarked data, a design unit area thickening rate of 0.052 t/h/m^2 and a design filtration rate of 358 kg/h/m^2 is recommended as the basis Platreef 2022 FS. Based on the testwork in combination with benchmark data, it is expected that a filter cake moisture content of 12 - 14% (w/w) can be achieved.

The concentrate de-watering equipment, as sized for the Platreef 2022 FS, is considered adequate for the required duty. It is however noted that there is the potential to reduce the size of the 0.77 Mtpa concentrate dewatering equipment as a potential cost saving opportunity. This will be considered during the project implementation phase.

17.3.5 Tailings Dewatering and Transfer

The initial 0.77 Mtpa process plant will have a tailings filtration circuit with the filtered product reporting to a dry stacking facility and will continue to produce a filtered tailings product when the production rate is increased to 5.2 Mtpa through the addition of the 4.4 Mtpa Phase 2 process plant. The TSF design has been updated to reflect a dry-stack tailings facility. During Phase 1 (0.77 Mtpa) tailings filter cake will be trucked to the dry stack TSF facility. In Phase 2 tailings from the 0.77 Mtpa will be sent to backfill and the combined thickened tailings from the 4.4 Mtpa concentrator will be pumped to backfill or alternately to the TSF for further de-watering.

The backfill plant will operate intermittently depending on mining backfill requirements. The design of the backfill plant is based on treating approximately 56% of the concentrator tailings over LoM. A Drift-and-Fill mining method will be employed for the initial 0.77 Mtpa mining production and backfill will be achieved using Cemented Rock Fill (CRF). No tailings from the concentrator plant will be required for the backfill operations during this time. Once the production rate is increased with the phased addition of the 4.4 Mtpa concentrator mining will be via transverse Long-Hole Open Stoping (LHOS) and the backfill will be achieved using Cemented Paste Fill (CPF). Tailings from the concentrator plants will be used to make-up the CPF mix.

The 0.77 Mtpa concentrator tailings dewatering, and disposal circuit includes conventional thickening and vacuum filtration. A 20 m diameter tailings thickener has been selected to treat guard cyclone overflow, with a target underflow solids concentration of 55 to 60% (w/w). A 145 m^2 horizontal vacuum belt filter has been selected for the tailings filtration duty based on a filtration rate of 650 kg/h/m^2 and a target tailings filter cake solids concentration of >85% (w/w). Filtered tailings will report to either the TSF or alternately to the backfill plant which includes a re-claim hopper for processing the filtered tailings.

A high-level trade-off was conducted to assess the potential benefit of a vacuum disk filter for the 0.77 Mtpa concentrator tailings filter duty. These assessments indicated that a vacuum disk filter option offered the potential for approximately 25% saving in both capital and operating cost for this circuit. This saving opportunity has not been included in the current design and will be evaluated during the project implementation phase.

The 4.4 Mtpa concentrator tailings dewatering and disposal circuit design is based on the installation of a single tailings thickener and clarifier, multiple stage tailings disposal pump trains and booster station to treat the combined tailings from both the 2.2 Mtpa flotation modules. The tailings dewatering and disposal circuit includes conventional thickening and clarification of thickener overflow. There is also allowance for brine disposal to the TSF via the tailings disposal tank.

The tailings from each 2.2 Mtpa module reports to the tailings thickener via a guard cyclone. A 35 m diameter tailings thickener has been selected to treat guard cyclone overflow, with a target underflow solids concentration of 57–60% (w/w). The design also includes a 35 m diameter clarifier to remove any solids from the thickener overflow. The achievable thickener underflow density requires confirmation based on thickening testwork for a classified feed. This confirmatory thickening testwork will be conducted ahead of detailed design.

Three tailings pipelines and pump trains will be installed for pumping the 4.4 Mtpa concentrator tailings slurry to the TSF, namely a single 2.2 Mtpa line and two lines (duty/standby) for the combined 4.4 Mtpa capacity. The tailings line is approximately 8 km long, and there is an elevation difference of 126 m between the plant and the TSF. The tailings pumping system design requires a booster station pumping system located at a distance of approximately 5 km from the concentrator plant.

During operation of the backfill plant, the 4.4 Mtpa concentrator plant will not pump tailings to the tailings facility, however, when the backfill plant is offline, the concentrator plant will need to process tailings at 100% of mill capacity. Thus, the tailings thickening and disposal system has a maximum capacity of 275 dry tph. The intermittent use of the tailings disposal system incorporates the use of a flushing system to the TSF when the switchover to backfill occurs. This is to minimise settling of solids in the pipeline due to intermittent interruptions. The process water from the backfill plant will be returned to the process plant tailings thickener to be used as process water within the concentrator plant.

17.4 Concentrator Plant Production Schedule

The concentrator plant production schedules for the Platreef 2022 FS showing feed tonnes and grade, recoveries and metal production is presented in Table 17.3 and Table 17.4.

Table 17.3 Process Production Plant Feed

	Year Number	Total	–2	–1	1	2	3	4	5	6	11	21
	Year To									10	20	LOM
Concentrator Feed	kt	125,212	–	–	190	700	700	735	1,870	22,825	51,700	46,493
NSR BDT20	$/t	156	–	–	210	200	215	190	178	168	158	146
Platinum	g/t	1.94	–	–	2.63	2.43	2.80	2.35	2.26	2.11	1.95	1.81
Palladium	g/t	1.99	–	–	2.65	2.63	2.68	2.29	2.25	2.22	2.01	1.81
Gold	g/t	0.30	–	–	0.40	0.35	0.40	0.39	0.34	0.30	0.30	0.30
Rhodium	g/t	0.13	–	–	0.17	0.18	0.20	0.15	0.15	0.15	0.13	0.12
Copper	% Cu	0.16	–	–	0.19	0.19	0.19	0.19	0.18	0.17	0.17	0.16
Nickel	% Ni	0.34	–	–	0.41	0.40	0.39	0.39	0.36	0.34	0.34	0.33
Sulfur	% S	0.82	–	–	1.01	1.00	0.95	0.96	0.88	0.84	0.81	0.80
3PE+Au	g/t	4.37	–	–	5.86	5.58	6.08	5.18	5.00	4.78	4.39	4.04

Note: NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

Table 17.4 Concentrate and Metal Production

Year Number		Total	–2	–1	1	2	3	4	5	6	11	21
Year To										10	20	LOM
Concentrator Recoveries												
Platinum	%	87.23	–	–	88.90	90.50	90.90	90.13	89.00	88.66	87.28	85.94
Palladium	%	86.76	–	–	88.70	90.30	90.70	89.91	88.78	88.37	86.85	85.21
Gold	%	78.54	–	–	78.80	80.43	80.80	80.38	79.47	79.48	78.66	77.77
Rhodium	%	80.28	–	–	82.90	84.51	84.90	84.00	82.78	82.16	80.35	78.45
Copper	%	87.70	–	–	88.87	90.01	90.09	90.34	88.65	87.77	87.79	87.31
Nickel	%	71.58	–	–	77.11	77.82	77.37	77.54	73.39	72.03	71.89	70.48
Sulfur	%	70.79	–	–	73.94	75.24	74.40	74.64	71.99	71.31	70.64	70.28
3PE+Au	%	86.21	–	–	87.93	89.59	89.95	89.12	88.07	87.74	86.29	84.78
Mass Pull	%	–	–	–	7.06	5.90	6.43	5.44	5.18	4.94	4.46	4.03
Concentrate Produced	kt	5,545	–	–	13	41	45	40	97	1,128	2,308	1,874
Concentrator Grade												
Platinum	g/t	38.21	–	–	33.14	37.26	39.58	38.99	38.86	37.86	38.12	38.52
Palladium	g/t	39.00	–	–	33.21	40.21	37.76	37.93	38.57	39.80	39.23	38.34
Gold	g/t	5.32	–	–	4.52	4.75	5.06	5.73	5.21	4.83	5.23	5.75
Rhodium	g/t	2.43	–	–	2.04	2.52	2.59	2.34	2.36	2.52	2.42	2.39
Copper	% Cu	3.26	–	–	2.44	2.88	2.61	3.14	3.05	2.95	3.26	3.50

	Year Number	Total	–2	–1	1	2	3	4	5	6	11	21
	Year To									10	20	LOM
Concentrator Recoveries												
Nickel	% Ni	5.44	–	–	4.46	5.27	4.70	5.61	5.07	4.98	5.47	5.71
Sulfur	% S	13.09	–	–	10.62	12.75	11.01	13.22	12.25	12.13	12.91	13.99
3PE+Au	g/t	84.97	–	–	72.91	84.74	85.00	85.00	85.00	85.00	85.00	85.00
Recovered Metal												
Platinum	koz	6,813	–	–	14	49	57	50	121	1,373	2,828	2,320
Palladium	koz	6,954	–	–	14	53	55	49	120	1,443	2,910	2,310
Gold	koz	948	–	–	2	6	7	7	16	175	388	346
Rhodium	koz	433	–	–	1	3	4	3	7	91	180	144
Copper	klb	399,108	–	–	721	2,622	2,590	2,765	6,505	73,389	165,951	144,567
Nickel	klb	664,740	–	–	1,317	4,794	4,667	4,940	10,823	123,798	278,340	236,060
Sulfur	klb	1,599,885	–	–	3,136	11,604	10,932	11,647	26,131	301,690	656,786	577,957
3PE+Au	koz	15,149	–	–	31	112	123	109	264	3,082	6,306	5,120

17.5 Process Description

The process descriptions for the Platreef 2022 FS are based on the detailed process flow diagrams (PFDs) that include mass balances. A summary block flow diagram for the 0.77 Mtpa and 4.4 Mtpa concentrator plants are presented in Figure 17.1. and Figure 17.2.

17.5.1 ROM Handling

Phase1: 0.77 Mtpa Concentrator

Un-crushed Run-of-Mine (ROM) with a F_{100} top size of 600 mm and a F_{95} size of 400 mm will be routed to a ROM stockpile.

Upon commencement of the concentrator plant operations, this stockpiled material will be reclaimed via a Front-End Loader (FEL) and fed to a modular crushing circuit.

Phase2: 4.4 Mtpa Concentrator

ROM is crushed underground to a top size (F_{100}) of 270 mm. The pre-production ROM material is conveyed by a ROM handling system and can be routed to either one of the two 5,200 t ROM silos or the ROM stockpile.

Upon commencement of the concentrator plant operations, this stockpiled material will be reclaimed via a ROM reclaim system consisting of a static grizzly, surge bin, vibrating screen and reclaim conveyor fitted with a magnet for tramp metal removal. The reclaim conveyor feeds the secondary screens.

Once the concentrator plant is in production, ROM ore is conveyed from Shaft 2 headgear, at a peak flow rate of 1,350 dry tph and a top size of 270 mm, into either one of the two 5,200 t ROM silos. Each ROM silo is equipped with apron feeders for extraction onto the ROM Screen Feed Conveyor that feeds the secondary screens in the crushing circuit. Tramp metal is removed prior to crushing by means of a tramp metal magnet situated on the conveyor.

17.5.2 Crushing and Screening

Phase1: 0.77 Mtpa Concentrator

The crushing, and screening circuit consists of a modular crushing and screening plant which includes primary, secondary and tertiary crushing. The secondary and tertiary crushers are operated in closed circuit with a crusher circuit screen.

A Front-End Loader (FEL) feeds material from the RoM stockpile into a RoM Bin, which is fitted with a static grizzly to remove any +500 mm ahead of the primary Jaw Crusher.

The Jaw Crusher is fed by a vibrating grizzly feeder. Grizzly feeder oversize is fed to the primary crusher and the combined product (grizzly feeder undersize and crusher product) is conveyed to the Crusher Circuit Screen by means of the Jaw Crusher Discharge Conveyor. Screen top deck oversize (+36 mm) is conveyed to the secondary cone crusher which has a dedicated feed bin and vibratory feeder. Screen bottom deck oversize (+13 mm) is conveyed to the tertiary cone crusher which has a dedicated feed bin and vibratory feeder.

The secondary and tertiary crusher products are combined with the primary crusher product as feed to the Crusher Circuit Screen. The primary crusher product conveyor is fitted with a magnet for removal of tramp metal.

The Circuit Screen Undersize Product Conveyor transfers the final crusher circuit undersize product material (P_{100} =13 mm) to the overland transfer conveyor which conveys the crusher product to the 1,600 t mill feed silo.

Phase2: 4.4 Mtpa Concentrator

The concentrator crushing and screening circuit consists of secondary and tertiary crushing stages in closed circuit with secondary and tertiary screening stages.

Ore from the ROM Screen Feed Conveyor, ROM Reclaim Conveyor, and Secondary Crusher Product Conveyor are transferred to the Secondary Screen Feed Bin. Ore from the Secondary Screen Feed Bin is transferred via vibratory feeders to the secondary screening circuit which is comprised of two double-deck screens. The top deck (+30 mm) of both the secondary screens reports to the Secondary Cone Crusher Feed Conveyor which is equipped with a magnet for the removal of tramp metal. The bottom screen deck (+12 mm–30 mm) material of both the screens reports to the Tertiary Cone Crusher Feed Conveyor, which is also equipped with a magnet for the removal of tramp metal. The secondary screen undersize (–12 mm) from both secondary screens reports to the Screen Product Conveyor.

The Secondary Cone Crusher circuit is comprised of two cone crushers operated in parallel. Each cone crusher is fed via a feed bin and vibrating feeder. The secondary crusher product (–35 mm) is removed by the Secondary Cone Crusher Product Conveyor and fed back to the Secondary Screen Feed Bin for re-screening.

The Tertiary Cone Crusher circuit is comprised of two cone crushers operated in parallel. Each cone crusher is fed via a feed bin and belt feeder.

The tertiary crushed material (–10 mm) is removed by the Tertiary Cone Crusher Product Conveyor and fed to the Tertiary Screening Circuit, which is comprised of two single-deck screens (+10 mm). Oversize reports to the tertiary cone crushing area while undersize reports to the Screen Product Conveyor.

The Screen Product Conveyor transfers the final product material (P_{100} =13 mm) to either of the two 8,000 t mill feed silos. A diverter chute on top of the first silo diverts the material to Mill Silo No.1 or Mill Silo 2 via an additional feed conveyor.

17.5.3 Milling

The crushing and screening circuit product (P_{100} of 13 mm) for each concentrator plant will be stored in a silo before being transferred onto the Mill Feed Conveyor via a variable speed feeder. The 0.77 Mtpa concentrator has a single 1,600 t silo and the 4.4 Mtpa concentrator has two off 8,000 t silos (one per 2.2 Mtpa milling module).

Each milling circuit consists of a single ball mill with a grate discharge liner arrangement as follows:

- Phase 1: Single 18'ftØ x 24½'ft EGL, 4.5 MW fixed speed grate discharge ball mill with a design throughput rate of 96.3 dtph
- Phase 2: Two off milling modules each fitted with a 22½'ftØ x 34½'ft EGL ball mill and 12 MW (2 x 6 MW) variable speed geared pinion drives. Each mill has a design throughput rate of 275 dtph

In each circuit, the ball mill operates in closed circuit with a classification cyclone cluster. The mill feed material (F_{100} of 13 mm) is fed to the mill feed hopper where process water is added for in-mill density control. Copper collector is also added into the mill feed hopper to assist in maximising copper recovery in the high-grade flotation circuit.

The milled material discharges onto the vibrating mill discharge screen for scats removal. Mill scats are deposited onto a scats stockpile via a scats removal conveyor. Scats from the stockpile are removed via front end loader and taken to the waste handling area, or alternatively scats can be re-loaded onto the mill feed conveyor.

The screened material is collected in the mill discharge sump and pumped to the mill classification cyclone cluster, which produces an overflow product of P_{80} = 75 µm. The cyclone underflow is recycled to the mill feed hopper for further regrinding. The cyclone overflow gravitates to the Rougher Flotation Feed Tank via a two-stage sampling system. The oversize from the linear screen is removed as trash.

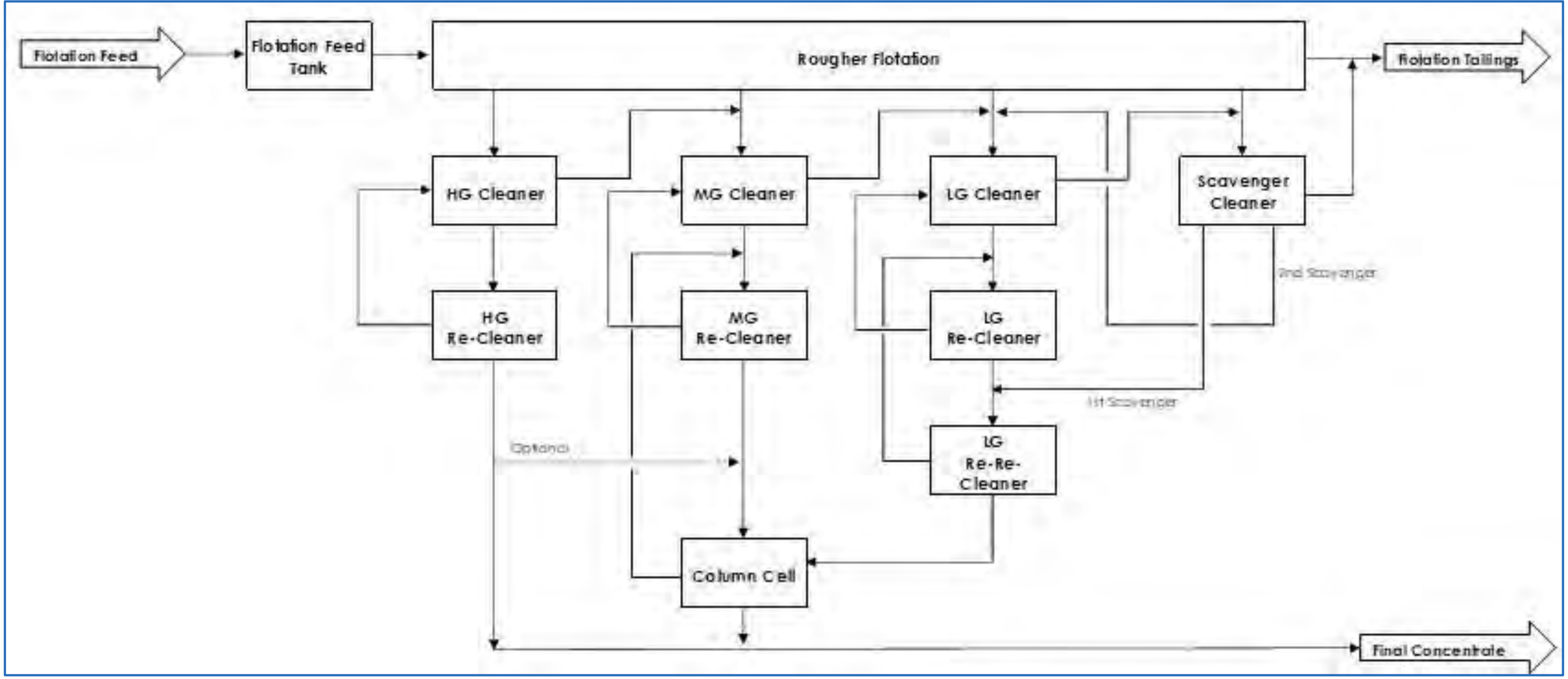
17.5.4 Flotation

The 0.77 Mtpa concentrator plant floatation circuit consists of single module while the 4.4 Mtpa flotation circuit consists of two identical modules, each capable of treating 2.2 Mtpa.

The same flowsheet has been used for each of the flotation circuits. There are variances in the cleaner circuit mass balance for the 0.77 Mtpa and 4.4 Mtpa concentrator plants as the 0.77 Mtpa circuit design caters for a high mass pull scenario as required in the early years of mining when treating a high-grade feed of up to 6.1 g/t 3PE +Au which reduces to an average of c.4.3 g/t once the 4.4 Mtpa concentrator comes online.

The flotation circuits are described below and a high-level summary flow sheet is presented in Figure 17.4.

Figure 17.4 Flotation Circuit High Level Flow Sheet



DRA

The rougher flotation feed, at approximately 28% to 30% solids (w/w), is pumped from the Rougher Flotation Feed Tank to the Rougher Flotation Circuit. The rougher flotation bank consists of forced air, tank cells with total a residence time of 75 minutes as follows:

- 0.77 Mtpa concentrator: A single rougher flotation circuit comprised of 1x 20 m³ and 5 x 70 m³, forced air, tank cells
- 4.4 Mtpa concentrator: Two off rougher flotations modules each comprised of 1x 30 m³ and 8 x 130 m³, forced air, tank cells.

Four concentrates will be produced namely high grade (HG), medium grade (MG), low grade (LG) and scavenger concentrate that report to the HG, MG, LG and Scavenger cleaners respectively.

The rougher flotation tailings gravitate to a tailings sump via a two-stage sampling system. The rougher tailings, together with the scavenger cleaner tailings, is pumped to the final tailings thickening and disposal circuit.

Collector, depressant and promotor are added into the MG and LG Rougher feed box, and coagulant is added to the rougher tailings tank to assist with coagulation in the tailings thickener.

HG rougher concentrate is pumped to the HG Cleaner flotation circuit. Depressant and frother are added to the HG Cleaner feed box. The high-grade cleaners consist of 5 x 0.5 m³, forced air, trough cells for phase 1 and one bank of 2 x 3 m³, forced air, trough cells per flotation module for Phase 2. The HG Cleaner concentrate is collected and pumped to the HG Re-Cleaner flotation circuit. The tailings of the HG Cleaner gravitates to the HG Cleaner Tailings Sump, from where it is pumped to the MG Cleaner feed box.

Pumped HG Cleaner concentrate feeds the HG Re-Cleaners consisting of 3 x 0.5 m³, forced air, trough cells for Phase 1 and one bank of 4 x 0.5 m³, forced air, trough cells per flotation module for Phase 2. Depressant and frother are added to the HG Re Cleaner feed box. Concentrate from these cells is pumped to a vezin sampler at the concentrate handling section. An option also exists to pump the HG Re Cleaner concentrate to the column flotation cell. The tailings of the HG Re Cleaner gravitates to the HG Cleaner feed.

MG Rougher Concentrate and HG Cleaner tailings are pumped to the MG Cleaner feed box where depressant, collector and frother are added. The medium-grade cleaners consist of 5 x 3 m³, forced air, trough cells for phase 1 and 3 x 10 m³, forced air, tank cells per module for phase 2. Concentrate from these cells is collected and pumped to the MG Re-Cleaner flotation circuit. The tailings of the MG Cleaner are pumped to the LG Cleaner circuit.

The pumped concentrate from the MG Cleaners, along with added depressant and frother, is fed to the MG Re-Cleaners. The MG Re-Cleaners consist of 2 x 3 m³, forced air, trough cells for Phase 1and one bank of 4 x 3 m³, forced air, trough cells per flotation module for Phase 2. Concentrate from these cells is collected and pumped to the Cleaner Column flotation circuit. The combined MG Re-Cleaner concentrate, LG Re-Re-Cleaner concentrate, and HG Re-Cleaner (optional) concentrate combine as feed into the column flotation cell. Concentrate produced from the cleaner column flotation cell is pumped to a dedicated sampler at the concentrate handling area. The tailings of the cleaner column flotation cell are pumped to either the MG Cleaner feed box or alternately the MG Re Cleaner feed box.

LG Rougher Concentrate, MG Cleaner Tailings and Scavenger Cleaner Concentrate are pumped to the LG Cleaner feed box, where depressant, collector and frother are added. The LG Cleaners consist of 3 x 50 m^3 forced air, tank cells for phase 1 and one bank of 5 x 50 m^3, forced air, tank cells per module for Phase 2. Concentrate from these cells is pumped to the LG Re-Cleaner flotation circuit. The LG Cleaner tailings is pumped to the Scavenger Cleaner feed box.

LG cleaner concentrate, with added depressant and frother, is pumped to the LG Re Cleaner feed box. The LG Re-Cleaners consist of 3 x 10 m^3 forced air tank cells for Phase 1 and one bank of 6 x 10 m^3, forced air, tank cells per module for Phase 2. Concentrate from these cells is pumped to the LG Re-Re-Cleaner feed box. The LG Re Cleaner tailings gravitates to the LG Cleaner circuit. The LG Re-Cleaner tailings is pumped to the LG Cleaner feed box.

LG Re-Cleaner concentrate and the first Scavenger Cleaner concentrate is pumped to the LG Re-Re-Cleaner feed box, where frother is added. The LG Re-Re-Cleaners consist of 2 x 3 m^3 forced air, trough cells for Phase 1 and one bank of 6 x 3 m^3 per module for Phase 2. Concentrate from these cells is pumped to the cleaner column flotation cell and the LG Re-Re-Cleaner tailings gravitates to the LG Re Cleaner feed.

Scavenger Rougher concentrate and LG Cleaner tailings are pumped to the Scavenger Cleaners feed box. Depressant, collector and frother are added. The Scavenger cleaners consist of 5 x 50 m^3, forced air tank cells for phase 1 and 6 x 70 m^3, forced air, tank cells per module for Phase 2. The first concentrate from the Scavenger Cleaner cells feed the LG Re-Re-Cleaner circuit, and the rest of the concentrate is fed to the LG Cleaner flotation circuit. The tailings of the Scavenger Cleaner feeds the Scavenger Cleaner Tailings Sampler, located at the rougher tailings sump area.

The flotation circuit design includes a FloatStar software system to ensure optimal level and mass pull control in the flotation circuit. In addition, online analysers on the final concentrate and flotation column concentrate streams will allow for real-time analysis of the concentrate grade.

17.5.5 Concentrate Handling and Filtration

The HG Re-Cleaner concentrate and Column cell concentrates from each flotation circuit feed dedicated vezin samplers ahead of a vibrating trash screen. The screened concentrate gravitates to a concentrate thickener. Flocculant is added to the thickener feed. The thickened concentrate at 55-60% solids (w/w) is pumped to a combined final concentrate vezin sampler before reporting to one of two concentrate storage tanks. The overflow product from the concentrate thickener is utilised as spray water in the flotation circuit.

Slurry from the concentrate storage tanks is fed to the horizontal plate pressure filter. The filter cake, with a moisture content of 12–14%, discharges onto a transfer conveyor which feeds a reversible shuttle conveyor. The concentrate cake is conveyed into concrete storage bunkers. The filter filtrate reports to the concentrate thickener. The concentrate cake product is loaded onto trucks and sampled by an auger sampler before dispatch.

17.5.6 Tailings Dewatering and Transfer

The combined tailings from each flotation circuit is pumped to a guard cyclone. Guard cyclone overflow gravitates to a tailings thickener, where flocculant is added in the feedwell. The thickened underflow and guard cyclone underflow are combined in the filter feed tank (0.77 Mtpa concentrator) or tailings disposal tank (4.4 Mtpa concentrator). The overflow product from the tailings thickener is utilised as process water for the respective concentrator plants.

The Phase 1 thickened tailings will be filtered, and the filter product will report to either the TSF or alternately to the backfill plant which includes a re-claim hopper for processing the filtered tailings.

The 4.4 Mtpa concentrator design has been updated to reflect a dry-stack tailings facility where thickened concentrator tailings will report to either the TSF (for further de-watering) or alternately to the backfill plant which is comprised of two paste fill plant modules. Three tailings pipelines and pump trains will be installed for pumping the 4.4 Mtpa concentrator tailings slurry to the TSF, namely a single 2.2 Mtpa line and two lines (duty/standby) for the combined 4.4 Mtpa capacity. Two tailings pipelines and pump trains will be installed for pumping tailings slurry to the paste backfill plant with each system able to accommodate the feed requirements for a single paste fill plant module.

The process water from the backfill plant will return to the 4.4 Mtpa concentrator plant tailings thickener to be used as process water.

17.5.7 Water Circuit

For each concentrator plant, process water is stored in a process water tank, which is fed with tailings thickener overflow water, excess flotation spray water and TSF return water. Each milling-flotation module is equipped with a dedicated process water pump installation. Clean water from the clean water tank provides for the process water make-up requirements. Excess process water will be pumped to the mine return water dam.

A dedicated fire water tank located in the concentrator plant area supplies fire water for firefighting purposes.

The potable water treatment plant will treat borehole water via ultra-filtration and reverse osmosis to produce potable water which is distributed throughout the mine site. The Masodi water treatment plant will filter water from the Masodi water supply system before reporting to the clean water circuit.

17.5.8 Air Services

Low pressure blower air for the flotation circuits is supplied by positive displacement blowers.

Plant and instrument air are supplied by compressors, delivering compressed air at 1,300 kPa(g) which is then pressure reduced to 800 kPa(g). Instrument air passes through an air filtration and drying system. The remainder of the air is used as compressed air. Dedicated air receivers, prior to each flotation section, provide instrument air buffering for valve operations. Further air receivers are placed throughout the plant close to large consumers such as the flotation column cell.

The drying air for each concentrate filter press is drawn from dedicated compressors and air receivers. Filter pressing air is supplied by two high pressure compressors delivering compressed air at approximately 1,600 kPa(g). The compressors and air receivers are situated adjacent to the respective filtration building.

17.5.9 Reagents

Dedicated reagent systems for each concentrator allow for reagent make-up and dosing for both the 0.77 Mtpa and 4.4 Mtpa concentrator plants. A summary of each reagent make-up and dosing system is provided below.

Flocculant granules are delivered in either 25 kg or 1,000 kg bags and are manually loaded into a bulk-bag bin receiver. The flocculant granules, together with reagent water, are transferred to a wetting and mixing system. The flocculant is diluted to 0.5% w/v strength in the flocculant dosing tanks. Further in-line dilution occurs while being pumped to the respective dosing points.

Depressant granules are delivered in 1,000 kg bags and loaded into a depressant storage hopper. The depressant granules together with reagent water are transferred to a wetting and mixing system. The depressant is diluted to 1.0% w/v strength in depressant dosing tanks prior to being pumped to the respective dosing points.

Collector (SIPX) pellets will be received in 25 kg bulk bags and added manually into the collector mixing tank. Reagent water will be added into the agitated collector mixing tank. After mixing, the collector solution is transferred to the collector dosing tank, after which it is pumped to the respective dosing points.

Copper collector will be received in 1 m^3 intermediate bulk containers (IBC), added manually into the copper collector dosing tank, and pumped to the respective dosing points.

Promotor will be received in 1 m^3 IBCs, added manually into the Promotor dosing tank and pumped to the respective dosing points.

Frother will be received in 1 m^3 IBCs, added manually into the frother dosing tank and pumped to the respective dosing points.

Coagulant will be received in 1 m^3 IBCs, added manually into the Coagulant dosing tank and pumped to the respective dosing points.

17.5.10 Control Philosophy

High-level concentrator plant control philosophy and piping and instrumentation diagrams were prepared.

17.6 Concentrator Engineering

This section describes the engineering process and factors that influence and shape the final concentrator plant layout and physical designs. It further provides relevant technical information and descriptions of main equipment associated with the concentrator plant.

17.6.1 Basis of Engineering

The basis of engineering provides the legal and technical framework on which the concentrator design is founded. It consists of three main pillars:

- Legislation and Standards.
- Environmental Management Plan.
- Design Criteria and Specifications.

The pillars are further detailed to indicate the impact on, and nuances associated with the Platreef Project site and concentrator design.

National policies and legislation have been considered to ensure technical viability and socially responsible design. Where applicable, national, provincial and local municipal laws and by-laws have been considered and incorporated in the design and estimate. The estimate allows for technically competent and legally appointed personnel to review and approve detailed engineering designs during the execution phase, as required by South African engineering law. Relevant international standards e.g. ISO, EN, BIS, DIN and national technical standards e.g. SANS, have been identified and adhered to during the concentrator design.

Standards and applicable legislations have been listed in the relevant design criteria and specification documents.

Adherence to national noise, dust and light pollution limits, prescribed by national and World Bank standards, has been considered along with visual impact. Where applicable, layouts and equipment design incorporate the required guidelines to ensure compliance.

An earth berm around the mine perimeter functions as a barrier for light pollution, a deflector for noise and a visual pollution barrier.

The rock handling area, consisting of conveyors, crushing and screening areas was previously identified as major noise and dust contributors. Surface vent fans also contribute to noise generation. The noise consultant further modelled vent fan noise propagation with and without the perimeter berm wall.

Dust extraction and suppression has been considered across the stockpiles, rock handling and processing systems. The Phase 1crushing and screening plant provides for dust suppression sprays, on the conveyor and crusher feed and discharge points. The Phase 2 crushing building, screening building and all silos are designed with independent dust extraction systems. Conveyor discharge chutes, transfer towers and tipping areas all include dry fog dust suppression systems.

Haulage roads will be regularly sprayed with water to suppress dust and treated with dust suppression chemicals to reduce water usage, while stockpiles can be treated with coagulating polymers to prevent dust billowing in sensitive areas.

Project-specific design criteria were developed. These documents are based on DRA-developed design criteria for cost effective, technically sound and maintainable plant design. Combined criteria exist for surface and underground, with nuisances clearly specified. The design criteria include experience based, operationally verified and calculated design detail that encompasses the cumulated experience DRA has generated throughout various projects. The design criteria are further supported by detailed specifications that stipulate technical requirements to contactors. The design criteria and specifications ensure conformity of design, corrosion protection and paint colouring across the entire mining site.

17.7 Conclusion

The metallurgical testwork programme has yielded sufficient information to develop a definitive metallurgical flow sheet. The Platreef 2022 FS design allows for process and ore variability through conservative design.

The engineering design principles are aligned with the process design. The engineering design has considered the required international and national laws and standards. Design criteria and detailed specifications ensure conformity of design and safety across the mine site. The concentrator design is based on previously constructed and proven process modules designed by DRA, taking lessons learnt and experience into account.

The engineering design has taken cognisance of the required environmental and social impact of the concentrator in terms of noise, dust, light and visual pollution requirements stipulated by various national and international bodies. Expert consultants were contracted to incorporate best practice design and equipment into relevant sections of the plant to minimise the overall environmental and social impact.

The concentrator plant has been designed in accordance to the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities. The modular approach toward the milling and flotation plants allows for redundancy and phasing of capital spend.

18 PROJECT INFRASTRUCTURE

18.1 Overview

The Project site is located approximately 12.5 km north of the centre of Mokopane in the Limpopo Province and falls under the Mogalakwena Municipality. The mine lease area is on the Turfspruit, Macalacaskop and Rietfontein farms. Year-round access to the site is by paved, all-weather national highway (N11) from Mokopane (formerly Potgietersrus). This road is a two-lane tarmac road suitable for heavy loads year-round. The N11 is a national road, falling under the jurisdiction of the South African National Roads Agency (SANRAL), an intersection to SANRAL specifications has been constructed to the mine gate.

The site has an existing bulk electrical supply of 5 MVA and a 100 MVA supply will be added as part of the project.

The site is currently supplied water from boreholes on Ivanplats' Uitloop and Turfspruit properties. The extraction of the water from the well field is licenced in terms of the South African National Water Act. The increased steady state water demand will be met by a grey water supply from a local wastewater treatment works located near the mine.

The mine lease area is situated in the Mogalakwena River valley. Mountainous areas run to the east and west of the lease area, while the mining area itself is relatively flat. Mountainous areas are to be found in the north-eastern corner of the lease area with several isolated ridges. An extensive storm water management system will be constructed by the project to manage the storm water run-off from these areas.

The land abutting the mine area is mainly used for agricultural activities and livestock, however the town of Mokopane is a well developed, urban area, with a commercial centre, medical facilities and established housing developments. The city of Polokwane lies 70 km to north-west of the site and can be reached by road in approximately an hour along the N1 national highway. The N1, reached via the N11, also links the site to the Gauteng Province in the south.

The internal road design philosophy is that as far as possible delivery vehicles will remain on the main entrance road to the required delivery points and parking areas, with internal roads reserved for the delivery of equipment from the stores to the work area. The internal roads and parking take into account the traffic flow inside the mine area. Gates separate areas in order to restrict access, but without reducing serviceability and production. Specialist traffic flow studies were performed, and the recommendations were implemented into the road designs.

To mitigate the noise, dust and the visual impact of the mine, an approximate 7.5 m to a maximum of 10 m high berm on the periphery of the mining area will be constructed. The perimeter berm is divided in four sections with the sections scheduled to be built at different times.

Plot and block plans were developed with a holistic view of the complete mine lease area and directly affected communities in mind. Layout work for the mining area, which includes a shaft area, process plants, tailings facilities and various general infrastructure sections, were done with these areas in relation to each other after which these areas were looked at individually.

The Project surface infrastructure is thus broken down into three distinct areas:

- Mining surface infrastructure – all supporting infrastructure located within the shaft bank area, i.e. roads, buildings, dams, services, bulk earthworks and electrical reticulation, etc.
- Process plant infrastructure – all supporting infrastructure located within the process plant area, i.e. roads, buildings, dams, services, bulk earthworks and electrical reticulation, etc.
- General site infrastructure – bulk supply infrastructure found within the waste management area, the general office area and all supporting infrastructure which links the three areas together. Additionally, the bulk supply of water and power are included under this area.

Section 18 was primarily based on the following information:

- Ivanplats PDP Feasibility Study Report - Section 10 - Surface Infrastructure, DRA Report Number JZADBR5308-STU-REP-010, January 2022, by DRA Projects.
- Eskom Budget Quote pertaining to cost and design of the Bulk Electrical Power Supply
- Capital Costs pertaining to the design and construction of the Masodi Water Treatment Works Water Supply Cost.

18.1.1 Stormwater Management

The mine area is situated in a former riverbed, and it is exposed to large amounts of run-off rainwater. The catchment area of this stormwater extends to the hillside located 5 km away. It runs through Magonga, Baloi and Macheke, before it crosses the existing N11 onto the Platreef mining site. Currently, all run-off rainwater flows in a sheet flow fashion thus being of no consequence. However, once the mine infrastructure is constructed, concentrated flow of rainwater will be created in the area which needs to be managed via a stormwater management system in order to negate its effect on the surrounding communities. The stormwater management system consists of the following drains:

- Clean Water Cut-off Drains,
- Stormwater Run-off Drains, and
- Discharge Drain.

The storm water from the clean water cut off drains is collected in an attenuation pond to regulate the flow discharged through the drainage system past the communities down-stream of the mine. Stormwater collected on the mine site itself is not discharged, but collected and stored for use in the Concentrator Plants and Mining service water.

18.1.2 Stockpiling of Material

The ore and waste rock stockpiling encompass the stockpiling of all commodities generated during both construction and operational phases except for topsoil (topsoil will be stored in the perimeter berm). These commodities include:

- Ore,
- Waste rock uncrushed and crushed,
- Low-grade ore,
- Subsoils (soft and hard), if required, and
- Temporarily imported G4 material for platform construction.

18.1.3 Surface Structure

The buildings have been classified into the following categories:

- Architectural Buildings
- Pre-fabricated Buildings
- Workshops and Stores
- Change House
- Electrical Buildings

Certain identified architectural buildings will be designed and registered as Green Star Buildings.

The electrical design is based on the equipment specifications and electrical design criteria developed for the project. The electrical equipment is designed or selected to:

- Comply with Eskom energy efficient requirements.
- Provide for high plant availability.
- Provide an effective, simple solution which is maintainable by the plant operating personnel.
- Provide a safe working environment for personnel and equipment.
- Ensure that effort has been made to optimise the efficient use of energy and to minimise any adverse effects on the environment.

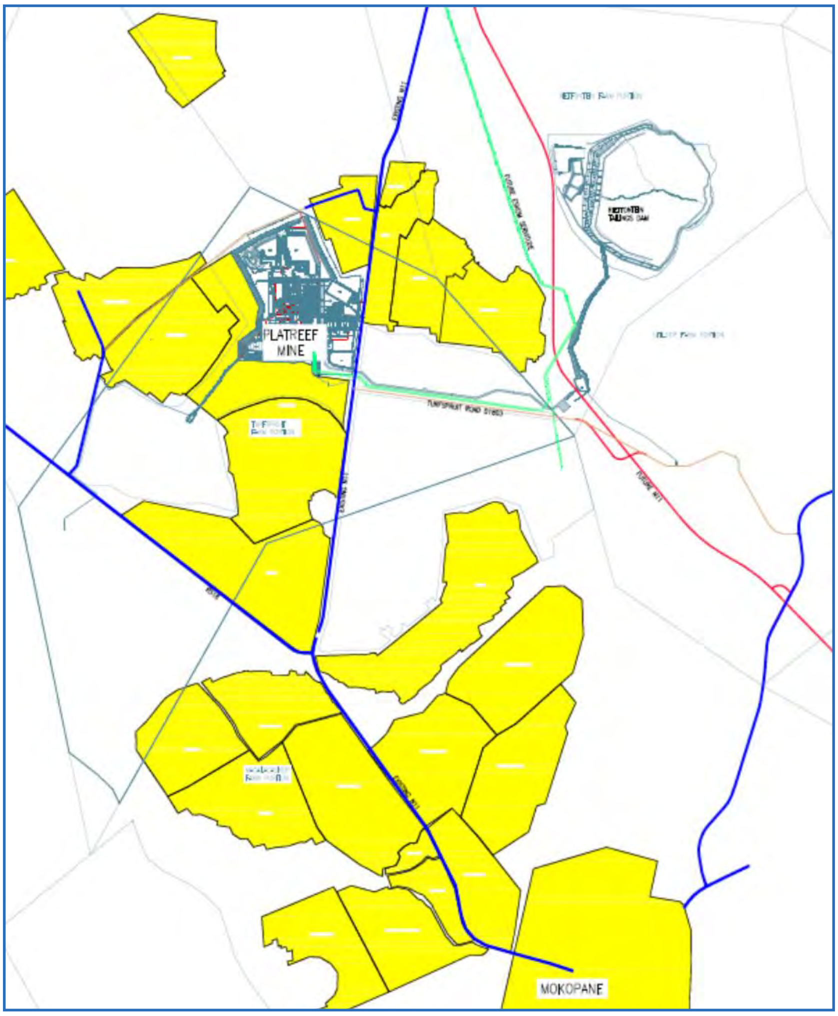
18.2 Surrounding Infrastructure

The Project site is located approximately 12.5 km north of Mokopane a town in the Limpopo Province. The city of Polokwane lies 70 km to north-west of the site. Polokwane is the capital and largest urban centre in the Limpopo Province. A commercial airport Polokwane International Airport is located to the north of the city.

The N11 continues from the mine gate through to Mokopane and intersects with N1 national highway to the south of Mokopane, which is the main route to the Gauteng Province. All these roads are suitable for heavy loads year-round.

The Project site is surrounded by many informal settlements and villages, with Ga-Kgobudi, Ga-Madiba, Ga-Magongoa, Mzombane and Tshamahansi being the closest. The close proximity of these villages to the Project site was taken into consideration in the design and engineering of all infrastructure and emphasised the importance of mitigating noise and dust pollution, as well as the visual impact that the Project will have on the communities.

Figure 18.1 Platreef Project Location within Local Region



DRA, 2017

18.3 Site Topography

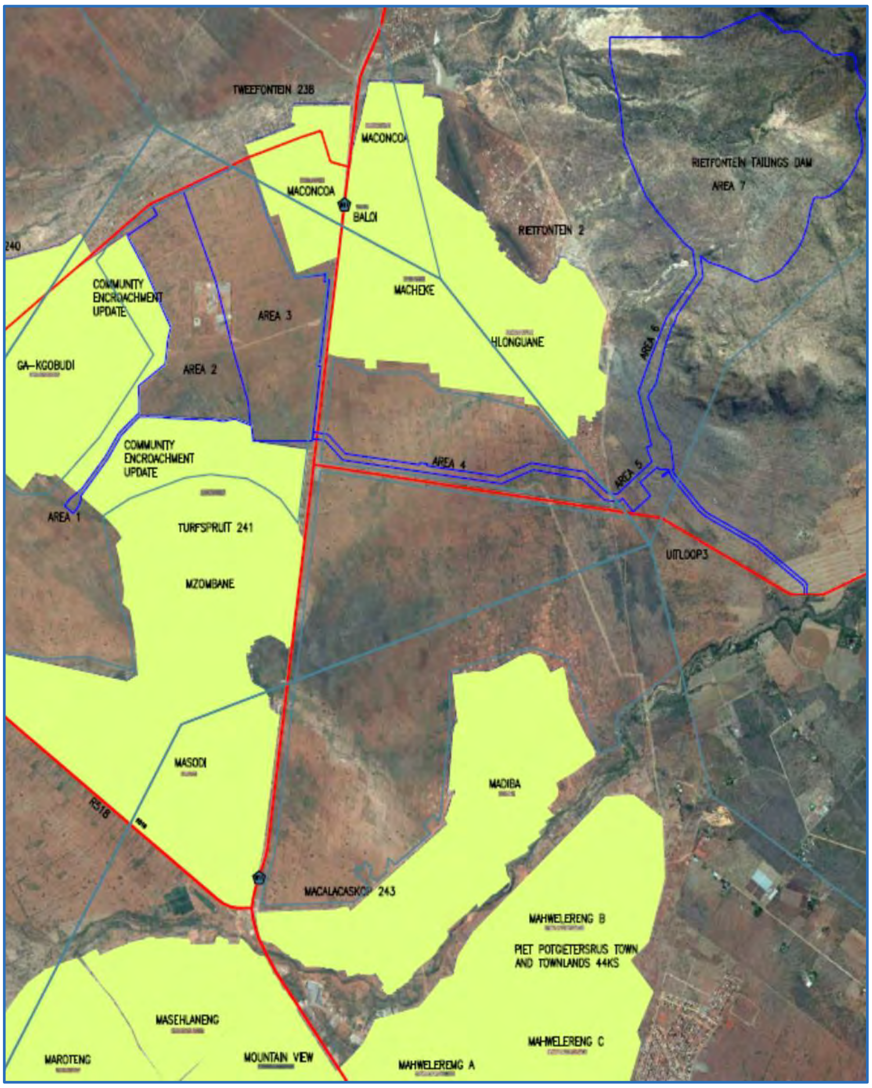
The mine lease area is situated in the Mogalakwena River valley, with the anticipated mining area situated in lease area as per Figure 18.2. Mountainous areas run to the east and west of the lease area, while the mining area itself is relatively flat. Mountainous areas are to be found in the north-eastern corner of the lease area with several isolated ridges. The current land within the mine area is used mainly for agricultural activities and livestock.

The majority of the mine lease area has gentle slopes of between 0–5°. Moderate slopes of between 6–15° occur in some areas. Isolated steeper slopes of between 16–21° occur along the banks of the Rooisloot and Klein-Sandsloot Rivers. The steepest slopes occur on the ridges and range between 22–69°.

The slope aspect and direction of the mine area is generally in a south-westerly direction towards the Mogalakwena River. Slopes in various other directions occur in isolated areas along the river valleys, channels and ridges.

The topographical model indicates that the elevation of the Project area increases from 1,030.5 mamsl in the Mogalakwena River floodplain in the south-western corner to 1,359 mamsl on the ridges in the north-eastern corner of the lease area.

Figure 18.2 Platreef Project Lease Area



DRA, 2017

18.4 Plot and Block Plans

18.4.1 Plot and Block Plan Development

Plot and block plans were developed during the previous feasibility study in 2017. As part of this phase development plan in the Platreef 2022 Feasibility Study, the team adapted these original plans to suit the phased implementation and operation of the Project, taking cognisance of the existing infrastructure on the site.

Previous work on the plot plan had positioned number 1 and 2 Shafts and subsequent positioning of shaft-related infrastructure around these. The existing security fence line defines the mining area and separates it from the surrounding communities. This fence line determined the perimeter and available area for the complete mining area plot plan development. The intersection with the N11, fixes the position of the main access to the mine.

The Project has been split into two main phases as follows:

- Phase 1 commences with the handover of the Shaft 1 and the development of the underground mine. The initial RoM production will be processed in a 770 ktpa concentrator plant, to be constructed in parallel with the underground development. The infrastructure construction is limited to the minimum required to support the operation of the mine and this plant.
- Phase 2 will commence with the continued sinking of Shaft 2, which will serve as the main production shaft during full production operations. The remainder of the infrastructure will be constructed in order to support the larger mining operation, the additional 4.4 Mtpa concentrator plant and the ramp up of mining production to steady state.

The phased approach has retained most the of design elements described in the previous 2017 FS report, with modifications and additions made as required.

Plot and block plans reflect a holistic view of the complete mine lease area and directly affected communities. Layout work for the mining area, which includes the shaft area, process plants, tailings facilities and various general infrastructure sections, was done with these areas in relation to each other, after which these areas were looked at individually. All layout work was done in close co-operation with the Platreef Project owner's team. The following major factors specifically influenced these plot and block plans:

- The requirement to allow footprint for potential expansion.
- The hydrology requirements prohibiting infrastructure from being placed in the 1 in 100-year flood lines.
- The proximity of neighbouring communities and their potential encroachment onto mine lease areas.
- Various licence application and environmental and social requirements.

18.4.2 Plot Plans

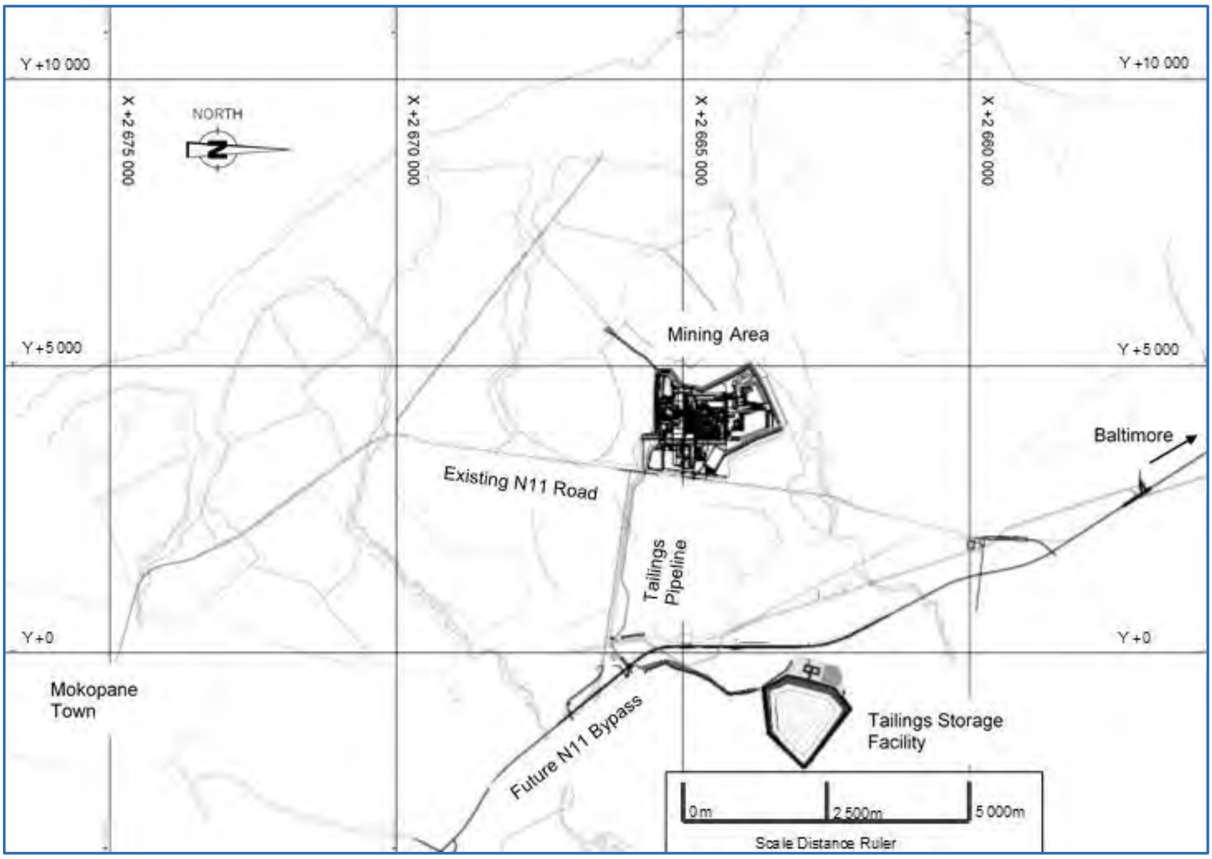
The plot plans give consideration to three main regional areas:

- The Platreef mine
- The Rietfontein TSF Site

The Tailings Pipeline servitude connecting these two areas

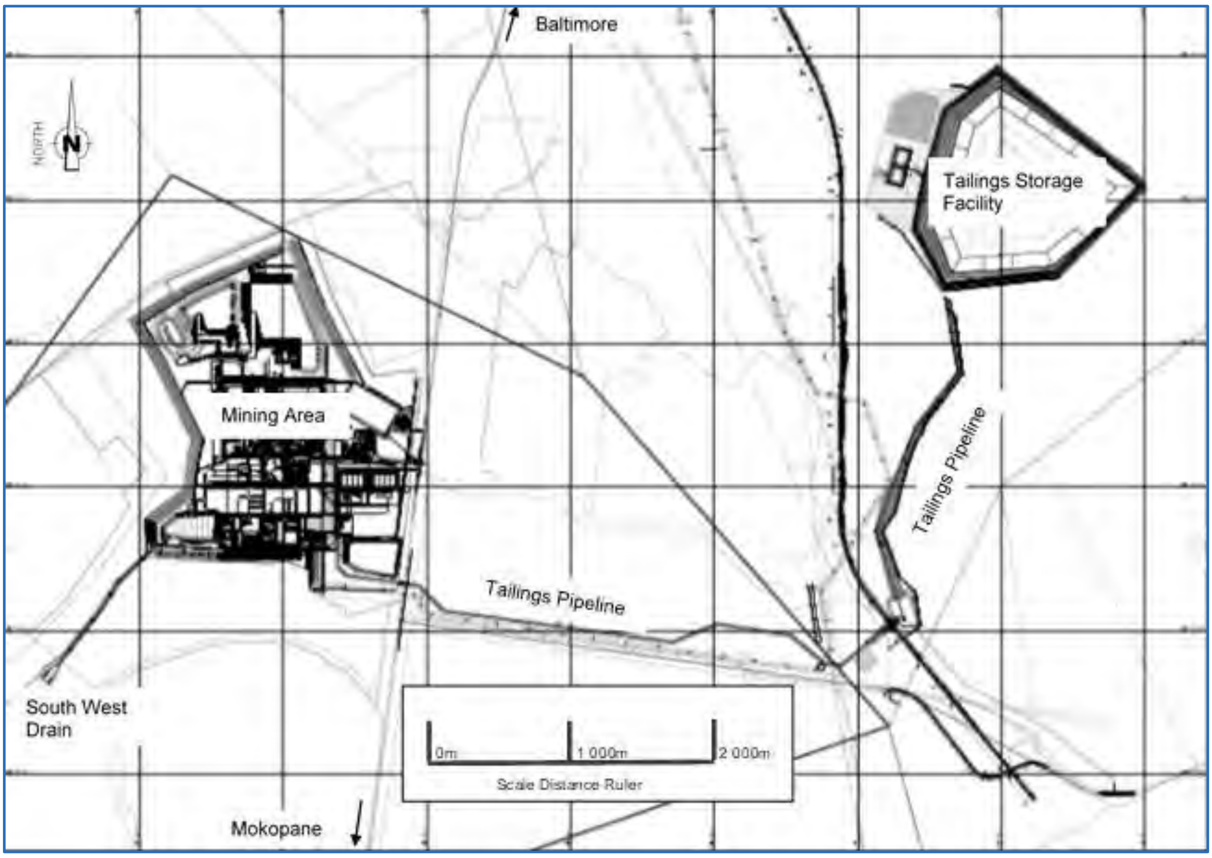
This overall plot plan of the Lease Area is shown in Figure 18.3 and Figure 18.4.

Figure 18.3 Lease Area Plot Plan



DRA, 2017

Figure 18.4 Mine and Rietfontein TSF Area Plot Plan

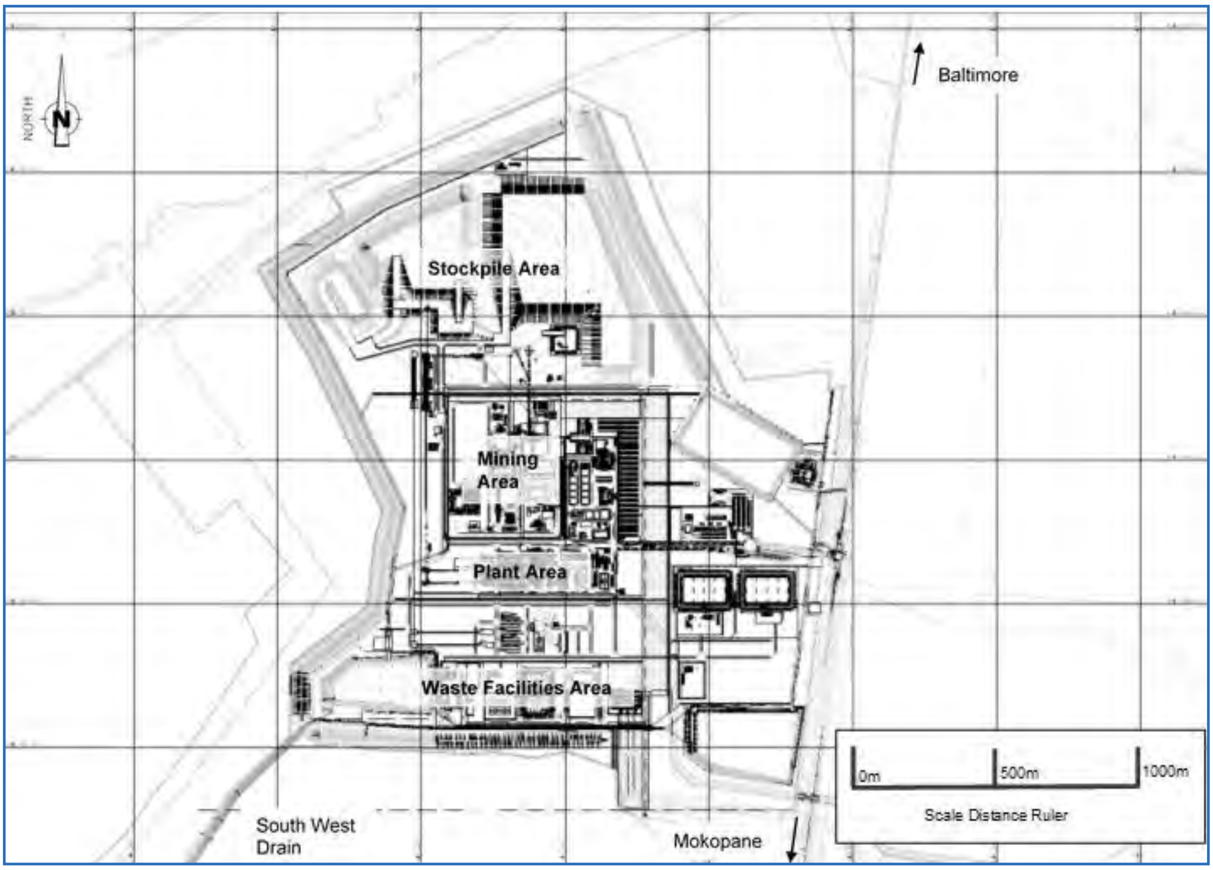


DRA, 2017

The mining lease area plot plan outlines the mine surface infrastructure within the registered mine lease area. The Mine and Rietfontein tailings storage facility (TSF) area plot plan outlines the mine area, the TSF pipeline corridor and the TSF dam perimeter in relation with each other. The mine area plot plan outlines all the mining and process plant infrastructure within the fence line west of the N11.

The mine area plot plan is shown in Figure 18.5.

Figure 18.5 Mine Plot Plan



DRA, 2017

An irregular-shaped fence line and perimeter berm indicates the available footprint to the west of the N11, wherein the Platreef 4.4 Mtpa mine is located, and is referred to as the mining area. The position and proximity of surrounding communities is also shown on this plot plan.

Bulk services enter the mining area from the east, with bulk power entering at the south-eastern corner and provision for bulk water entering adjacent to the N11 intersection access road to the mine. The treated municipal grey water will enter the mining area from the south-west corner. A 132 kV electrical substation is located near the southern boundary of the mining area between the waste facilities and Attenuation Pond 2. Two raw water dams are located in the eastern portion of the mine area, to hold a bulk water buffer.

The main access point is from the N11, with an emergency second access and exit to the north of the mining area. Culverts are catered for underneath the N11 to allow for crossing of power, services and the tailings lines from the east to the west of the N11.

Internal road routing is done so that optimal traffic flow is achieved with regard to deliveries to the shaft and process plant areas as well as accommodating concentrate transportation on site.

In line with the Integrated Water Use Licence Application (IWULA) and site topography, storm and pollution water systems are suitably located and integrated in the mining area. These systems comprise large drains and channels (mostly running north to south), a Stormwater Attenuation Pond in the south-eastern corner and a Stormwater Control Pond in the south-western corner of the mining area. Surface run-off water generated from the shaft and process plant areas is channelled to the Stormwater Control Pond.

The previously established location of the dedicated waste facilities in the south-western corner of the mining area has been retained. These facilities include a landfill site, sewerage plant, tyre storage, etc.

In line with the Environmental Management Programme (EMP), the EMP specialist study requirements for dust, visual impact and noise, the IWULA and IWMLA, perimeter berms are located on the southern, western and northern perimeter of the mining area. Waste rock and ore stockpiles are also situated in the northern section of the mining area.

The position of Shaft 1 and Shaft 2 are fixed and is located towards the centre of the mining area. An area approximately 400 m long and 500 m wide houses the shaft-related surface infrastructure and is fenced off by a concrete wall.

The Concentrator Plants are positioned to the south of the Shaft area. The previous configuration has been altered to position the 770 ktpa Concentrator Plant immediately south of the shaft area and the 4.4 Mtpa (and space allocation for further expansion) south of this.

A space allocation has been made for a future hydro-metallurgical plant (Kell Plant) to the east of the Concentrators.

The engagement centre and clinic are situated next to the main entrance of the N11 on the eastern side of the mine area.

18.5 Tailings Storage Facilities

The Phased Development Plan will see the development of two Dry Stack Tailing Storage Facilities (DTSF) for the Platreef Mine; one at the Rietfontein site and a smaller facility built as part of Phase 1 located near the Platreef Shafts.

The Preliminary Economic Assessment completed in 2020 for the Platreef Phased Development plan proposed to use the site previously allocated for a waste rock dump as the Phase 1 DTSF. This concept was developed further in the Platreef 2022 FS and Ivanplats (Pty) Ltd (Ivanplats) appointed Golder Associates Africa (Pty) Ltd (Golder) to prepare the environmental authorisation amendments and supporting engineering design, for the DTSF, at its Platreef Mine.

The Phase 1, Platreef DTSF provides a maximum capacity of 6 Mt of tailings storage, allowing a service life of approximately eight years at 700 ktpa production rate.

The proposed Phase 2 Tailing Storage Facility (TSF) site is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed TSF site is considered a feasible site considering all applicable engineering and environmental standards for tailings storage facilities. The TSF has an operating life of 25-years, during this time approximately 55 Mt of tailings will be stored within the TSF, the remainder of the tailings will be used as backfill in the underground mine. The TSF is compliant in terms of the required tonnage profile production split between the Backfill requirement and TSF of 35% on average but is conservatively designed for 40% of non-ore material reporting to the TSF.

Since the Platreef 2017 FS a hybrid paddock deposition methodology was proposed. However, Ivanplats has decided to change the TSF deposition methodology from upstream design to dry stacking in the Platreef 2022 FS. Following on a study undertaken by Golder Associates Africa in December 2016, it was concluded that stacked tailings storage facilities are deemed to be safer in that there is no hydraulic deposition, hence the risk will be minimal, to flood the surrounding areas with tailings in the unlikely event of a catastrophic failure. Stacked tailing storage facilities are more water efficient, in that the majority of water in the tailings is captured in the dewatering plant, pumped directly back to the concentrator and re-used back into the process.

The dry stacked tailings storage scheme as proposed is feasible, the stacked tailings facilities are envisioned to comprise of a starter dam, constructed primarily of rockfill, engineered tailings, nominally compacted tailings, and random fill. The tailings are to be dewatered with thickeners and disc filter technologies located at the Rietfontein site.

In the case of the Platreef site, tailings will be dewatered within the concentrator plant.

The material will be placed dry on the proposed layout area through the method of load and haul. The facilities will be accessed via a series of ramps, from the ramps, construction will take place in 5 m lifts with the operational benches of 10 m in width and an operational slope of 1V:2H. Once a lift has been completed, a ramp will be constructed to the higher elevation of the next consecutive lift.

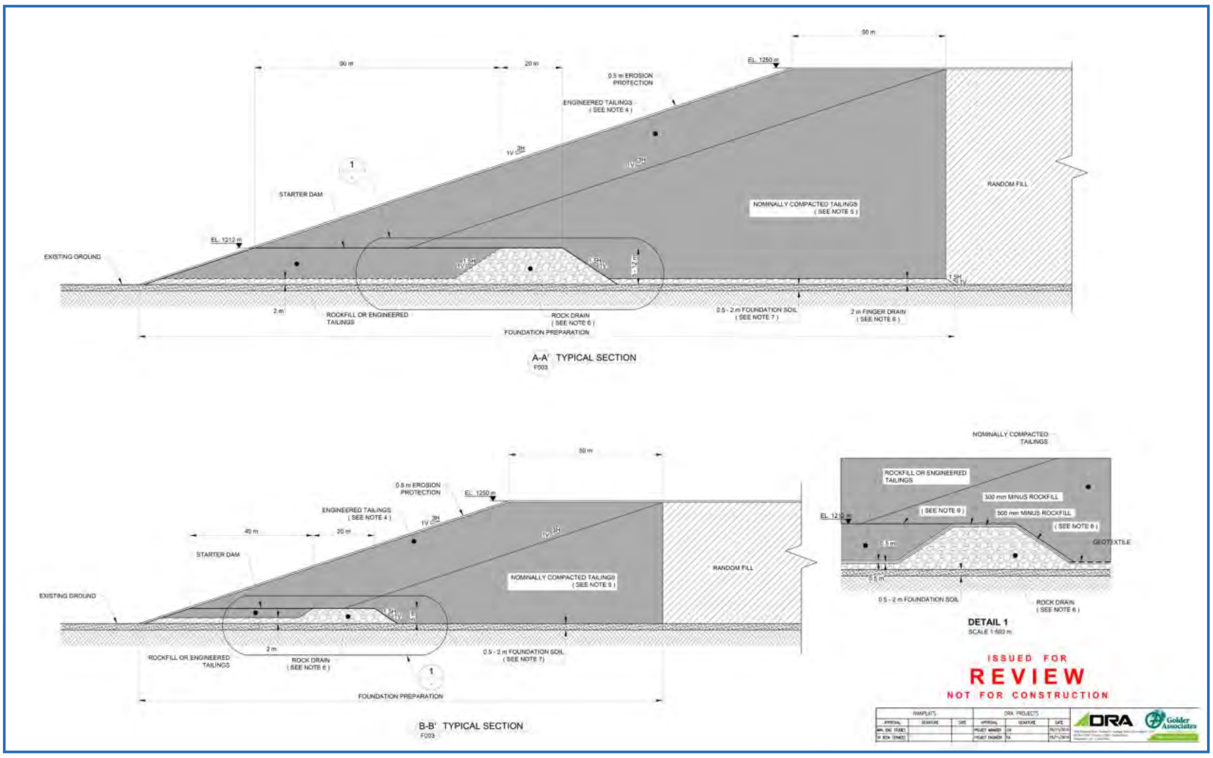
In additional to load and haul a series of mobile conveyors will be used on the Rietfontein site to place the tailings efficiently.

The DTSFs are lined with a reinforced geosynthetic clay liner (GCL) overlain with a high-density polyethylene (HDPE) liner. This system is compliant with the prescribed South African National Environmental Management Waste Act, 2008 (NEMWA) R636.

A staged development of both the facilities is a feasible strategy that will result in capital deferral. The Platreef TSF can be constructed in two phases 4 years apart and the Rietfontein facility in two phases 10 years apart. The Rietfontein conveyor system will be constructed and extended over the first five years of operation of that facility.

Figure 18.6 shows typical sections of the dry stacking TSF system.

Figure 18.6 Dry-Stacking TSF

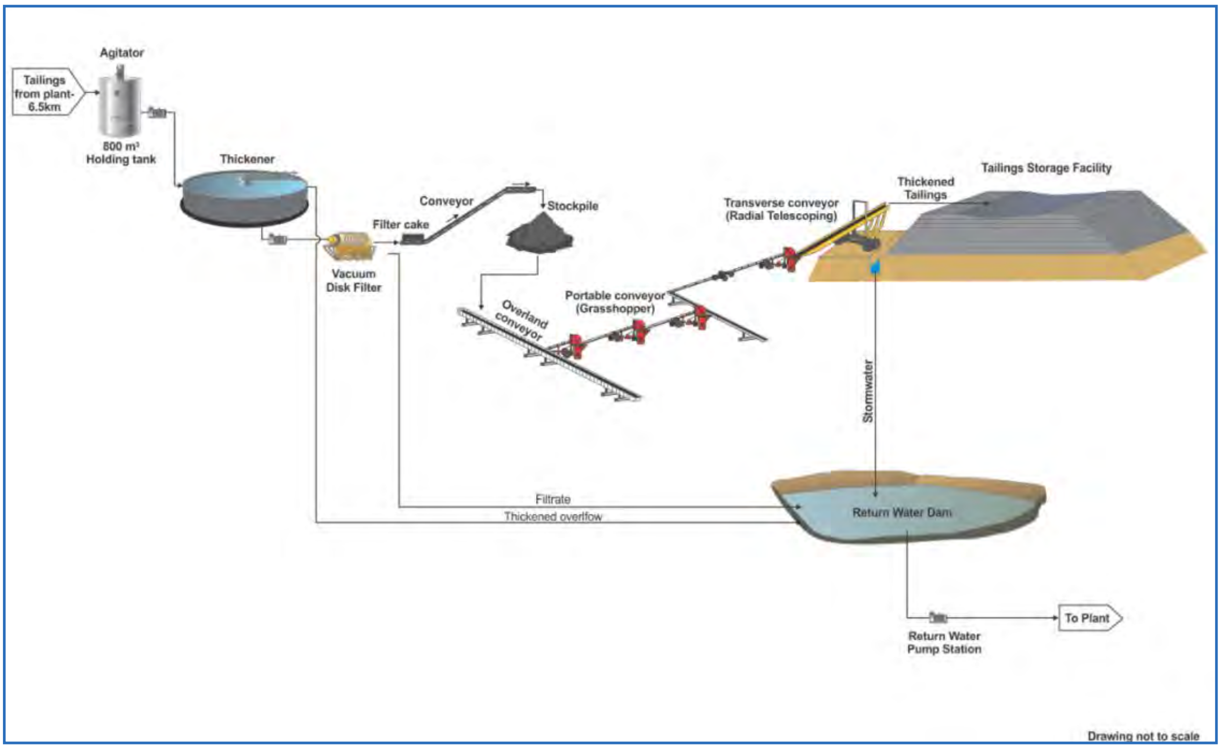


GAA, 2017

Water is recovered from the tailings that will be delivered to the dewatering facility using disc filters, the tailings are then transported by conveyors from the dewatering plant.

Aside from the rockfill in the starter dam and drainage elements, the facility will be developed using tailings, achieving the required dewatering and developing construction methods for the tailings will be required immediately upon start-up. Figure 18.7 shows a general overview of the complete dry stacking tails system.

Figure 18.7 Dry-Stacking Tails System



GAA, 2017

18.5.1 Conclusions and Recommendations

The use of the dry stack method extracts water from the tailings prior to placement, the dried (±16% moisture content) tailings are then placed in the storage area. This approach reduces the risk of failure as it can be reasoned that there would be a minimal phreatic level build up within the Dry TSF compared with a hydraulic deposition method. Both of the tailings storage facilities as proposed during the FS indicates that the facilities satisfy all considerations of stability investigated, namely static and pseudo static analysis. The DTSF during operations shall be monitored according to SANS 10286 and the applicable tailings management frameworks must be implemented, which includes a rigorous surveillance and monitoring programme.

The following recommendations are suggested for the consideration of Ivanplats (Pty) Ltd for the detailed design stage of the Rietfontein DTSF. Detail design of the tailings dam will include the following:

- Dam break analysis to confirm the extent of influence
- Consultation with the regulator
- Confirmation that the stormwater management system is performing.
- Development of a tailings management framework document should be established to ensure compliance with industry leading tailings management practices.

- Further development of the Construction Quality Assurance (CQA) plans to be implemented during the detailed design stage to ensure construction quality is maintained.
- Development of an Operations Manual for the DTSF.
- Development of a site-specific EPP for the DTSF to ensure compliance with local and international regulations.
- Development of a functional closure plan.

18.6 Water Supply

18.6.1 Bulk Water Source

The bulk water requirement for the mine is divided into the water required for construction and water required for operations. The water requirements include construction, dust suppression and water for developing the mine. The water volumes required for the development of the mine were based on recovering 88% of the water sent underground for mine development and 88% of the groundwater inflow. These assumptions will provide a conservative estimate of the water required for development. The sources of bulk water during the construction period are local groundwater abstracted from licensed boreholes on Ivanplats' Uitloop and Turfspruit properties, as well as storm water run-off collected on site. The yield from the boreholes on Uitloop and Turfspruit is sufficient to meet the construction bulk water requirements and will be used as the primary source of the potable water for the mine during operations.

Ivanplats requires an average bulk water supply of 7,700 m^3/d for the operational phase of the mine. Ivanplats is actively pursuing the following sources of bulk water:

- A supply from the Olifants River Water Resources Development Project (ORWRDP). This supply will be from the Phase 2B pipeline from the Flag Boshielo Dam.
- A local source of treated sewage effluent (grey water) from the Masodi Waste Water Treatment Works (WWTW).

On 17 January 2022, Ivanhoe concluded an agreement to receive local treated water to supply most of the bulk water needed for the first phase of production at Platreef. The Mogalakwena Local Municipality has agreed to supply a minimum of 3000 m^3/d of treated sewage effluent from the town of Mokopane's new Masodi Waste Water Treatment Works (WWTW). Initial supply will be used in Platreef's ongoing underground mine development, surface infrastructure construction and plant operations. The agreement provides to increase the supply up to a maximum 10 000 m^3/d of treated water, depending on the roll out of the supporting municipal infrastructure.

Under the terms of the agreement, Ivanplats will provide financial assistance to the municipality for certified costs of up to a maximum of R248 million (approximately $16 million) to complete the construction and commissioning of the Masodi WWTW. Ivanplats will purchase the treated water at a reduced rate of R5 per m^3 for the first 10 million litres per day to offset a portion of the initial capital contributed.

Provision has been made to treat the water received from the Masodi WWTW to ensure it is suitable for use in the process plant. Ivanplats remains an active member of the Joint Water Forum, which is pursuing the Phase 2B pipeline from the Flag Boshielo Dam as an alternative solution for future expansions.

Bulk Water Requirement

The net operational water requirements were considered for the most likely groundwater inflow scenario. The net operational water requirements are the water requirements after supply from on-site sources such as the re-use of sewage effluent, as well as run-off collected in Attenuation Dams and in Stormwater Pond 3. The net water requirements need to be supplied from a bulk water source, as described above. The variation in the net water requirements are due to the variations in run-off caused by the seasonal and annual variations in rain. In accordance with the project plan, the source of 7,700 m^3/d of bulk water needs to be available by Q2'30.

The bulk water supply will be supplemented by the local groundwater source are throughout the life of the mine. Either the envisioned the Masodi Grey water supply or the ORWRDP would be sufficient to meet the projected net water requirements.

18.7 Power Supply

18.7.1 Overall Power Requirements

A bottom-up estimating methodology was used to arrive at a predicted electrical consumption and NMD for the proposed installations at the Platreef site. The NMD is the maximum electrical power demand in kVA, over a half-hour period.

The complete list of connected loads is summed to arrive at a total connected load. The connected loads are reduced to connect running loads by excluding standby circuits and further reduced to absorbed power running loads as given by the process and mechanical data.

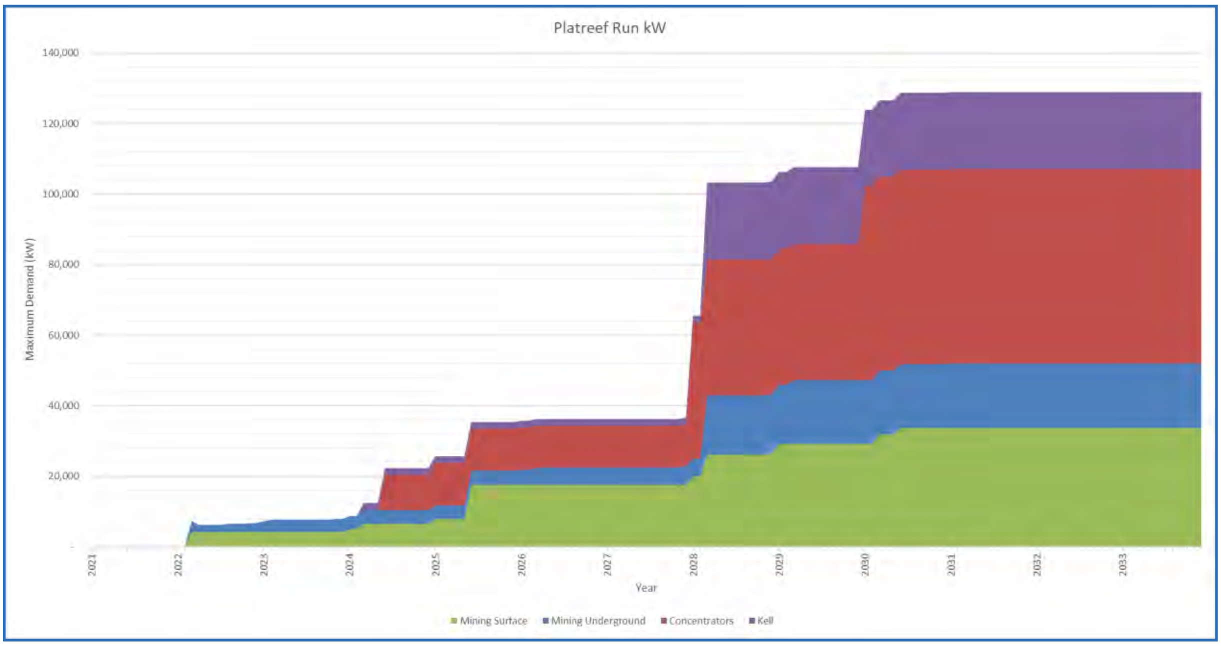
The results of these calculations are presented in Table 18.1.

Table 18.1 Notified Maximum Demand

Area	Connected (kW)	Running (kW)	Running (kVA)	Running (kVA with PFC)
2500 (Surf)	46,427	33,702	39,787	34,390
2600 (UG)	29,052	18,249	21,831	18,622
3000 (Plant)	79,219	55,090	63,932	56,223
Total Site	154,698	107,041	125,550	109,235

The electrical load build-up is presented in Figure 18.8. The electrical consumption over the life of mine is presented in the operational expenditure.

Figure 18.8 Electrical Load Build Up

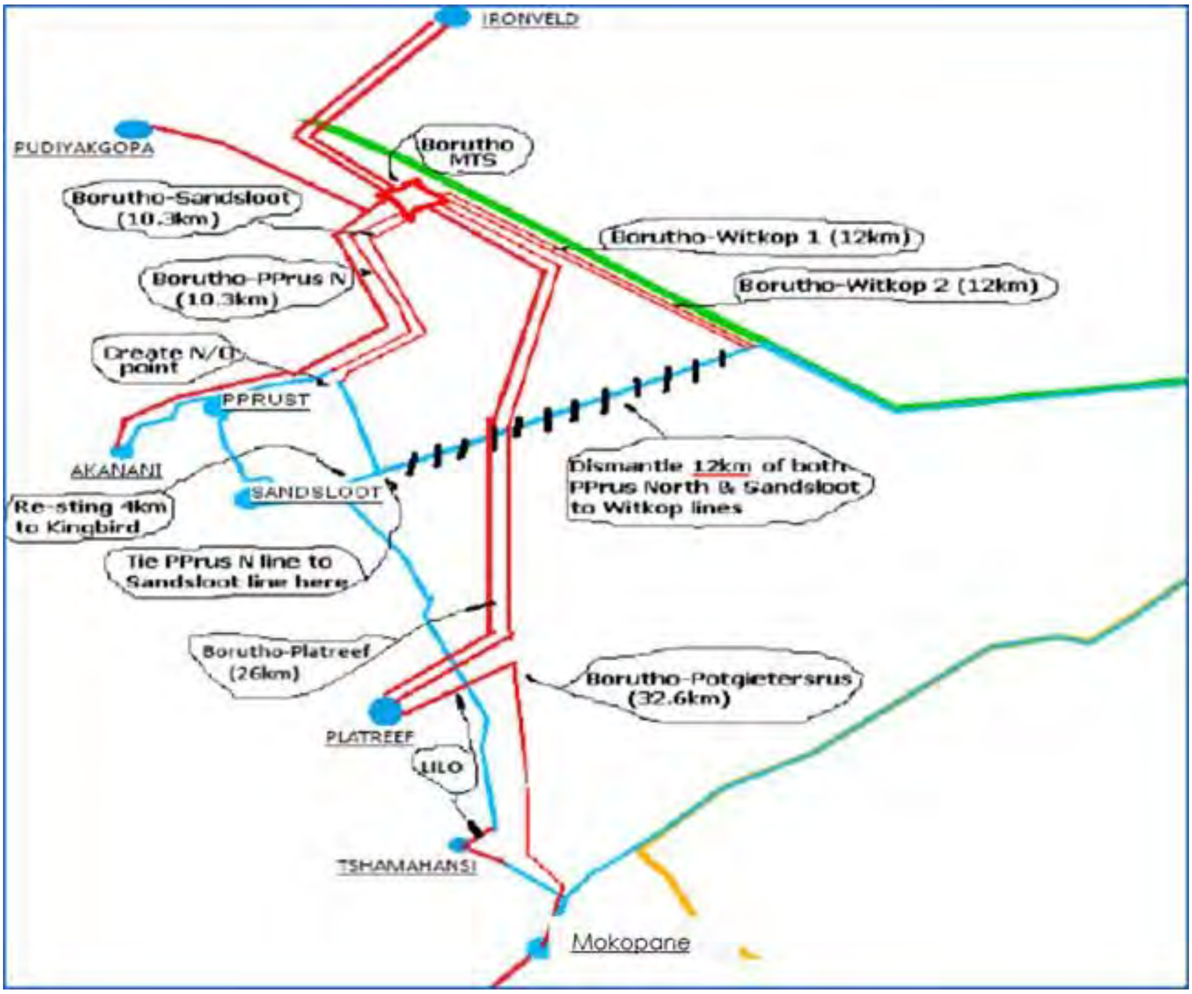


18.7.2 Bulk Power Supply

The bulk power supply is to be sourced from Eskom, the South African national power utility. The application for 100 MVA of power has been submitted to Eskom, and the budget quote received. The budget quote was accepted, and the appropriate deposits were provided. The original application requested a fully redundant premium supply project package from Eskom. The application scope has been updated to an Eskom 'self-build' project. Eskom has provided a complete design package for the works, and the construction of the works is a project responsibility. An approved Environmental and Social Impact Assessment (ESIA) together with the land and rights package for the works, has been completed and received as part of the Eskom details design package responsibility.

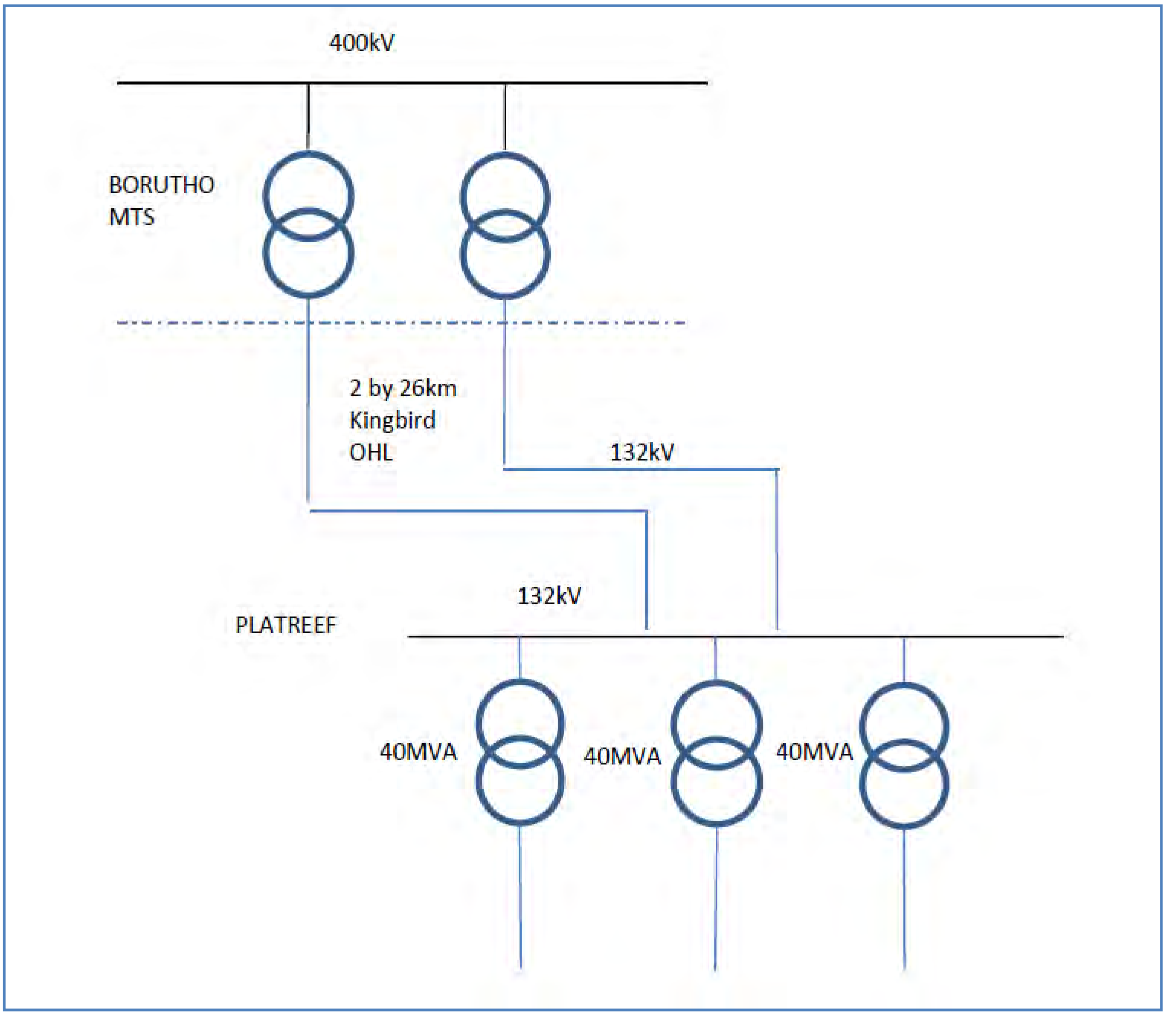
Upon completion of the works, these will be handed over to Eskom to form part of the utilities supply network. As represented in Figure 18.9 and Figure 18.10, the Platreef Project is to be fed from the Eskom Borutho Main Transmission Station (MTS). Two 132 kV overhead line (OHL) feeder bays have been provided by Eskom.

Figure 18.9 Eskom Supply Network



ESKOM, 2017

Figure 18.10 Bulk Power Scope of Work



DRA, 2017

From the Borutho MTS, 2 x 26 km Kingbird 132 kV OHLs are to be constructed to feed the Platreef 132/33 kV substation. At the Platreef 132/33 kV substation, three (3) x 132/33 kV 40 MVA transformers are to be installed with future capacity for the installation of a further 40 MVA transformer.

The supply is designed to provide N+1 redundancy on both the OHL and the transformers for up to 120 MVA. A future 4th transformer is catered for, the need for a 5th transformer will be determined as the mine development continues in order to maintain redundancy with a forecasted future NMD of 131 MVA.

The forecast completion of construction of the OHL (Overhead Line) and substation is Q3 2023. The final budget quote, self-build and electricity supply agreements have been drafted for inclusion in the capital estimate.

Construction of the overhead lines and substation for the 100 MVA supply has commenced and is due to be completed in Q3 2024.

The electricity supply agreement caters for a ramp up period. From an initial supply of 15 MVA in 2023, the supply NMD will be increased as required for the load build up. Eskom indicated that a supply increase past 120 MVA is not foreseen as a problem, as Kusile power station will be fully operational beyond 2023. Application for increased power demand can commence when the 100 MVA agreements have been concluded.

18.7.3 Alternative Power Supply

In addition to Eskom bulk power supply agreements and as part of the long-term sustainability plan, Ivanplats are investigating alternate, renewable sources of power for the mine. Ivanplats have previously indicated their interested of becoming an off-taker of up to 80 MW of renewable energy from an independent power producer (IPP). The qualifying criteria would be that at inception, all energy sold to Ivanplats will be at prices below the Eskom Megaflex tariff structure. The annual tariff increase will be limited to Consumer Price Index (CPI), resulting in a predictable long-term energy pricing forecast as well as mitigation against Eskom's future pricing risk.

Regulatory uncertainty in South Africa inhibited the development and investment in local IPP companies. Recent promulgation of regulations allowing the licensing of power plants up to 100 MW has resulted in Ivanplats approaching the market with a view to understanding the available photo voltaic solutions that are options for an alternative or supplementary power supply. An area of 27 hectares within the property has have been identified as a possible site for such a plant.

Other benefits associated with the PV solution will include a lower carbon footprint and anticipated operating cost savings.

With the rapid advancement of energy storage technology, it is envisaged that a second phase of this project could provide energy storage capacity and add significant flexibility in terms of energy usage during the peak tariff periods. Such an agreement would contribute positively to the long-term sustainability of the Platreef Project.

18.7.4 MV Distribution to Surface Substations

Power will be distributed to the following substations at 33kV. A dual supply to each substation is allowed for:

- 33kV Mining Substation
- Vent Raise No 1 33kV substation
- Vent Raise No 2 33kV substation
- Vent Raise No 3 33kV substation
- 33kV 4.4 Mtpa Concentrator substation
- 33kV 770 ktpa Concentrator substation
- 33kV Crushing substation

- 33kV Backfill Plant substation
- 33kV Kell Plant substation (not equipped).

18.7.5 Construction Power Supply

The existing 5 MVA power supply is sourced at 33 kV from Eskom, the South African national power utility. Construction power will be drawn from this supply.

Power for the various construction activities on site are obtained from the substation at 11 kV or the 33 kV OHL as appropriate.

18.7.6 Alternative Power Supply

Application has been made to Eskom to increase the NMD of the existing supply to 8 MVA.

18.7.7 Emergency Power

Emergency backup supply is to be supplied from a 20 MVA 11 kV generator plant consisting of multiple containerised prime rated 2.5 MVA generators. The containerised generator solution was preferred as it is a simpler build up in single unit increments as required and is amenable to the phased construction approach. A single feed to the consumer substation has been provided for.

18.8 Earthworks

18.8.1 Earthworks - Phased Construction

The various phases of the project require construction of the terraces for the erection of the various structures, drainage systems, perimeter berms and the stockpiling of ores, construction materials and various wastes.

18.8.2 Terraces

The founding design provided as part of the geotechnical report suggests blending the excavated material with G4 material in order to achieve a required bearing capacity. Allowance was made to produce some of the G4 material required by crushing and screening the excavated material and the balance from a commercial source.

Due to the geotechnical conditions in the mine area and the different bearing capacity requirements, it necessitates splitting the terraces into three terrace groups comprising of low, medium and high specification terracing.

Low specification terrace design will be used in laydown areas and temporary terraces required for the construction phase of the project, with a bearing capacity of 50 to 80 kPa.

Medium specification terraces cater for typical permanent structural surface infrastructure, buildings, workshops and stores, where the design requires a bearing capacity of 150 kPa with minor expected differential settlement.

High Specification terraces cater for large and high structures or vibrating structures where the design requires a bearing capacity of 250 kPa with no differential settlement. The design prescribes a deep excavation down to refusal level on bedrock.

Terrace will be constructed in line with the project time lines.

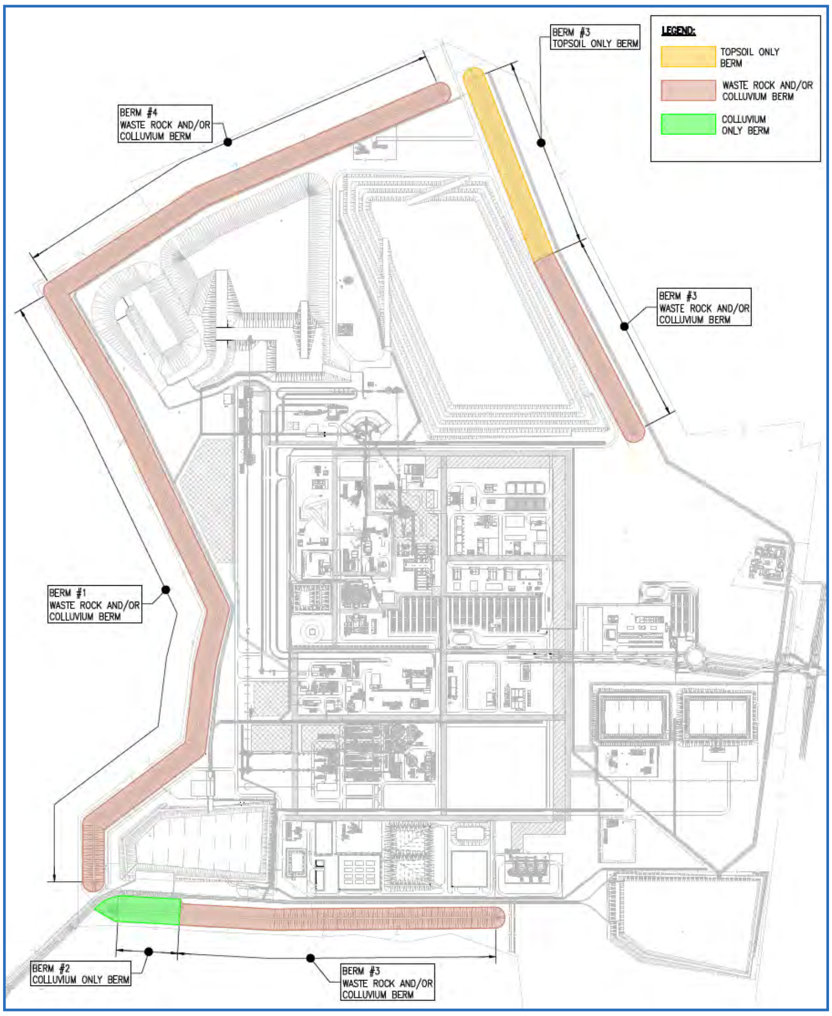
18.8.3 Perimeter Berm

To mitigate the noise, dust and the visual impact of the mine, the relevant specialist studies were performed, modelling an approximate 7.5 to maximum of 10 m high berm on the periphery of the mining area as shown in Figure 18.11. The since-approved Environmental Management Plan (EMP) and subsequent Environmental Authorisation (EA), have both been granted on the notion that this berm will be constructed.

This perimeter berm doubles as a suitable stockpile for overburden generated during construction as well as surplus waste rock from underground development. One of the key visual impact study requirements is to clad the berm with topsoils and to hydro-seed the completed sections as the berm is being built.

The perimeter berm is divided in four sections with the sections scheduled to be built at different times.

Figure 18.11 Perimeter Berms



DRA, 2017

This design still includes flexibility to change the sequence of building the berm should later developments require the project to do so.

The Department of Water and Sanitation (DWS) has previously assented (following a presentation of the design on the 1 July 2016) to the use of a barrier system without a Class C (single geomembrane) liner system. The study design and cost estimate only considered the site preparation clay liner and subsoil drainage for the berms. The assumption was made that the berms will be constructed as spoil berms with construction spoil material and or waste rock placed by operations. A slope stability analysis has not been done on all the different possible fill scenarios as part of the study and it is recommended this be carried out before proceeding with construction.

18.8.4 Material Storage Stockpiles

The ore and waste rock stockpiling encompasses the stockpiling of all commodities generated during both construction and operational phases, except for topsoil (topsoil will be stored in the perimeter berm). These commodities include:

- Low-grade material
- Ore
- Waste rock uncrushed and crushed
- Subsoils (soft and hard)
- Temporarily imported G4 material for platform construction.

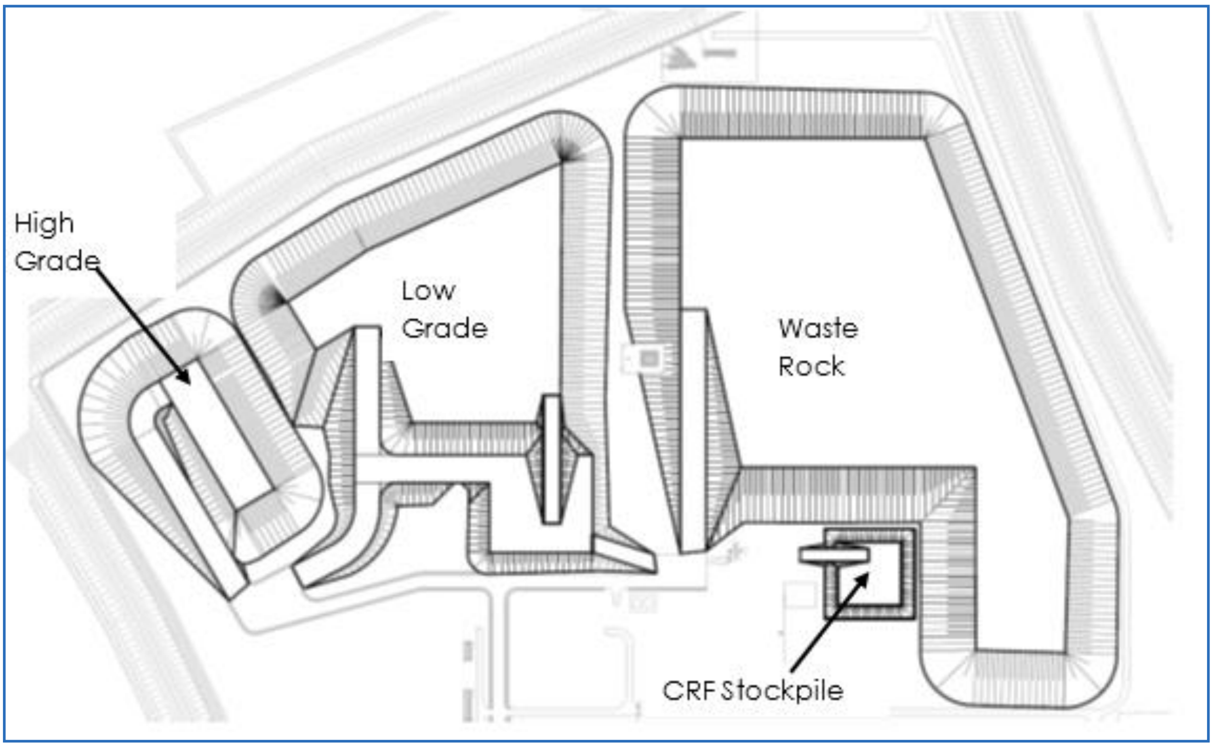
The overall stockpiling strategy in terms of design and operation thereof has been done in line with the EMP requirements for minimising dust and visual impact. Thus, the height at which material is discharged from conveyors for stockpiling purposes is to be kept at a minimum. Materials deposited onto low temporary stockpiles are manoeuvred further by means of mobile earthmoving equipment (trucks, loaders and dozers) to build the individual type of stockpiles.

All stockpiles have been designed with the required footing and drainage as per the requirements stipulated in the IWULA application. The footprint of each stockpile was modelled to optimise the footprint for the required capacity.

A further requirement for waste rock is to re-use the waste rock as earthworks backfill material and as CRF for underground backfilling. To allow for this, waste rock has to be crushed and screened to suitable sized materials. A mobile crushing and screening plant will be established with the CRF plant to facilitate this.

The previous site selection process placed the ore and waste stockpiles on the northern periphery of the site, and provided for a High Grade Ore Stockpile, Low Grade Ore Stockpile and Waste Rock Stockpile, configured as shown in Figure 18.12 below.

Figure 18.12 Material Storage Stockpiles



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During the PEA, the Waste Rock Stockpile site was identified as the preferred location for a Dry Stack Tailings Facilities for the small, Phase 1 concentrator.

The phased development approach reduces the requirement to stockpile high grade ore prior to commissioning, due to the change in the ramp up profile of the mine.

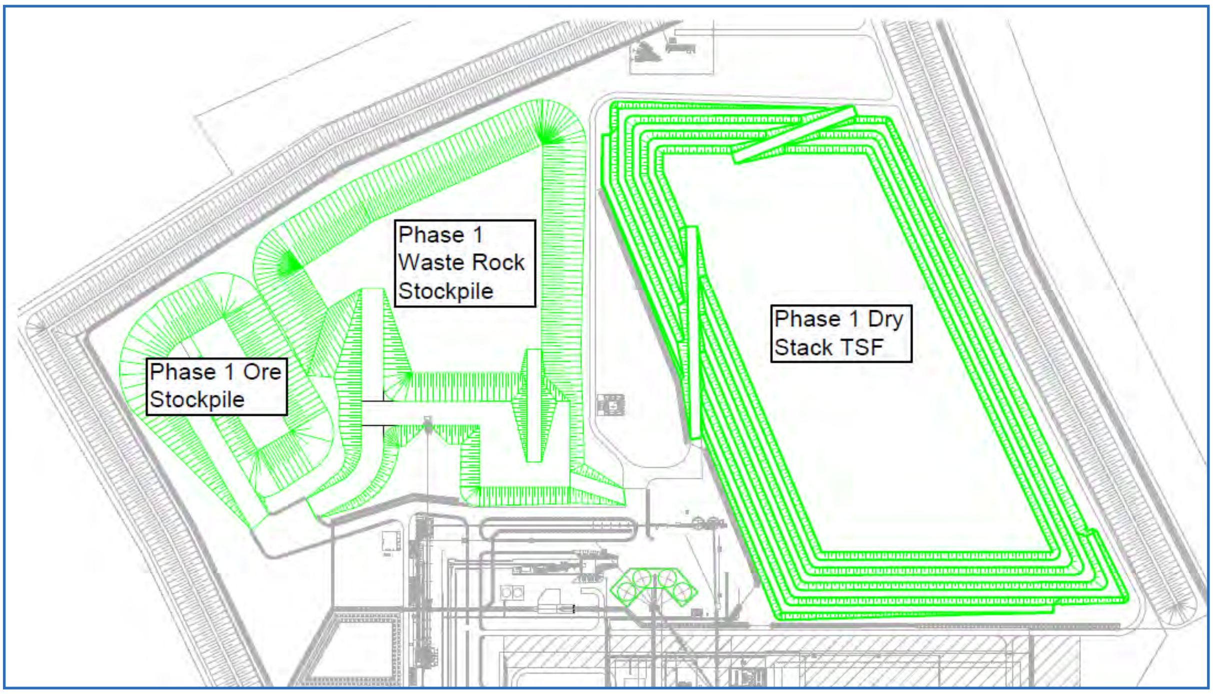
Thus, the concept from the PEA was adopted whereby the stockpiles would be reconfigured to allow the use of the High Grade Ore Stockpile footprint for both High Grade and Low Grade ore. Prior to commissioning of the 770 ktpa plant, the northern section of this stockpile is to be utilised for Low Grade ore and the southern section for High Grade ore. This will facilitate the introduction of the High Grade ore into the Phase 1 crushing circuit during commissioning and production ramp up.

It is not intended that significant amounts of High Grade ore be stockpiled or buffered once the concentrator plant is operational, and ore stockpile will primarily be dedicated to Low Grade ore storage from 2025 onwards.

Waste rock generated during the first phase of the project will be directed first to the 170 kt stockpile within the Shaft area, from where waste will be reclaimed to the crusher plant for use in the CRF plant. Surplus waste rock that cannot be directed to this stockpile will report to the Waste Rock stockpile.

The configuration of this area, showing the full extents of the first phase stockpiles and Platreef TSF are indicated below in Figure 18.13 Platreef TSF and Stockpile configuration.

Figure 18.13 Phase 1 Stockpiles and Dry Stack TSF Configuration



18.8.5 Earthworks Commodity Flow Philosophy and Strategy

On account of the various earthworks commodities that occur and are required during the construction phase, a basic commodity flow model was developed for the project. The commodity flow directly affects the capital estimates and is therefore important to illustrate. The intent of this model is to indicate the net quantities of material and commodities that would need to be stockpiled whether temporarily or permanently. It further indicates commodity requirements, and if importing of commodities is required onto the site.

The different type of earthworks commodities relevant are as follows:

Topsoil

Topsoil is defined as the first 200 mm of material removed (cut to stockpile) from any excavation done on surface. Topsoil may only be used to clad the perimeter berms. The remainder of the topsoil needs to be stockpiled in a defined area for topsoil storage only. A portion of the perimeter berm will be used for this and will be demarcated as such.

Subsoil (Soft Excavation)

Subsoil is defined as the next type of soft soil material removed (cut to stockpile) from excavation after the topsoil has been removed. Subsoil could also be used in certain circumstances as fill material if suitable. If not suitable for fill material, the subsoil will be stockpiled within the perimeter berm and / or in the waste rock stockpile area.

Hard Excavation

Hard excavation is the removal of material within the excavation which cannot be efficiently removed or loaded by track-type excavator and, which can only be removed / excavated once it has been blasted. The material from this type of excavation could also be classified as waste rock. Where possible, this material will be crushed and used as fill material. Excess hard rock will be stockpiled within the perimeter berms or stockpile area.

G4 Material

G4 material complies with certain material properties and classification that make it suitable for use in terrace construction. G4 material is either obtained commercially (off site crushing operation) or on site by crushing available waste rock.

Waste Rock

Waste rock is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content below the mine cut-off grade.

Ore

Ore is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content greater than the required minimum concentrator mill feed grade.

Low-Grade

Low-grade is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content lower than the required minimum concentrator mill feed grade.

18.8.5.2 Surface Earthworks Material Flow

Earthworks material from the excavations created by surface construction activities can be divided into two categories either:

- Cut to stockpile is loading of material generated from excavations, hauled, dumped and stored in a dedicated spoil or stockpile area.
- Cut to fill is loading of material obtained from stockpiles or commercial sources, hauled to the required position, processed, levelled and compacted to the required specification.

The envisaged surface earthworks material was modelled considering the type of commodity and category. The total amount of topsoil available is estimated to be 386,896 m^3 which could be used to clad the perimeter berm and stored for future rehabilitation.

The design seeks to optimise the use of excavated material as fill. The material balance flow concluded that a total of 274,866 m³ of material during Phase 1 and 972,274 m³ of material during Phase 2 are required to be stored / stockpiled on surface. It is recommended that this material be stockpiled within the perimeter berm areas. Phase 1 will have a hard rock material shortfall of an estimated 267,373 m³, this shortfall will be supplied from the waste rock from the underground mining development.

18.8.5.3 Underground Earthworks Material Flow

Ore and waste rock will be hoisted to surface, the high grade ore will be directed to the Concentrator Plant, while waste rock will be utilised either as CRF or for construction fill. The remaining material reporting to the Waste Rock Stockpile for storage. Table 18.2 indicates the expected underground material flow and volumes will be hoisted to surface.

Table 18.2 Cumulative Surface Earthworks Material Flow

Year	Waste to Surface (Mm³)	High Grade Ore (Mm³)	Low Grade (Mm³)	CRF Required (Mm³)	Construction Fill Required (Mm³)	Waste Stockpiled (Mm³)
2024	0.740	0.105	0.0314	0.078	0.267	0.395
2028	2.088	2.116	0.260	0.832	0.520	0.737
2052	4.960	62.600	4.600	0.832	0.520	3,609

18.8.5.4 Surface Storage Capacity

Table 18.3 indicates the available storage capacity per commodity and the required volume that needs to be stockpiled on surface.

Table 18.3 Surface Storage Capacity

Area	Total Stockpile Capacity (m³)	Required end of Phase 1 (m³)	Required for Phase 2 (m³)	Required LoM (m³)	Total (m³) Spare (+) or Shortfall (-)
Perimeter Berm 1	550,000	0	550,000	550,000	0
Perimeter Berm 2	310,000	0	310,000	310,000	0
Perimeter Berm 3	300,000	130,000	300,000	300,000	0
Perimeter Berm 4	340,000	150,000	340,000	340,000	0
Perimeter Berm Total	1,500,000	280,000	1,500,000	1,500,000	0
Phase 1 Waste Rock Stockpile	2,580,000	394,627	736,667	2,580,000	0
Phase 1 LG Ore Stockpile	710,000	31,393 + 105,420 (HG)	454,834	4,622,000	-3,912,000
Phase 2 Waste Stockpile	5,000,000	0	0	1,028,767	3,971,233
CRF Waste Rock Stockpile	100,000	100,000	100,000	100,000	0

If required, the imported G4 material for layer works will also be stockpiled temporarily in the waste rock stockpile area.

By directing tailings to the Rietfontein site from Year 4 of operation, the availability of northern part of the DTSF area will be preserved for the expansion of the Phase 2 Waste Rock Stockpile from Year 10 onwards. Reclamation of the tailings stored in the DTSF for use as feed to the paste backfill plant after the completion of Phase 2 construction provides further space for the expansion of this stockpile, reverting in essence to the original stockpile concept.

The use of the eastern corner of the property for long term waste rock storage is not preferred as it limits opportunities for the expansion of the mine and would require changes to the stormwater management system.

Based on these material flow summaries and available storage capacities from Table 18.3 the following can be summarised:

- Perimeter berms will be constructed out of 1,247,140 m³ surface materials (topsoil, subsoil and hard material) and 252,860 m³ of underground waste rock required to complete these berms.
- The phased approach to the construction of the berms, ore and waste storage facilities is sufficient to support mining and plant operations during the construction and initial period of operation.
- Sufficient space can be provided for the storage of material hoisted to surface.

It is recommended that opportunities to assist with the establishment of an empowerment company, with the objective of extracting value from the waste as aggregate for commercial concrete batch plants be investigated as this could reduce the required footprint of the waste storage facilities.

18.8.6 Roads, Walkways and Parking Areas

The main access road passing the site is the N11. As this is a national road, it falls under the SANRAL jurisdiction.

The N11 intersection was designed and constructed in line with SANRAL requirements to allow for heavy vehicles turning north and south back onto the N11.

The internal road design philosophy is that as far as possible delivery vehicles will remain on the main entrance road to the required delivery points and parking areas, with internal roads reserved for the delivery of equipment from the stores to the work area. The internal roads and parking take into account the traffic flow inside the mine area. Gates separate areas in order to restrict access, but without reducing serviceability and production.

Different types of surface finish and layer works have been designed for different types of application and road uses within the mine area. The following types have been used as part of the design and layout:

- Asphalt road layer works used for the main entrance access road.
- Paving road Type 1 layer works used for internal access roads in and around the shaft, process plant, workshop and store areas.
- Paving road Type 2 layer works used for parking and walkway areas.
- Concrete road layer works used in the process plant areas between the mill building and the filter building.
- Gravel road layer works used mainly for maintenance access.
- Haul road layer works used for the main hauls required in the rock handling area between the different types of stockpiles.
- Service road layer works used for security access road around the perimeter and area with little expected traffic.

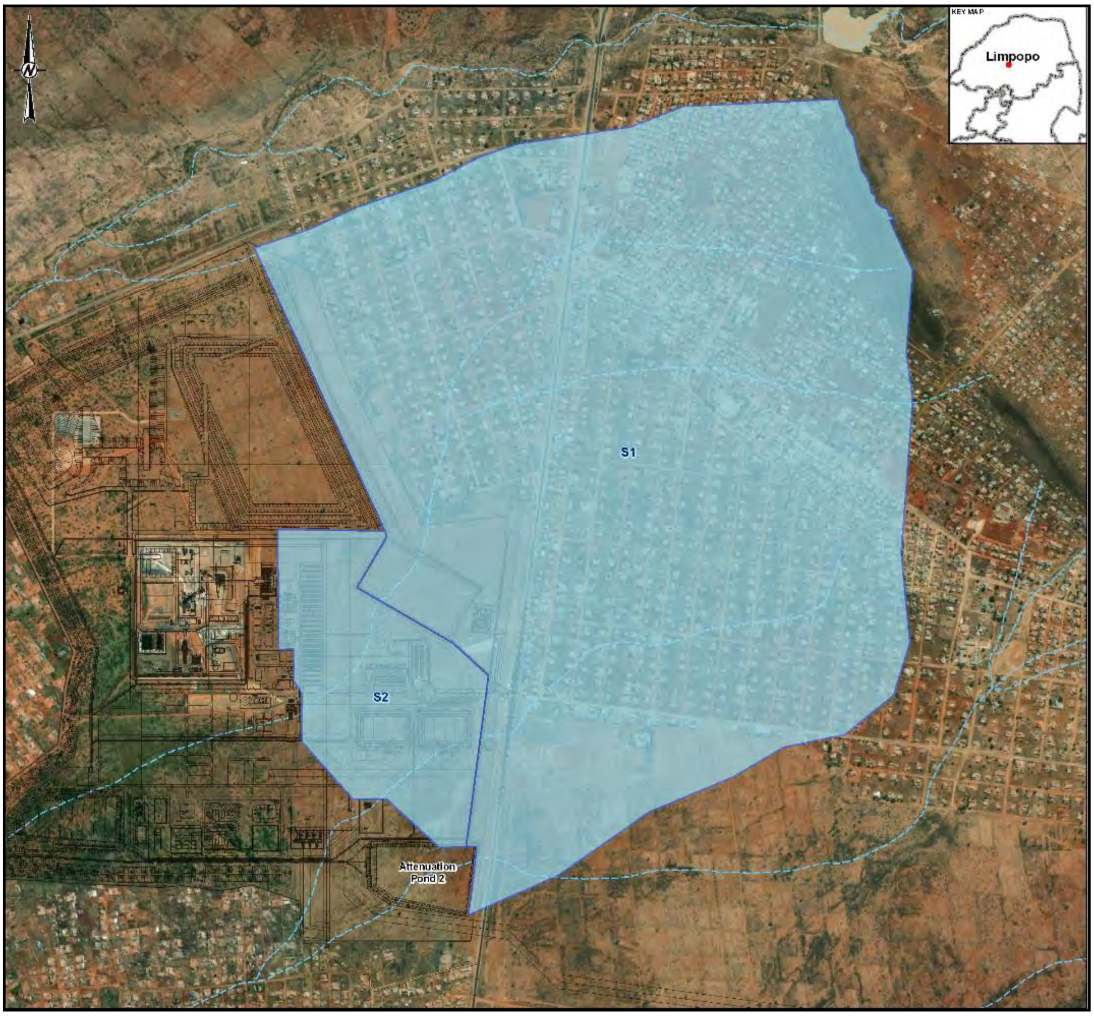
Internal roads during Phase 1 for the 770 ktpa concentrator plant and Shaft 1 mining area have been limited to gravel roads. These roads will be upgraded to paved roads to align with the specifications as listed above during the construction of Phase 2 for full production mining and processing.

The temporary haul roads constructed during Phase 1 are a limited structure that based on the geotechnical conditions are anticipated to require more frequent maintenance until aligned to the above specification in Phase 2.

18.8.7 Stormwater Management

The mine area receives clean stormwater runoff draining from the urban areas located to the north and east of the mine site. Regulation 704 of the National Water Act of 1998 requires that clean stormwater runoff is kept separate from the polluted runoff generated on the site. To meet these requirements, clean and polluted stormwater management systems were designed for the site. The clean water catchment area of this stormwater goes up to the 5 km aloof hillside. It then flows through Magonga, Baloi and Macheke, before it crosses the existing N11 onto the Platreef Resources Mining Site. Refer to Figure 18.14 below, which depicts the north-eastern clean water catchment areas that affect the mine site.

Figure 18.14 Clean Water Catchment Areas



Currently, all run-off rain water flows in a sheet flow fashion, thus being of no consequence. However, when the mine infrastructure is constructed, a concentrated flow of rain water will be created which needs to be managed via a stormwater management system, to negate its possible effect on surrounding communities.

Golder Associates Africa (GAA), as the hydrology consultants, have modelled this runoff and made recommendations for the position of cut-off drains and an attenuation dam, based on the following:

A one-hundred-year flood line is applied and all structures on the mine will be protected against this. A one-hundred-year flood line is a line drawn on a contour plan showing the edge of the water level of a river during flood condition.

A one-fifty-year-flood/stormwater event (1:50) was used to calculate the stormwater run-off and peak flow, to size the required stormwater infrastructure and design thereof. This is a flood event that has a 2% probability of occurring in any given year.

Freeboard of a minimum 0.8 metres has been applied. Freeboard with respect to water storage dams can be defined as the distance between the full supply level and the lowest point on the dam wall crest overflow.

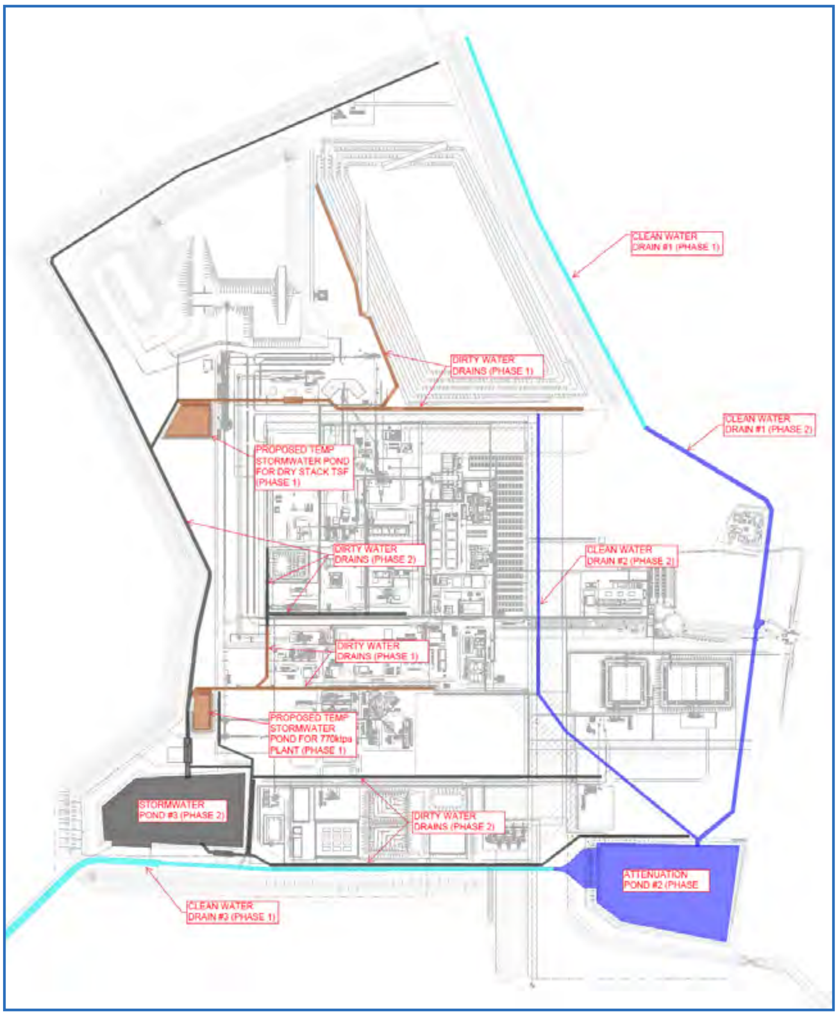
Drainage channels, some up to 14 m wide, and a large attenuation pond have been advocated to delay the release of water, in order to avoid flooding of the southern villages, especially Mzombane.

The stormwater management system consists of the following:

- Clean Water Cut-Off Drains #1 & #2
- Clean Water Discharge Drain #3
- Dirty Stormwater Runoff Drains

Refer to Figure 18.15 for the proposed overall primary stormwater drains and ponds layout.

Figure 18.15 Overall Primary Stormwater Drains and Ponds



18.8.7.1 Clean Water Cut-Off Drains

The clean water cut-off drains #1 & #2 are designed to channel all rain water entering the mine area from the north-eastern side through the N11 servitude, towards the attenuation pond in the south-eastern corner of the mine site. These drains are designed to cater for a 1 in 50-year storm event. Due to the gradient and expected volumes resulting in high flow velocities, these drains are concrete lined.

The stormwater will then be attenuated in the attenuation pond in order to reduce the flow rate in the south-western clean water discharge drain #3. The discharge drain routes the water through the communities which reduces the risk of flooding and / or damage within the community. This drain is design to cater for a 1 in 50-year storm event, however it is only concrete lined to cater for a 1 in 10-year storm event which risks erosion and possible damage of the storm water drain during events larger than the 1 in 10-year storm event catered for.

Inspection of the drain following such events should be included in the mine operational procedures.

During Phase 1, only the northern section of the eastern clean water cut-off drain #1 along the perimeter berm #3 will be constructed in order to divert the clean stormwater away from the dry stack TSF during initial stages of operation. The water will then be dissipated into natural sheet flow across the undeveloped mining area. The attenuation pond will not be constructed during Phase 1, which leads to an increased risk of erosion, damage and / or flooding during major storm events. A section of the south-western clean water discharge drain #3 will also be constructed during Phase 1 in order reduce to risk of damage and / or flooding within the downstream communities, however during major events, this drain might overflow.

The clean water drainage system will be completed during Phase 2 of the project.

18.8.7.2 Dirty Stormwater Runoff Drains

The dirty stormwater runoff drains are a network of drains running through the mining area collecting all runoff water and directing it towards the dirty Stormwater Pond #3. These drains vary in sizes, and all are concrete lined. Primary collection drains are sized to accommodate the 1 in 50-year storm event. These primary drains are the drains separating the various sub-areas (area such as the TSF, stockpiles, shafts, concentrator plants and waste facilities) within the overall mining area. The secondary drains are design to cater for a 1:10 year storm events and these drains are located within the sub areas.

During Phase 1, only the primary drains around the dry stack TSF and the concentrator plant will be constructed. These drains will then feed into two separate temporary stormwater ponds, suitably sized to cater for a 1 in 50-year storm event for the respective catchment areas, while the permanent Stormwater Pond #3 is being construction as part of early Phase 2 construction works. The existing stormwater drains and pond will cater for the inner shaft area.

The expansion and completion of the stormwater drainage system is required to support the mining operations and is to proceed immediately follow the completion of Phase 1.

18.8.8 Water Storage Facilities

Table 18.4 indicates the main water storage facilities, lining type and their capacities in the different areas of the mine.

Table 18.4 Water Storage Facilities and Capacities

Water Storage Facility	Area	Capacity (m^3)	Lined
Raw Water Dam 1	6,000	24,000	HDPE Lined
Raw Water Dam 2	6,000	49,000	HDPE Lined
Attenuation Pond 2	6,000	150,000	Unlined
Stormwater Pond 3	6,000	225,000	HDPE Lined
TSF Temporary Stormwater Pond	6,000	25,000	HDPE Lined
Plant Temporary Stormwater Pond	6,000	8,000	HDPE Lined
Stormwater Pond #3	6,000	225,000	HDPE Lined
Borehole Water Storage Tank	6,000	250	Sectional Steel Tank
Masodi WWTP Pumps Feed Tank	6,000	500	Sectional Steel Tank
Raw Water Supply Filter Feed Tank	6,000	1,000	Sectional Steel Tank
PWT Brine Storage Tank	6,000	1,000	Sectional Steel Tank
Potable Water Storage Tank	6,000	250	Sectional Steel Tank
Clean Water Storage Tank 1	6,000	1,000	Sectional Steel Tank
Clean Water Storage Tank 2	6,000	1,000	Sectional Steel Tank
Dust Suppression Tank	6,000	1,000	Sectional Steel Tank
Plant Potable Water Tank	3,000	45	Sectional Steel Tank
Plant Fire Water Tank	3,000	500	Sectional Steel Tank
Mine Return Water Storage	2,000	500	Sectional Steel Tank
Shaft Potable Water Tank	2,000	250	Sectional Steel Tank
Mine Return Water Dam 1	2,000	1,000	Concrete Tank
Mine Return Water Dam 2	2,000	1,000	Concrete Tank
Shaft Fire Water Tank	2,000	500	Sectional Steel Tank

18.8.9 Services

18.8.9.1 Water Treatment Plant

The area in which the Platreef Mine is situated is commonly known as a water scarce area, and as such the need to conserve water and re-circulate water within the mine site is required.

The raw water source to the mine is expected to originate from a local waste water treatment works (WWTW). During Phase 1 this will be the bulk water supply that will mainly be used by the concentrator plant and underground mining services.

Although agreements will be put in place to manage the quality of raw / treated effluent water supply, there could potentially be instances where the water is not to specification. In order to cater for Out-of-Specification (OOS) raw water, a water treatment / filtration plant was provided for at the mine site.

The raw water supplied from the WWTW will be pumped into a water filtration feed tank, the water will then be disinfected and filtered in order not to have a turbidity of higher than 5.0 nephelometric turbidity units (NTU). The treatment plant is based on a modular design concept to cater for the phased approach. The product water will then be stored as clean filtered water in one of the raw water ponds.

Raw water will be pumped to one of two raw water dams. To accommodate the different water requirements during the phases of the project, a raw water dam of 24,000 m^3 will be construed during phase 1, to be followed by the second raw water dam of 49,000 m^3 in Phase 2.

The clean filtered water will be pumped through a site-wide network of pipes for use within the mine and concentrator plant.

Water from Stormwater Pond #3 will also be pumped into the water filtration feed tank for reuse as clean filtered water.

Water will be abstracted from 4 licenced boreholes within the mine area and will be used as potable water site-wide. This water will be pumped into a borehole water tank, then treated by a potable water treatment plant to the quality described in the SANS 241 standards. The treatment process will consist of ultrafiltration and reverse osmosis. The product water will be stored in a potable water storage tank from where it will be distributed site wide. The brine reject will be stored in a brine tank and then be utilised as dust suppression water on the dry stack TSF. A combined total of 310 m^3/day of water is licensed for abstraction from these boreholes, which will be an adequate potable water supply for LOM. The peak potable water demand is estimated to be 189 m^3/day.

18.8.9.2 Assay Laboratory

Phase 1

An allowance has been made for a containerised laboratory that will be capable of processing metallurgical samples from the 0.77 Mtpa Phase 1 concentrator plant as well as geological, mining and water samples.

The containerised laboratory for Phase 1 includes the following facilities:

- Sample preparation
- Sample analysis by XRF, ICP, AA, Leco and fire assay
- Ancillary services e.g. Safety equipment, Dust and fume extraction

The Phase 1 containerised laboratory will be operated by an external contractor.

Phase 2

A centralised mine site laboratory has been allowed for as part of Phase 2. This laboratory aims at servicing the geology, mining, process plants (concentrators), infrastructure and environmental analysis requirements. This laboratory is a combined facility to receive, prepare, analyse and assay all samples from the relevant departments. However, certain samples would require being sent to an external laboratory for specific control and legislative purposes.

For geology samples, grade control channel sampling and evaluation core sampling has been allowed for as part of an internal geology laboratory. This will include any preparation work required to the point of delivery to the mine site laboratory for assay preparation, and analyses.

Mining samples will be sent to the mine site laboratory preparation section for sample preparation prior to analysis. The ROM sample will firstly be sent to a mini crushing and screening section to reduce the sample mass prior to analysis. This section forms part of the mine site laboratory complex and will be accommodated for externally to the mine site laboratory building.

Concentrator plant samples will be routed to the metallurgical laboratory section for sample preparation, prior to analysis and assaying.

Infrastructure samples comprises of bulk raw water supply, borehole monitoring water, Stormwater Pond #3 dirty water, filtered water, treated potable water and treated sewage effluent water. Samples will be sent to the mine site laboratory for control monitoring purposes.

Environmental samples comprise of surface water, ground water, soil, occupational hygiene and drinking water. These samples will be analysed at the mine site laboratory for control purposes and the applicable samples will be collected and delivered to an external laboratory.

The laboratory will have a unique barcoding system that will be linked to the main Platreef control network.

The Phase 1 laboratory operations will be integrated into the Phase 2 centralised mine site laboratory once this facility has been constructed.

18.8.9.3 Buried Services

All buried services are designed according to SANS these include:

- Earthworks, i.e. trenching (SANS 2000-DP-1)
- Bedding for pipes (SANS 2000-DP-2)
- Bedding for pipes (SANS 10120-2 LB)
- Concrete and miscellaneous metal work (SANS 2001-CC2)

Drawings were created during the 2017FS for each service, detailing the assumed route. The buried services allowed for include the following:

- Potable Water
- Fire Water Reticulation
- Sleeves for electrical and instrumentation cabling

During Phase 1, only sections of the buried services will be installed to cater for Phase 1 infrastructure requirements. These buried services mainly consist of the following:

- Portions of the fire water and potable water reticulation around the 770 ktpa concentrator plant and shaft area, including an allowance for minor branches to supply points within the concentrator plant area.
- Two (2) IT sleeves between Phase 1 infrastructure and 6 electrical sleeves at various road crossings.
- Masodi WWTW treated effluent supply line for bulk water supply to the mine area.
- Borehole water supply.

18.8.9.4 Sewer Reticulation

The domestic sewerage for the mine area is designed so that all sewerage is collected from various points, and then flows through a 100 to 250 NB, depending on the flow, unplasticized polyvinyl chloride (uPVC)gravity fed buried pipe system to a concrete-lined sump at the sewerage plant.

Sewerage sludge from the plant will be placed in drying beds long enough for it to become solid and workable, before it is added to the composting process. Treated sewerage effluent will be discharge to the dirty Stormwater Pond #3 and ultimately recirculated for re-use.

The sewerage treatment plant is based on a modular design concept to cater for the phased development approach. The reticulation will also be constructed in phases to support infrastructure development.

18.8.10 Weighbridges

Two (2) weighbridges have been allowed for the expansion. One is located near the entrance to the plant, shaft and stores area and the second in the 4.4 Mtpa plant area near the concentrate building, specifically to measure concentrate. Both weighbridges are positioned enabling vehicles entering or leaving the area to be weighed without disrupting traffic flow.

A single weighbridge will be constructed as part of Phase 1, to cater for the 770 ktpa concentrator plant.

18.8.11 TSF Pipeline Servitude

The tailings pipeline servitude is split in to two portions. The first portion of the servitude runs inside the mine up to the boundary fence in the south-eastern corner, and the second portion runs from this point at the fence line, crossing the N11, all the way up to the TSF 7.5 km north-east of the mine area. Refer to Figure 18.16 for the tailings line servitude and Figure 18.17 for the proposed cross-section of the tailings line servitude.

In Portion 2 of the servitude a fence line runs either side of the servitude, separating the tailings lines, road, power line, berms and paddocks from the surrounding areas. A power line runs parallel to the tailings lines in order to provide power for the booster pump station and the return water pumps.

Berms run alongside the pipelines in order to catch possible spillages and to guide it towards paddocks. Allowance has also been made to install culverts where required to provide crossing access over the tailings line servitude for people and livestock.

Figure 18.16 Tailings Pipeline Servitude

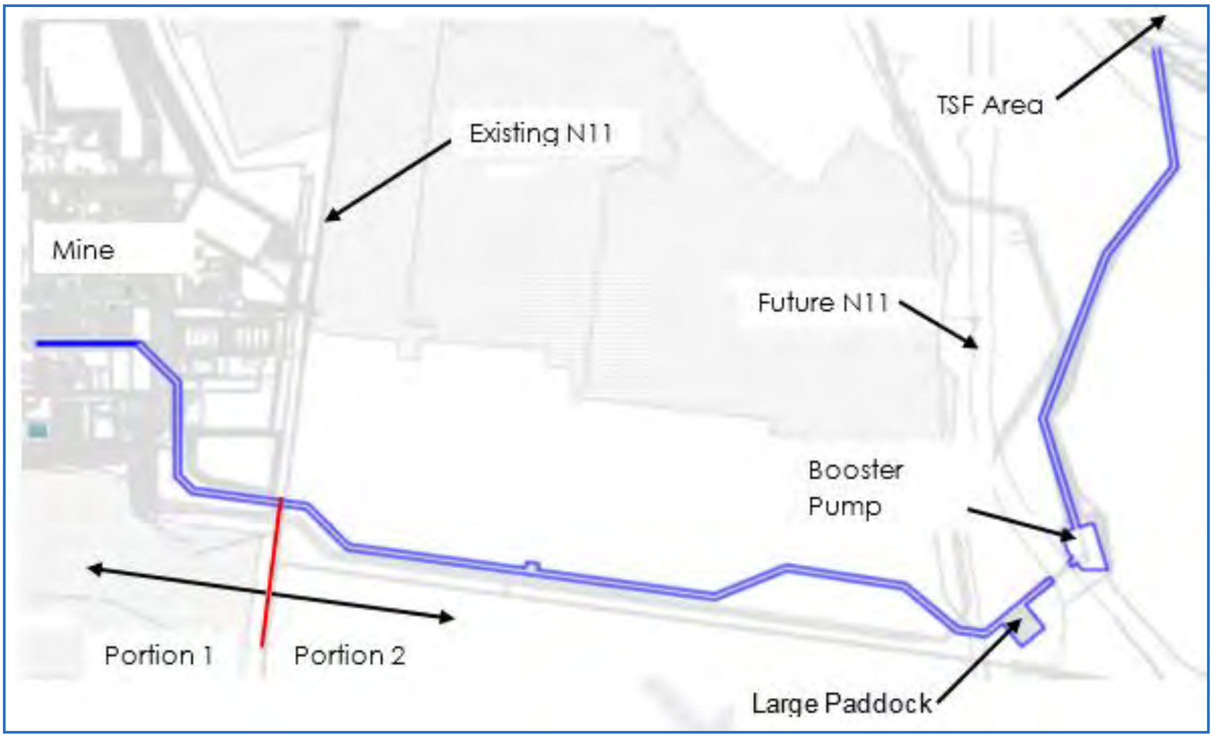


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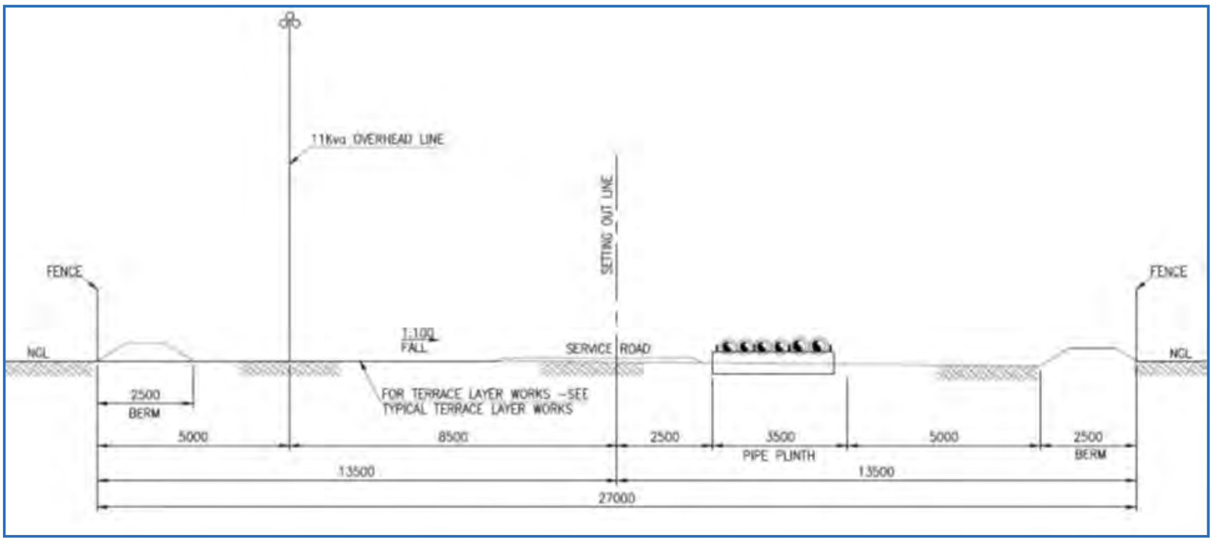
Figure 18.17 Tailings Servitude Section



DRA, 2017; All figures shown in mm.

18.8.11.1 Overland Piping

Tailings Lines

The tailings lines consist of pipelines to accommodate the 2.2 Mtpa and 4.4 Mtpa phases. The first phase is when the first 2.2 Mtpa plant is running. This will utilise one 250NB HDPE-lined mild steel pipeline. The second phase is when the second 2.2 Mtpa plant is brought online, providing a total throughput of 4.4 Mtpa. During the 4.4 Mtpa operation, provision has been made for one off duty and one on standby 300 NB HDPE-lined mild steel pipeline. Tailings from the process plant is pumped to a booster station situated approximately 5 km from the process plant. The total length of the tailings servitude is approximately 7.5 km between the process plant and the TSF. The tailings pipelines are routed in the tailings servitude and are supported on concrete sleepers at 6 m spacing intervals.

Tailings Water Line

The Tailings Return Water consists of two duty and standby 250 NB HDPE pipelines, which run from the TSF Return Water Dam to the Process Water Tank. This pipeline is routed in the tailings servitude, and the TSF corridor servitude and is supported on concrete sleepers at 6 m spacing intervals. The feed line providing clean water to the booster pump station consists of a 250 NB HDPE pipeline.

18.9 Buildings

The buildings have been categorised:

- Architectural Buildings
- Pre-fabricated Building

- Workshops and Stores
- Change House
- Electrical Buildings

18.9.1 Architectural Buildings

Architectural buildings mainly form part of Phase 2 implementation of the project, these buildings will be utilised during the full production.

The buildings are typically constructed out of face brick, complete with aluminium windows and wooden doors. The roof is constructed out of steel and / or timber roof trusses with inverted box rib (IBR) roof sheeting. Included in the buildings are all small power, lighting and furniture.

18.9.2 Pre-fabricated Buildings

As part of Phase 1 execution of the Project, an allowance has been made for prefabricated buildings to support the 770 ktpa concentrator plant. The shaft operations will operate from the existing prefabricated parkhome facilities within the shaft concrete wall area. The development contractor will establish his own temporary prefabricated parkhome and / or containerised site facilities.

18.9.3 Workshops and Stores

The workshops are typically designed as a sheeted steel structure building with civil bases, plinths and a surface bed. These buildings also have filled in brick work on the sides and brick offices and small stores on the inside to facilitate the people working in this building. Roller shutter doors as well as doors and windows are included. Included in the workshops are all small power, lighting, general tools and equipment, furniture and where indicated, an overhead crane.

18.9.4 Change House

The Phase 2 change house is also built as a sheeted steel structure building with civil bases, plinths and surface bed filled-in brick work with all the necessary sanitary facilities, benches, light fittings, extractors, shelves and geyser / boilers. The change house building has been designed in such a way that it encompasses other buildings such as the lamp room, boot wash facility, laundry room, store rooms and offices. In addition, the design provides a logical, sequential flow for employees who are either starting or ending a shift, while ensuring the safety and efficiency of the employees.

18.9.5 Electrical Buildings

The substation buildings are structural steel buildings with IBR roof sheeting, elevated concrete slab and filled-in brick work.

Phase 1 substations for the 770 ktpa concentrator plant (Area 3200), however, will be a prefabricated construction (E-Houses) placed on elevated structural steel supports with concrete foundations.

18.10 Fire Protection

The site-wide fire protection and detection system is based on the fire protection engineering design criteria, as well as compliance with the relevant codes of practice and the local authorities.

Until the commencement of Phase 2, the mine shaft area will utilise the existing fire water system. The 770 ktpa concentrator plant will include a fire pump station connected to a buried network of hydrants and pipes. This system will be fed from a 500 m³ fire water storage tank.

During Phase 2 the 770 ktpa concentrator plant buried ring main will be extended to the 4.4 Mtpa concentrator plant. Additionally, a dedicated fire pump station and 500 m³ tank will be installed to service the entire mine shafts area and the underground workings. The fire water ring main will be connected, with isolation valves, to the plant fire water ring main which also has a 500 m³ storage tank and set of pumps. During pump and / or tank maintenance in one area, the isolation valves can be opened to always ensure availability of fire water.

The tanks will be fitted with dual suction, c/w vortex inhibitors. Each tank section will have an infill, overflow, drain, suction, test return and diesel engine cooling water return line nozzles. The tanks have been designed to supply dedicated firewater via fire water mains to sustain firefighting from 2 hydrants operating simultaneously for 120 minutes.

Pressurised fire water will be distributed around the mining and shaft area, conveyors and localised general infrastructure buildings. Where practicable, fire water reticulation pipework will be buried and will be HDPE class 16 pipe work. Above ground reticulation pipe work will be SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be class 16. All isolation and section valves will be UL listed and FM approved.

Fire hydrants are to be fed off the fire water ring main and to be placed no further than 90 m apart in the required areas. The maximum permissible velocity is 6 m/sec in the hydrant reticulation pipework. Cognisance has been taken to ensure compliance with these limitations when sizing the ring main.

The fire water pump stations will include a diesel, electric and jockey pump delivering a duty of 7,500 litres/min @ 850 kPa.

All fire hydrant, hose reel and protection system risers will be steel and protected accordingly against corrosion.

18.10.1 Fire Detection

All substations and motor control centres (MCCs) have both a smoke detection system in the room as well as a Very Early Smoke Detection Apparatus (VESDA) in the cabinets.

Each building will have its own panel for remote monitoring of its status via potential free contacts.

Each building will be zoned accordingly, requiring a double knock (two adjacent zones) in simultaneous fire condition prior to the discharge of the gaseous suppression system, thus preventing the possibility of accidental discharge. Each panel will also contain potential free contacts used for the shutting down of associated equipment (main incomer, air conditioning system, etc.).

In general buildings such as the control room, a smoke detection system has been included. Each building will have its own panel for remote monitoring of its status via potential free contacts.

Linear Heat Detection cable will be installed along the full length of the conveyors and will be used for belt shutdown only. Due to the fact that the protection system along the conveyors is by means of sprinkler systems, the stopping of the belt in the presence of fire is of paramount importance in order to allow sufficient time for the sprinklers to activate.

The Linear Heat Detection cable is stainless steel braided and has a confirmed temperature initiation system. A simple break in cable or loss of resistance will not initiate belt shutdown unless there is a confirmed high temperature, eliminating the risk of unnecessary or accidental shutdown.

The conveyors have been separated into 200 m sections or zones. Each zone has its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

Flame detectors have been placed at strategic locations and will detect a moving fire in its incipient stage. The detection system will initiate belt shutdown as well as activate the solenoid on the associated deluge valve. Each detection system will have its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

18.10.2 Fire Suppression

Due to the variety of risks associated with this project, a vast number of suppression system types have been designed and catered for. In all instances, the systems comply strictly with the applicable codes of practice, both locally and internationally. Each system has been designed as a fit for purpose solution which protects the equipment and personnel and does not restrict the operation. Table 18.5 is a list of systems and typical locations (to be read in conjunction with the fire protection engineering design criteria (EDC)).

Table 18.5 Types of Suppression Systems

System	Typical Location
Medium Velocity Spray Systems	Lubrication rooms, lube packs, underground conveyors, and hydraulic power packs
High Velocity Water Spray Systems	Transformers
Free Agent Gas Suppression Systems	Substations and MCCs
Foam / Water Deluge Systems	Fuel and lube storage on surface and underground
Hose Reels and Extinguishers	Site-wide, on all structures, in all buildings and are located both on surface and underground

18.11 Electrical, Control and Instrumentation

18.11.1 Design Basis

The electrical design is based on the equipment specifications and electrical design criteria, developed for the project. The electrical equipment is designed or selected to:

- Provide for high plant availability.
- Provide an effective, simple solution which is maintainable by the plant operating personnel.
- Provide a safe working environment for personnel and equipment.

Every effort has been made to optimise the efficient use of energy and to minimise any adverse effects on the environment.

18.11.2 Voltage Selection

As per the electrical design criteria the selected voltages for the project are as follows:

- Medium voltage systems:
  - Distribution voltage: 33,000 V AC resistively earthed
  - Distribution voltage: 11,000 V AC resistively earthed
  - Nominal frequency: 50 Hz
- Low voltage systems:
  - Mobile Fleet voltage: 1000 V AC resistively earthed
  - Motor operating voltage: 690 V AC resistively earthed
  - Motor operating voltage: 525 V AC resistively earthed (legacy system will be phased out)
  - MCC control voltage: 110 V AC solidly earthed
  - Small power LV voltage: 400/230 V AC

18.11.3 Power Factor Correction

The power factor correction (PFC) is to be implemented at the medium voltage level to take advantage of the benefits of scale. The PFC caters for both power factor correction and harmonic filtering requirements. A distributed PFC philosophy has been applied. This solution provides greater flexibility in terms of incremental introduction as the site load increases. PFC harmonic filter banks will be installed at the following locations.

18.11.3.1 Main Consumer Substation

Three harmonic filter PFC banks are to be installed at the consumer substation:

- A single third harmonic filter bank on the left bus.
- A single third harmonic filter bank on the middle bus.
- A third harmonic filter bank together with a fifth harmonic filter bank on the right bus.

18.11.3.2 Mining Substation

Two harmonic filter PFC banks are to be installed at the mining substation:

- A single fifth harmonic filter bank on the left bus.
- A single seventh harmonic filter bank on the right bus.

18.11.3.3 Plant Substation

Two harmonic filter PFC banks are to be installed at the plant substation:

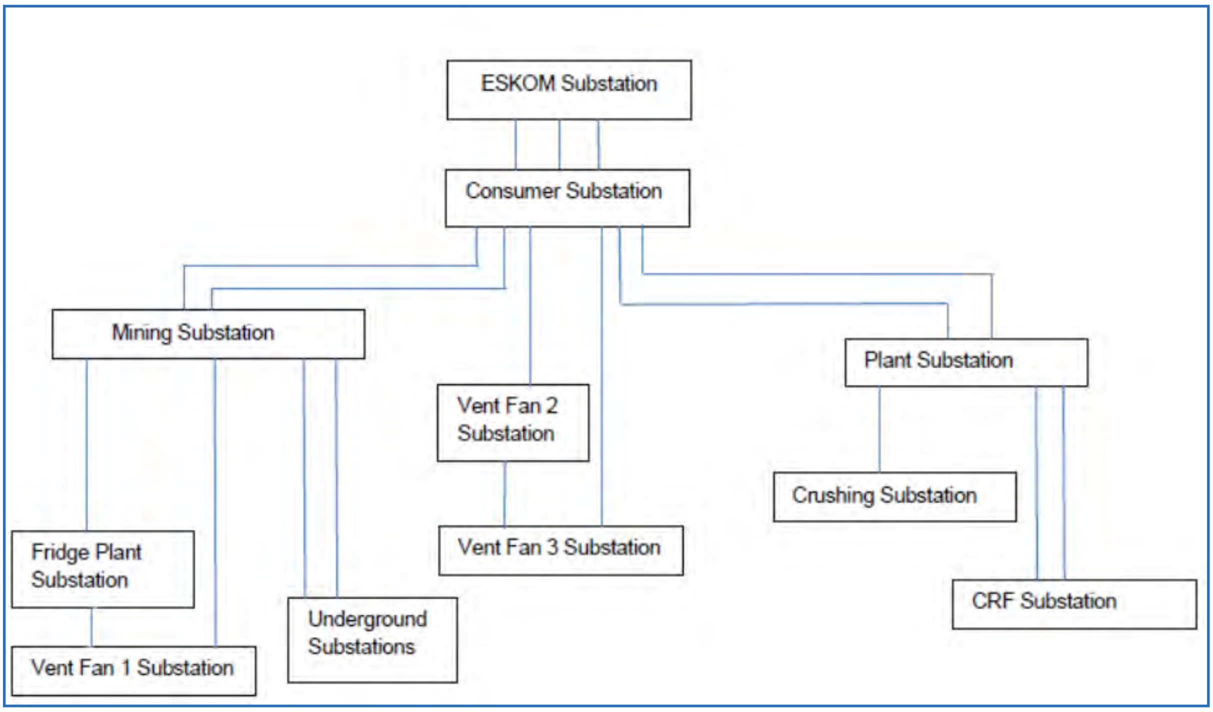
- A single fifth harmonic filter bank on the left bus.
- A single seventh harmonic filter bank on the right bus.

A power quality study will be required, before the final detailed PFC design can be completed.

18.11.4 MV Distribution

The MV distribution from the 40 MVA 110 kV/33 kV Eskom transformers and outdoor yard are included within in the general surface infrastructure. Figure 18.18 indicates the planned MV distribution.

Figure 18.18 MV Distribution



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In the consumer substation double bus 33 kV indoor switchgear is provided, for maximum load distribution flexibility. From there power is distributed on an 33 kV network to the 33 kV mining, plant, and other substations as illustrated diagrammatically in Figure 18.18. Single bus switchgear is utilised in all other substations.

18.11.5 LV Distribution

Power at 690 V has been derived from a suitable number of 630 kVA, 2,000 kVA, 33 kV/690 V Dyn11 ONAN step-down transformers. These will be connected with the neutral point resistively earthed. These transformers are utilised to power the MCCs.

The mine has an existing 550 V reticulation system that will be phased out over time.

18.11.6 Lighting

The non-essential lighting and small power supply in each plant area will be taken from independent sub-boards fed from 33 kV / 400 V and 11 kV / 400 V Dyn11 ONAN mini-substations. These include the numerous offices, workshops, change houses and other similar facilities. The neutral point of the mini-sub 400 V transformers is solidly connected to the earth. In outlying areas where there is a local MCC the small power will be taken from a 690 V / 400 V transformer fed from the MCC.

Only energy-efficient forms of lighting have been utilised, with facilities for person presence detection and/or automated remote switching included as appropriate for further energy savings.

18.11.7 Control System

The control system design is based on the equipment specifications and control and instrumentation design criteria, developed for the project. The control system architecture is designed around a fully distributed Programmable Logic Controller (PLC) and central Supervisory Control and Data Acquisition System (SCADA).

Two surface process control rooms have been catered for the site-wide operations.

The mining control room is located on surface near the main shaft area for the control of daily mining operations on surface and underground.

The surface process control room is located on surface at the plant area for control of the daily plant operations.

The equipment provided within these facilities is detailed in the control and instrumentation design criteria.

18.11.8 Instrumentation

The instrumentation system design is based on the equipment specifications and control and instrumentation design criteria developed for the project.

In general, with the exception of belt-scales and density metres that communicate via a Fieldbus, conventional "hard wired" type instrumentation is used in the design. Instrumentation is based on standard signal types. Instrumentation will be wired directly to weather-proof, field-mounted I/O marshalling boxes (remote I/O boxes or RIO boxes), located strategically around the plant. All RIO boxes will be connected on a fibre link back to the relevant control room.

18.12 Mining Surface Infrastructure

18.12.1 Surface Rock Handling

The rock handling area includes the following:

- Headgear load and transfer conveyor system from Shaft 1
- Headgear load and transfer conveyor system from Shaft 2
- 2 x 2,500 ton RoM and 2x 5,000 ton waste rock temporary stockpiles
- Permanent ore and waste rock stockpiles
- Rock handling mobile fleet
- 5,200 ton RoM silos.

18.12.1.1 Process Description

The Shaft 1 rock handling system handles both RoM and waste rock streams. RoM and waste will be hoisted to surface through Shaft 1 in batches. The surface conveyor system receives material from underground from the centre tower discharge chutes which feed the material on to the Shaft 1 loadout conveyor (2550-CVC-209) via vibrating feeders at a design rate of 300 tph.

The Shaft 1 loadout conveyor then transfers material to a radial stacker conveyor that can be positioned to feed one of four stockpiles. Provision has been made to allow a future waste conveyor from Shaft 2 to transfer waste rock material to the radial stacker conveyor and make use of this system as an emergency throughout.

The radial stacker permits the creation of four distinct stockpiles. It is intended that two stockpiles on the western side of terrace be utilised for RoM, as they are in close proximity to the 770 ktpa modular crushing plant and the eastern stockpiles be used for waste.

Depending on the material being hoisted, waste or RoM, the radial stacker conveyor can be suitably positioned to place material on either the RoM or waste rock stockpiles.

The material placed on the stockpiles is reclaimed using Wheel Loaders which in the case of the RoM material pickup, transport and offload RoM material directly into the RoM bin of the 770 ktpa modular crushing plant.

The waste rock is loaded into trucks by the Wheel Loaders, to be transported and tipped on to the larger waste rock stockpiles on the northern portion of the property.

Both RoM and waste rock stockpiles allow for equipment and personnel safety during stockpile removal operations. Material is placed on one stockpile, while removal operations continue on the other stockpile.

The Shaft 2 rock handling system handles both RoM and waste rock streams. The waste rock stream has a larger maximum material lump size compared to the RoM stream, resulting in different technical design perimeters.

The Shaft 2 headgear bin feeds a wide sacrificial conveyor at a design rate of 1,350 tph. The sacrificial conveyor is purpose designed wider to allow for a slower belt speed that assists with the extraction of scrap metal by over-belt magnets, and with accurate sampling by the cross-belt hammer sampler.

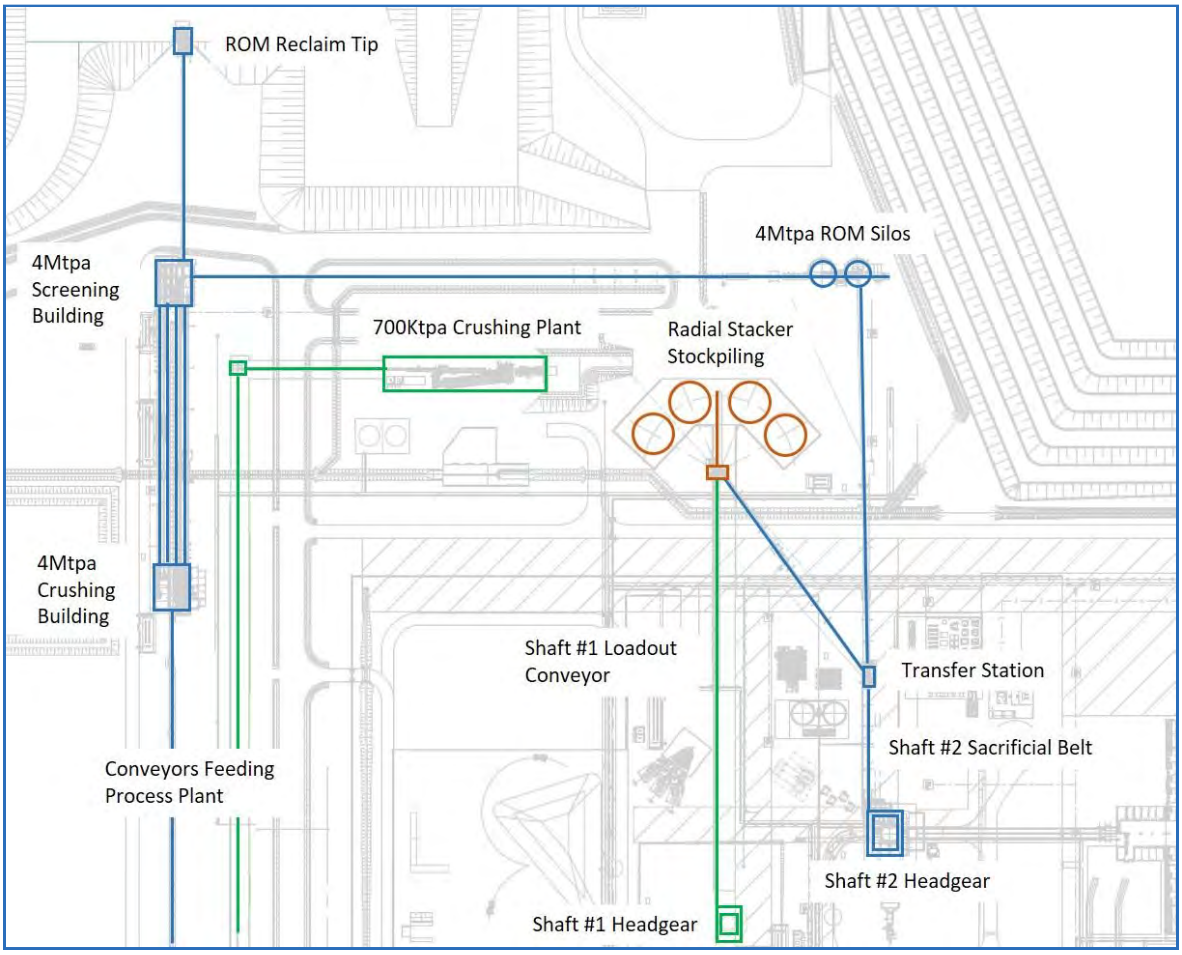
The sacrificial belt feeds a splitter transfer station, intended to split RoM and waste streams. From the transfer tower ore is conveyed to the RoM silos and waste is conveyed, via a transfer tower, to the radial stacker conveyor that feeds the stockpiling area.

As well as allowing the waste rock from Shaft 2 to be stockpiled, this allows a batch of RoM from Shaft 2 to feed to the Phase 1 crushing circuit if required or create an emergency ROM storage (if the ROM silos are not available).

Silos provide storage and buffering capacity, and are placed between the main process activities, namely, shaft hoisting, crushing and screening, and milling. The buffering capacities allow for the independent operation of these process activities, providing overall process continuity and system flexibility during maintenance and unscheduled stoppages.

Figure 18.19 indicates the complete surface rock handling layout for both Phases.

Figure 18.19 Surface Rock Handling



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18.12.1.2 Rock Handling

The following main aspects were considered during the rock handling circuit layout and design:

- Handling of both ROM and waste rock streams,
- Noise and dust minimisation,
- Process dependencies and buffering,
- Modular construction and operation, and

- Maintainability and personnel safety.

An iterative process produced the final rock handling layout. The layout design is a function of the following main constraints that provided the design envelope:

- Mine shaft positions.
- Concentrator plant positions.
- Available mine surface area.
- Proximity to mine boundaries and subsequent local communities.
- Material characteristics and required conveyor profiles.

Various equipment alternatives and positions were considered. The layout allows for both the 770 ktpa and 4.4 Mtpa concentrator plants. The layout considered the constructability of both plants, allowing for minimal disruption between the phases.

18.12.1.3 Mobile Rock Handling Equipment and Stockpiles

Various options were considered for the conveying of ROM and waste rock from the temporary stockpiles to the more permanent stockpile areas. Based on the estimated daily production of ore and waste from the mine shafts, a mobile fleet was deemed most suitable for flexibility and optimal use of invested capital.

The rock handling mobile fleet will consist of:

- Front end loaders,
- 40 t articulated dump trucks,
- 60 t excavator,
- Bulldozers,
- Mobile rock grab and rock pecker.

Haulage roads for the rock handling mobile equipment have been designed to minimise interaction with other mine vehicles while minimising major haulage distances. Haulage road dust suppression has been allowed for.

18.12.2 Explosives Handling and Distribution

The existing surface magazine will be utilised in conjunction with the emulsion system. Preliminary emulsion consumption figures were calculated based on the mine plan, and an external consultant was approached for a design and quote based on the requirements.

18.12.2.1 Emulsion

During Phase 1 of the Project, the PDP, emulsion will be delivered by and stored in a tanker on surface in proximity of Shaft 1. The emulsion will be transferred as required into cassettes / containers and sent underground via Shaft 1.

Permanent emulsion storage silo/s will be installed on surface with a vertical drop system via boreholes to 850 L and 950 L respectively, during Phase 2 of the project. Cassettes will be filled with emulsion underground for distribution. Oxidizer will be handled separately in cassettes via the shaft/s.

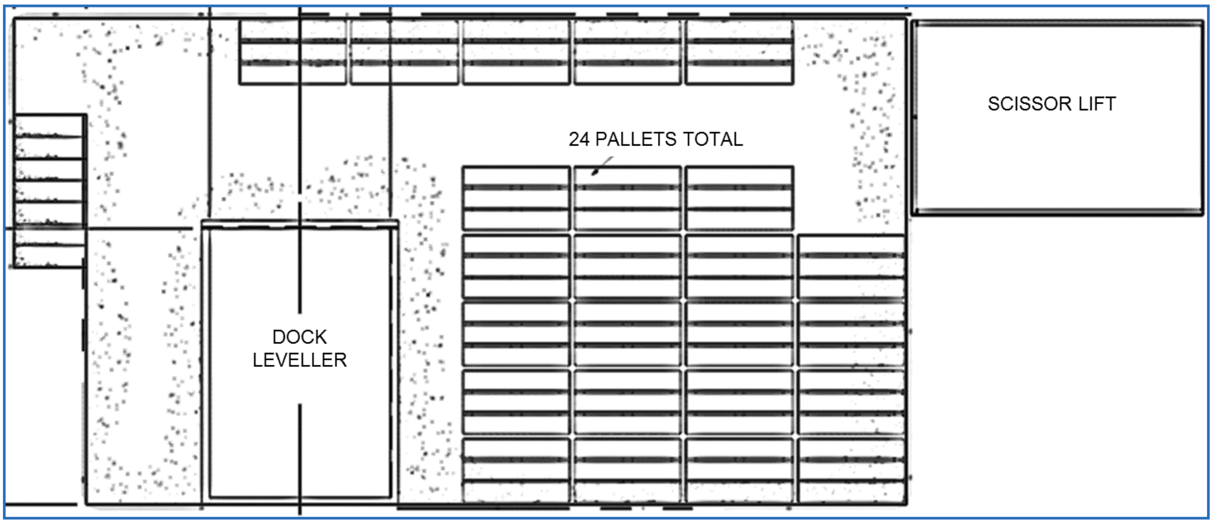
18.12.2.2 Explosives

Explosives and explosive accessories such as blasting cartridges, detonators etc. will be offloaded at a dedicated explosives offloading area. The offloading facilities area will be completely fenced off with lockable gates, warning signs and lights and fire extinguishers.

Explosive will be delivered on pallets. The truck offloading platform is equipped with a dock leveller to assist with offloading the pellets from the delivery truck. The palletised explosives will then be transferred from the platform to a scissor lift that will lower it to the ground level from where it will be packed into UV cassettes.

Figure 18.20 to Figure 18.22 show the layout of the offloading area and indicate the dock leveller, the pallet stacking and the scissor lift.

Figure 18.20 Explosives Offloading Facility – Plan View



DRA, 2017

Figure 18.21 Explosives Offloading Facility – Front View

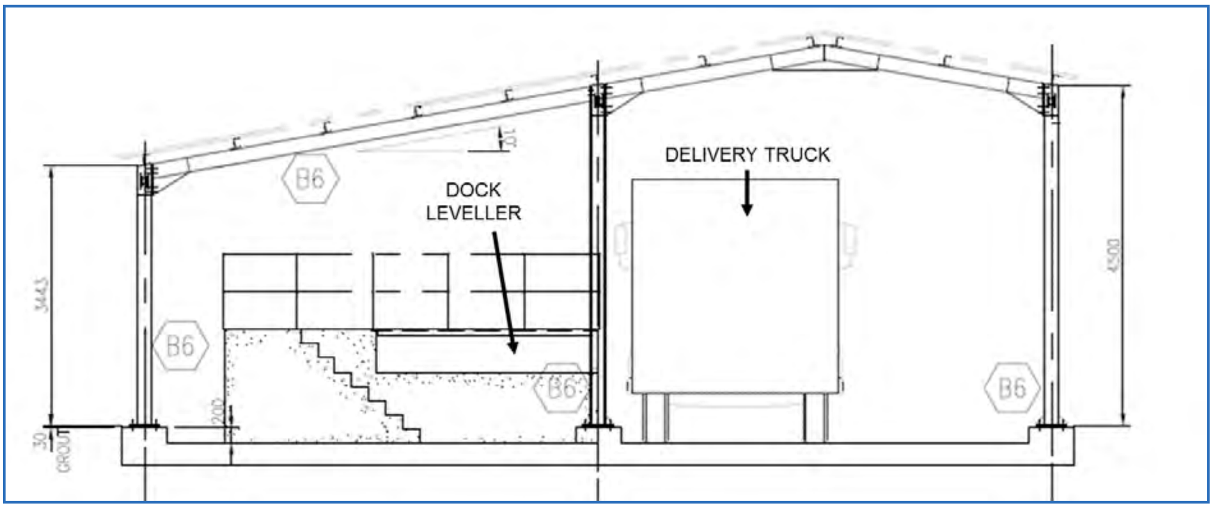


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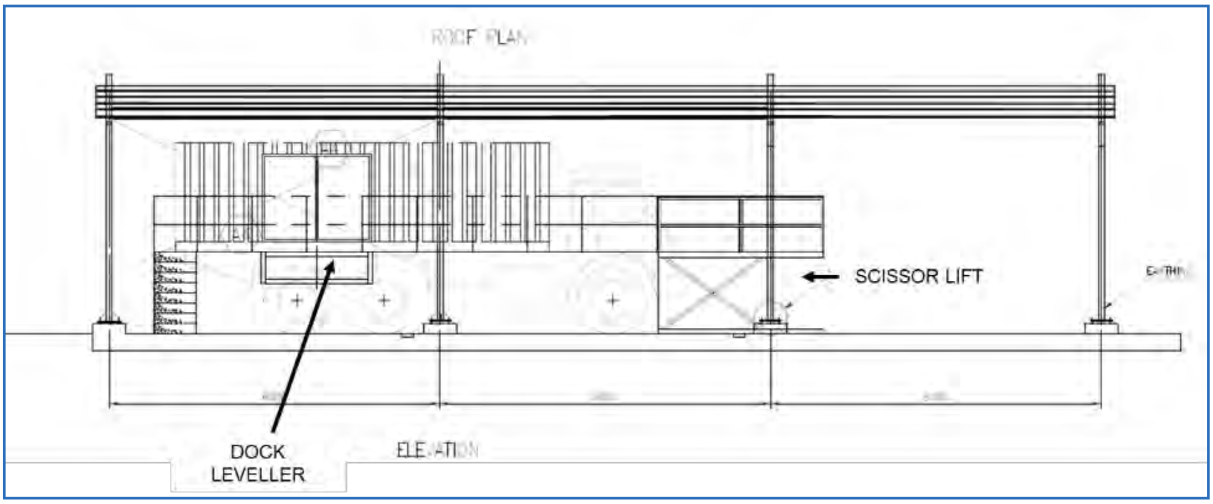
Figure 18.22 Explosives Offloading Facility – Side View



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18.12.3 Fuel and Lubrication

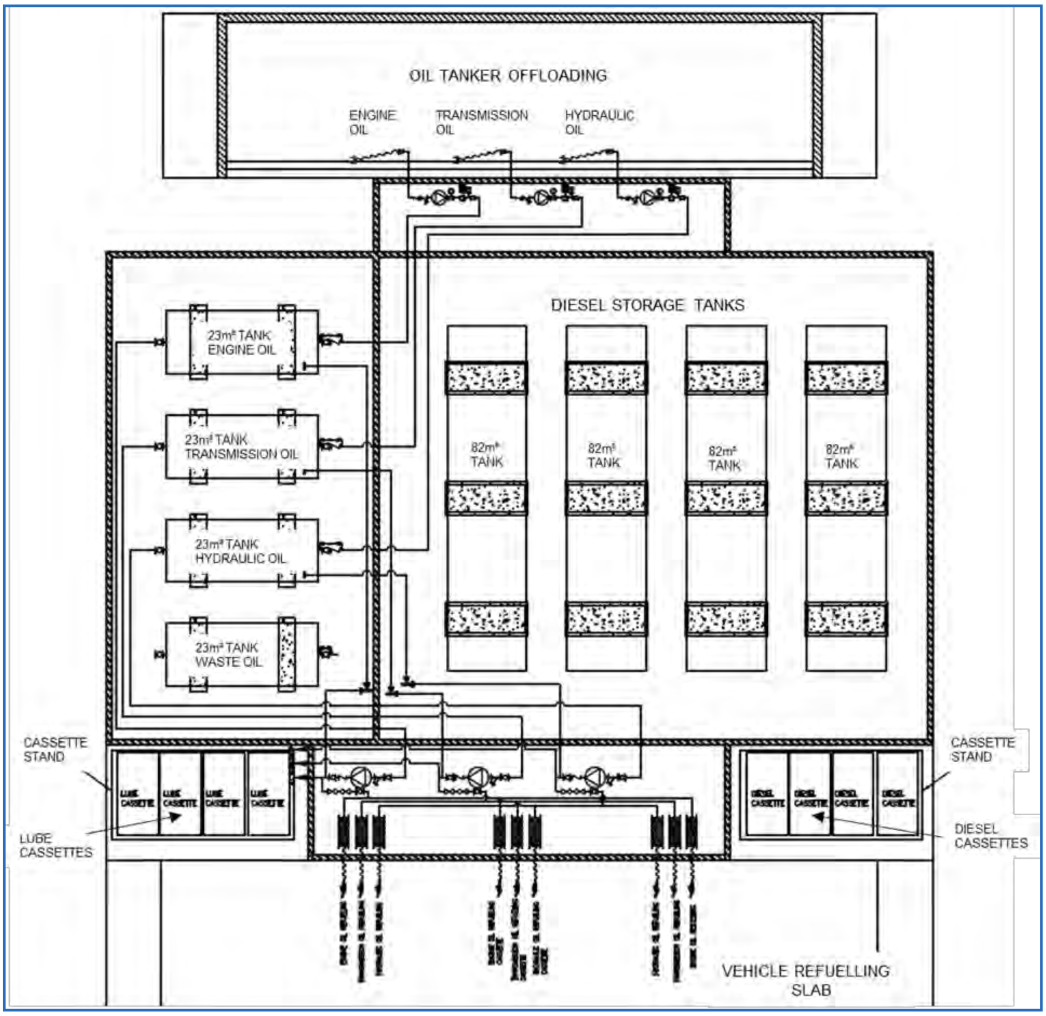
The fuel and lubrication system are a shared service between the surface and mobile underground fleet.

A separate fuel system is included at the emergency generators to supply fuel to the generators only.

The surface vehicles will refill at the depot. Underground vehicles will refill at refuelling stations. Fuel will be piped to the three main levels, and lubes will be transported in lube cassettes through the shaft material transport system.

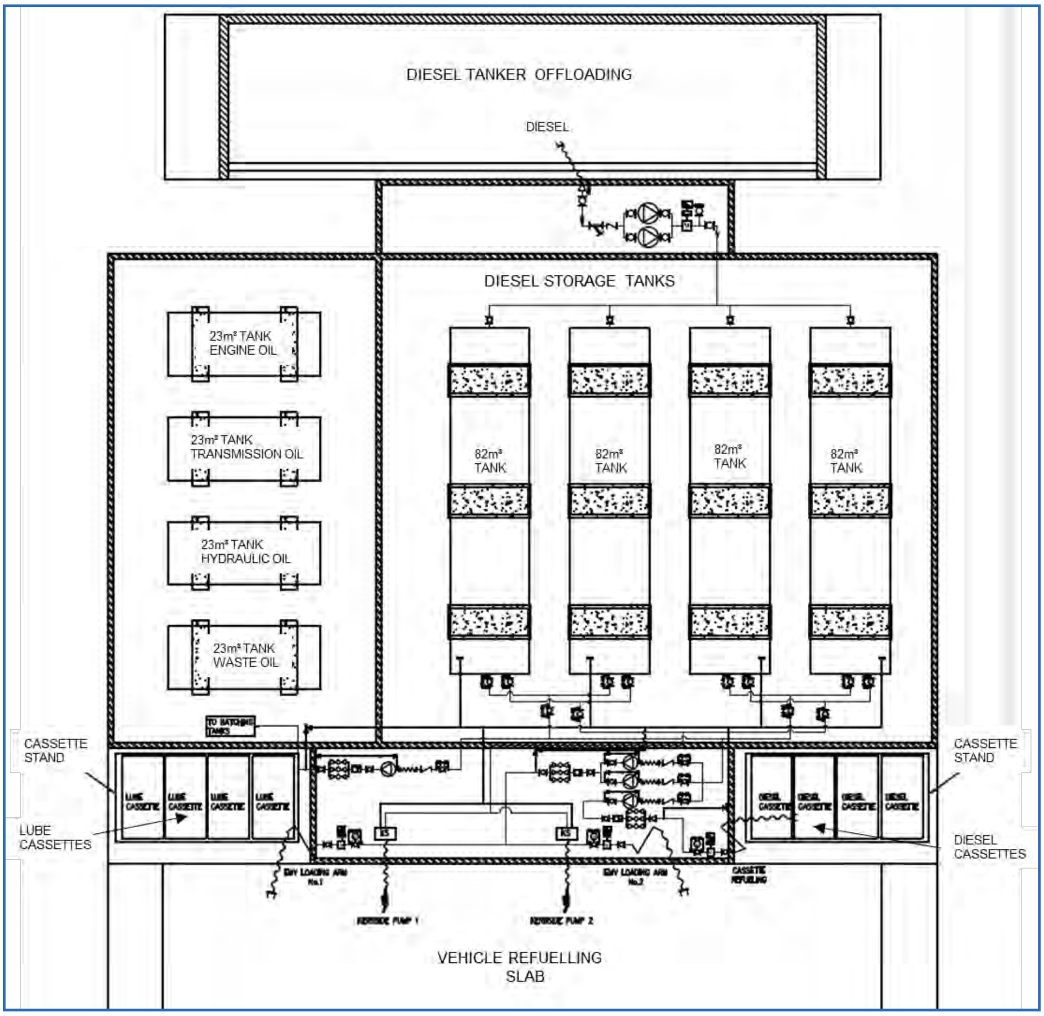
Figure 18.23 illustrates the plan layout of the surface refuelling and storage depot and indicates the lubricants offloading and dispensing. Figure 18.24 is the same layout, indicating the diesel receiving and dispensing.

Figure 18.23 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Lubes Receiving and Dispensing



DRA, 2017

Figure 18.24 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Diesel Receiving and Dispensing



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Fuels will be stored in bulk tanks and will be bunded in concrete bunds as per the relevant SANS standards.

Fuels will be dispensed on surface to EMV and LDV vehicles

Surface storage tanks were designed with sufficient fuel storage capacity to operate the mine for approximately 10 days with 4 x 82 m³ tanks = 328 m³.

The surface facility makes provision for:

- EMV Refuelling Points x 2
- LDV Refuelling Points x 2
- Cassette refuelling Point x 1

Bulk lubes will be stored in bulk tanks and will be bunded in concrete bunds as per the relevant SANS standards. Bulk lubes will be dispensed to the surface workshop and will be transferred to the underground fuel storage tanks using "cassettes" on a continual basis – no batching will take place via piping.

Three grades of oil will be used:

- Engine Oil
- Transmission Oil
- Hydraulic Oil

Surface storage tanks were designed with sufficient fuel storage capacity to operate the mine for approximately 30 days (per grade of oil). The tank sizes per grade of oil are as follows:

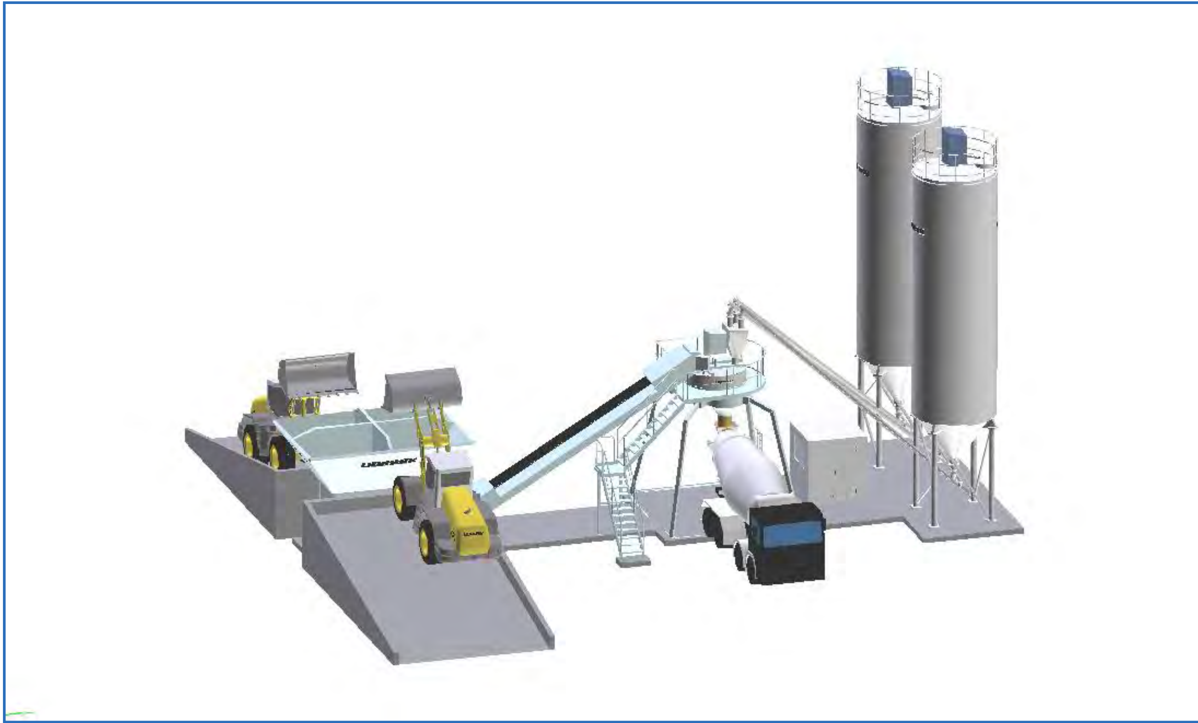
- Engine Oil: 1 x 23 m^3 tank
- Transmission Oil: 1 x 23 m^3 tank
- Hydraulic Oil: 1 x23 m^3 tank
- Waste Oil: 1 x 23 m^3 tank

18.12.4 Concrete and Shot Crete Facility and Distribution System

Shotcrete and concrete will be mixed in a batch plant located on surface established during Phase 1.

This plant is designed to adjust the mix according to the strength requirements with flushing facilities to clean out the complete system prior to any changeover. Figure 18.25 show the layout of the surface shotcrete and concrete batch plant.

Figure 18.25 Surface Shotcrete and Concrete Batch Plant



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The batch plant is equipped with a ring-pan mixer, which can mix various specifications of concrete including mixes incorporating steel and synthetic fibres. The mixer is capable of producing shotcrete, ready-mix concrete, mortar and soilcrete.

Production capacity varies up to a maximum of 60 m³/h depending on the mixing time, materials used and mix specification.

The aggregate batcher consists of four compartments for batching of four different materials which are weighed on a weighing conveyor then conveyed via an incline belt conveyor into the mixer.

Cement is stored in two silos and transported in batches by a screw conveyor into the mixer.

The plant includes an admixture system that is for two liquid admixtures including tanks, pumps and scale. This plant can be equipped with a dosing system to add steel or synthetic fibres onto the weighing conveyor. The system will weigh and dose fibres automatically according to the mix design. Fibres are mixed in the mixer to ensure homogeneous concrete and the best spraying result.

While not included a micro silica dosing system can be retrofitted to make it possible to use fit for purpose concrete specifications. Micro silica is dosed from bulk bags to the scale.

The concrete is mixed into a wet sludge and discharged into one of two boreholes feeding the underground receiving areas, located in dedicated excavations.

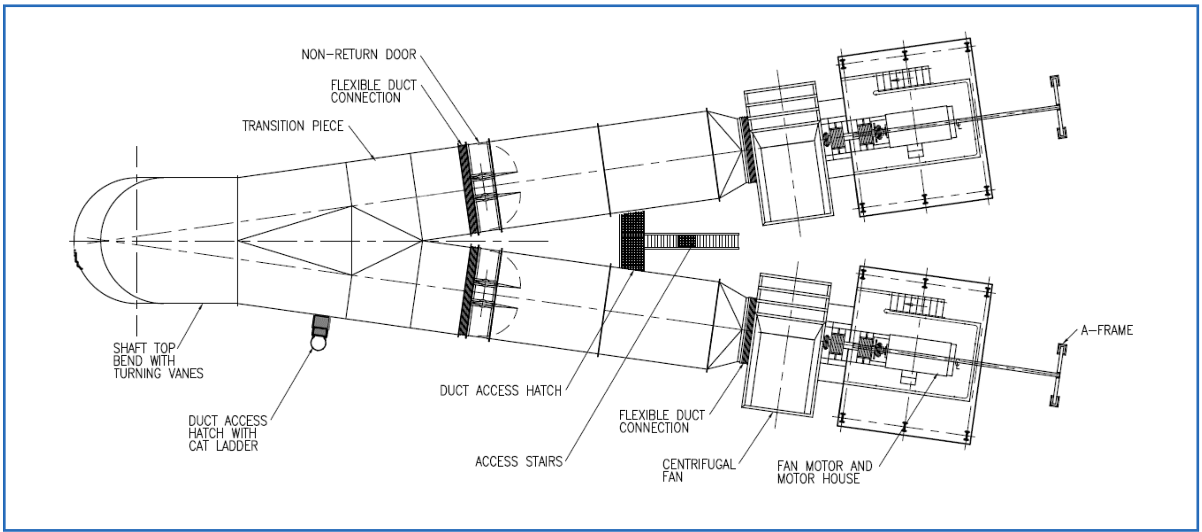
18.12.5 Vent Raise and Fans

The ventilation system is designed as an exhausting pull system. The two main intake shafts (Shaft 1 and Shaft 2), located at the centre of the mining district, will provide fresh air, while Ventilation Raise 1 (near the intake shafts), Ventilation Raise 2 (in the north), and Ventilation Raise 3 (in the south) will be exhausts.

The overall ventilation system for the project uses two fresh air intake shafts (10 m diameter and 7.5 m diameter) and three primary ventilation raises (6 m diameter each).

Fresh air is reticulated through the underground mine via a pull (exhaust surface fans) system. Each ventilation raise will have two similar exhaust centrifugal fans located on surface as shown in Figure 18.26. The three primary ventilation raises will be used to exhaust 1,500 m³/s of air to surface.

Figure 18.26 Main Ventilation Fans Layout



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18.12.6 Backfill Plants

18.12.6.1 Surface Installation of Backfill Plants

Ivanplats requested Paterson & Cooke to perform a feasibility level engineering study for the Cemented Rock Fill (CRF) and Cemented Paste Fill (CPF) Plants for the Platreef Project. Refer to Figure 18 27 for the surface position of the back fill plants in relation to the rest of the mining and process plant surface infrastructure.

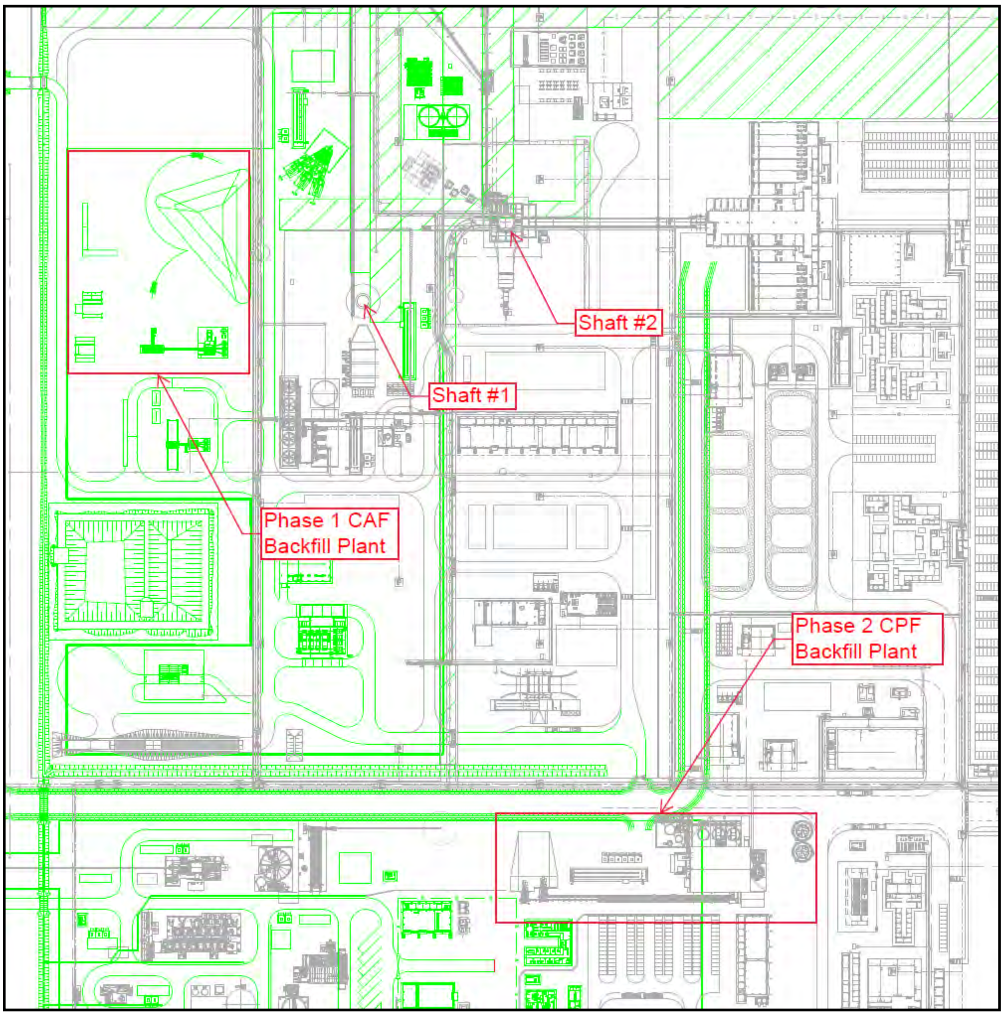
CRF would be the most suitable method of backfill for the initial stages of mining and development and will be implemented during Phase 1 of the project (also referred to as the "Interim Backfill System"), prior to the construction of a concentrator capable of supplying tailings to be used in the formulation of backfill.

CPF was selected as the as the "Long-Term Backfill System" that would serve the life of mine backfilling requirements and will be implemented during Phase 2.

The CPF plant will generally receive tailings from the 4.4 Mtpa Concentrator plant as a pumped slurry, but can also received dried tailings loaded into a feed hopper.

Refer to Section 16 of the technical report for further details on the backfill systems.

Figure 18.27 Shaft Area Services with Backfill Plants



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18.12.6.2 Waste Rock Crushing

The waste rock produced by the mining operation will be stockpiled on surface prior to crushing, this material will be used for cemented rock fill (CRF) for underground usage and G4 type of material for surface usage. A mobile waste rock crusher plant has been allowed for and suitably sized at 130 tph to deliver the required CRF quantities to the CRF. (Refer Section 16).

The opportunity of establishing this mobile crushing and screening circuit earlier than planned to support construction, by allowing it to be utilised for the production of G4 material for surface usage, thus supplementing the contractor's equipment, should be confirmed during the next phase.

The crusher plant will be placed within the shaft area in proximity to the CRF Waste Rock Stockpile and Crushed Rock Stockpile. The mobile rock handling fleet previously described will be used to transport waste rock to the CRF Waste Rock stockpile. The stockpile will be reclaimed by Front End Loader and fed to the crushing plant, which will place the crushed rock on a Crushed Rock stockpile from where the CRF plant will draw material from.

Prior to the availability of the concentrator plant tailings for paste backfill of the underground mine, CRF will be utilised as backfill.

18.12.7 Traffic Flow Study

Simulation Engineering Technologies (SET) was previously engaged to construct a simulation model representing the traffic flow into and out of the mine.

The purpose of the simulation study was to provide a check of the design parameters to identify any potential design bottlenecks or risk areas. The surface infrastructure was modelled to determine if the planned infrastructure is sufficient to support surface vehicle operations in terms of arrivals, movement and parking especially during peak periods. The base case was based on a two-way stop at the main road (N11) intersection.

The surface infrastructure simulation indicated that for the base case, inbound traffic is constrained during peak periods and results in vast queuing up to the intersection and onto the N11. This impacts mine and N11 traffic as well as personnel arrival times and vehicle travel times.

Outbound traffic flow is also constrained and results in vehicle queuing, especially for cars. The planned parking capacity for arriving delivery trucks at the main gate is sufficient. Furthermore, there are minor parking capacity concerns for certain delivery trucks of which the most important is concentrate deliveries. The weighbridge also experiences intermittent queuing.

It was found that processing time at the main gate is the biggest concern and the solution is to reduce the processing time at the main entrance.

18.12.8 Surface Fleet

Table 18.6 lists all the surface fleet equipment that has been allowed for.

Table 18.6 Surface Fleet Equipment

Type	Description	Quantity			Capacity	Area
		Phase 1	Phase 2	Total		
Cranes	250t Lieber All Terrain	0	1	1	250 t	6000
	120t Lieber All Terrain	0	1	1	120 t	6000
	50t Lieber All Terrain	0	1	1	50 t	6000
	30t Lieber All Terrain	0	1	1	30 t	6000
	90t Lieber Rough Terrain	1	0	1	90 t	6000
Telehandler	Telehandler MHT - X 10180	0	2	2	18 t	2000, 3000
	Telehandler MRT - X 3255 Privilege	0	2	2	5,500 kg	3000, 6000
Site Forklift	M-X 70-2 Rough Terrain Forklift	1	3	4	7,000 kg	2000, 3200, 6000
	MH 20 Rough Terrain Forklift	0	1	1	2,000 kg	6000
Reagent Forklift	ME 430 AC Electric Forklift	1	1	2	3,000 kg	3100, 3200
Skid Steer Loader	Gehl 4240E Skid Steer Loader	1	3	4	2,100 kg	2000, 3000
Waste Skip Tractor and Trailer	Massey Ferguson 440Se Xtra 61kW, two-wheel drive tractor with airbrake compressor and sun canopy	0	1	1		2000, 3000, 6000
	AIM Power X 6 ton Chassis Trailer with airbrakes and Road Ordinance electrics	0	1	1		2000, 3000, 6000
	AIM Power X Screwback Container with tailgate	0	1	1		2000, 3000, 6000
Front End Loader	CAT Small Wheel Loader 938K	1	0	1		3200
	CAT Large Wheel Loader 980K	3	1	4		3200, 6000
	CAT Medium Wheel Loader 950H	0	2	2		3000
Articulated Trucks	CAT Three Axle Articulated Truck 745C	4	3	7	41 t	3200, 6000
Excavators	CAT Medium Excavator 330D2L	0	1	1	30,000 kg	6000
	CAT Large Excavator 340D2L	0	1	1	40,000 kg	6000

Type	Description	Quantity			Capacity	Area
		Phase 1	Phase 2	Total		
	CAT Large Excavator 374F L (2017)	0	1	1	70,000 kg	6000
Dozers	CAT Medium Dozer D7R	1	2	3		6000
Truck with High up	18 Ton Flatbed Truck with High up	1	1	2		6000
Motor Grader	CAT K Series Motor Grader 140K	1	0	1		3200
	CAT M Series Motor Grader 120M	0	1	1		6000
Backhoe Loader	CAT Backhoe Loader 416F2	0	1	1		6000
Roller	CAT Vibratory Soil Compactor CS64B	0	1	1		6000
Water Bowser	15,000 L Water Bowser	1	0	1		6000
Diesel Bowser	8,000 L Diesel Bowser	1	0	1		6000
Ambulance	ECW Ambulance Two Bed	1	0	1		6000
	ECW Ambulance Four bed	0	1	1		6000
Fire Truck	Medium-Size Fire Truck 4 x 4	1	0	1	3,000 L	6000
	Large Fire Truck 6 x 4	0	1	1	8,000 L	6000
Rescue Vehicle	RIV Land Cruiser	1	0	1		6000
Rescue Pumper	Rescue Pumper	0	1	1		6000
Rescue Platform	TL Hydraulic Platform	0	1	1		6000
Water Tanker	Water tanker major pumper	0	1	1		6000

18.13 Security and Access Control

18.13.1 Control

18.13.1.1 Access Control

The access control system is a software-based system with a database which will be accessed on the network on a fibre link from the server to the field switches via the main controllers and to the sub-controller. Every door will need a controller with entry and exit of outdoor and/or indoor biometric readers in buildings, boom gates and turnstile.

The system works as a dual tag system, meaning both fingerprints and a card reader will be required to grant a personnel entry or exit.

The access report database will be stored on the server with Time and Attendance and its software integrated in the system. All the personnel who will be accessing the mine will be enrolled on the fingerprint system and be issued with a card.

The CCTV and access control system will be integrated into all main entry and exit points, allowing for the ability to view the video recordings of a personnel when badging on the biometric card reader's access control system.

18.13.1.2 CCTV

The closed-circuit television (CCTV) system is an Internet Protocol (IP) based network system with a 10 GB backbone design on a fibre link from the server to the field switches. This includes the server and storage for every 50 cameras with 30-days live and 30-days archive recordings respectively (total of 60-days recordings). All switches will be power over Ethernet to cater for all the IP cameras.

There will be outdoor static cameras for all the outdoor areas, including workshops and for overview coverage, while there will be indoor fixed cameras for all indoor offices and buildings. High-speed dome cameras will be used mainly for parameters and big overview coverage.

The CCTV and access control system will be integrated at all main entry and exit points with the ability to view the video recordings of personnel when badging on the biometric card reader's access control system.

18.13.1.3 Security Alarm

The security alarm system design is easy to service and maintain. This includes the monitoring of the fire escape doors with door contacts and passives (PIRs) to be located in a logical manner in the buildings. The system will include and integrate with the following:

- Radio and GPS link, and
- Panic button (mobile remote panic type).

The power pack will be connected to the uninterruptible power supply (UPS). All components will be wired to expander modules for easy installation and maintenance.

18.13.2 Fencing and Access Control Buildings

Fencing

The mining lease area is fenced off along the outside of the perimeter berm, with a 4 m high ClearVu fence and a 2.4 m high ClearVu fence along the N11. In and around the main entrance area, a 2.4 m high 2D Securifor fence forms a barrier between the engagement centre, main entrance, and the visitor and bus parking area.

For the internal fence lines, a 2.0 m high Nylofor fence has been allowed for. All dams will be fenced to prevent unauthorised access and to comply with legislation. The Shaft Area has already been fenced off with 4.5 m high concrete slabs. The south-western discharge drain #3 servitude has already been fenced with a 2.0 m Nylofor fence.

The fencing will be installed in phases to support the Project's Platreef 2022 FS. An additional fence for the small case 770 ktpa concentrator plant has been allowed for as part of Phase 1.

Additionally, a low specification diamond mesh fence has been allowed to fence off the perimeter of the Phase 1 bulk material crushing and conveyors from Shaft 1 feeding the 770 ktpa concentrator plant. This fencing is mainly to restrict animals from entering this area.

For the tailings line and dam servitude a 2.4 m high Nylofor type of fence line has been allowed for in order to secure the servitude.

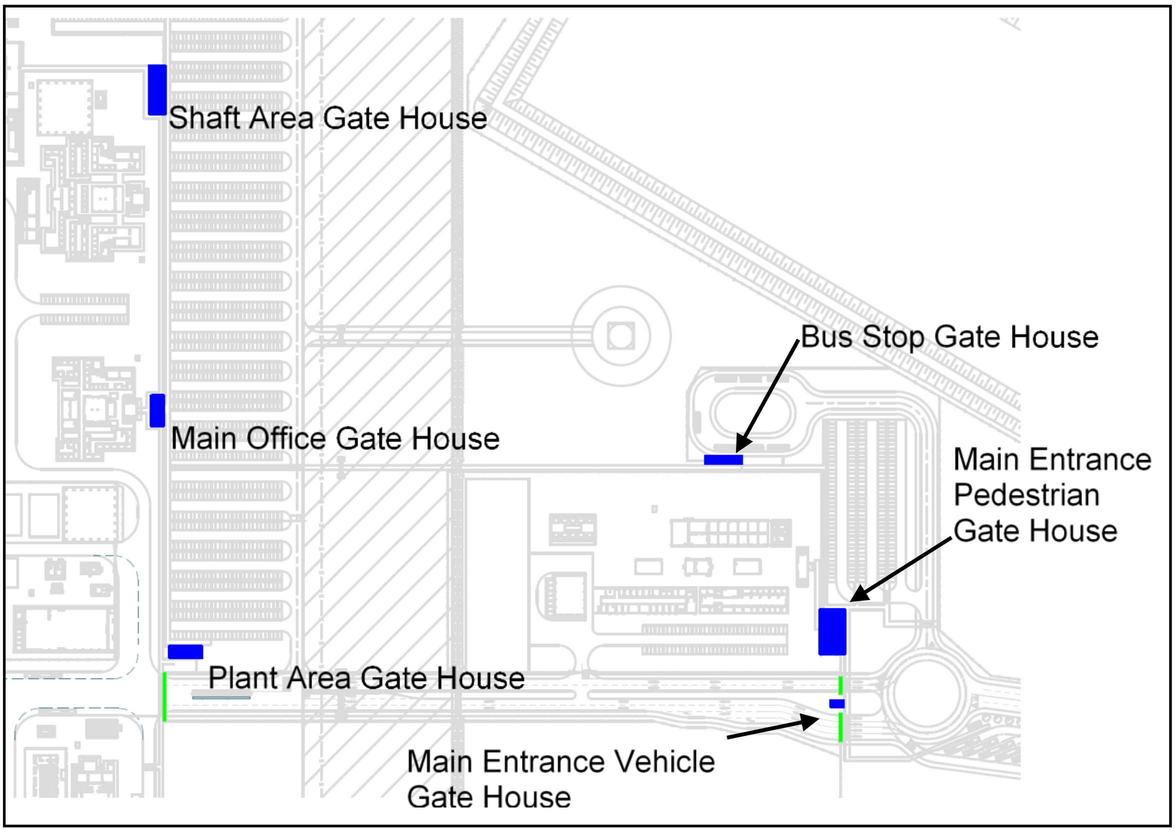
Access Control Points

Various access control buildings have been allowed within the mine area. Table 18.7 lists them all and Figure 18.28 indicates their location.

Table 18.7 Access Control Buildings

Area 2000			
WBS Number	Drawing Description	Building Description	Area/m²
257805	Shaft Area Gatehouse	Gate House with turnstiles controlling the entrance of pedestrians into the shaft area.	110
Area 3000			
328507	77 ktpa Plant Access Control	Pre-fabricated Gate House on a concrete slab complete with turnstiles controlling the entrance of pedestrians into the plant and store area as well as vehicles that want to enter the concentrator plant area. The weighbridge control room also forms part of this building	
341105	Plant Area Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the plant and store area as well as vehicles that want to enter the mine and store area. The weighbridge control room also forms part of this building.	80
Area 6000			
618105	Main Entrance Vehicle Gate House	Main vehicle access control at the main entrance to the mine controlling all vehicles that want to access the mine including deliveries and visitors.	30
618128	Main Entrance Pedestrian Gate House	Gate House with turnstiles controlling the entrance of visitors to the mine and employee access to the engagement centre.	360
618129	Bus Stop Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the mine area arriving with buses and taxis.	90
618130	Main Office Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the main office area.	50

Figure 18.28 Permanent Access Control Points



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18.14 Waste Management

18.14.1 Introduction

The Platreef Project will generate a number of general and hazardous waste streams that will be managed on site. As part of the proposed mining operation, the following waste-related facilities will be constructed to manage the various waste streams:

- Hazardous waste storage area for the safe and compliant temporary storage of various hazardous waste materials.
- General waste storage area for temporary storage of sorted material before disposal.
- Waste tyre storage area for the short- to medium-term storage of used tyres.
- Recycling / Material Recovery Facility (MRF) for sorting and separate storage of recyclable materials.
- Oil traps and oil storage tanks for oil interception and recovery, and temporary storage of used oil.
- Salvage yard scrap for temporary storage of scrap metal and re-usable mechanical parts.

- One stormwater pond for capture and storage of contaminated water generated on site.
- Sewage Treatment Plant to treat black water produced on site.
- Water filtration plant to treat the bulk make-up water supply from the local WWTW and / or water from the dirty stormwater pond. The filtered product water will be utilised for mining, concentrator plant and backfill plant make-up water requirements. Sludge / waste generated by the water filter plant will be disposed of at the dry stack tailing storage facility.
- Borehole water treatment plant consists of an ultrafiltration and reverse osmosis treatment process. The product water will predominately be used for the potable water requirements for the entire site. Waste / brine generated by this plant is considered to be within the tailing storage waste classification and will be utilised a dust suppression on the dry stack tailing storage facility.
- General Waste Disposal Facility (landfill), including leachate pond and sediment trap, for on-site disposal of general waste.

Within the central mining area, small intermediate solid waste transfer open storage yards are provided for general waste and salvage.

Most of the permanent waste facilities will only be constructed during Phase 2 of the project. During Phase 1 a smaller module of the sewage treatment plant, temporary polluted stormwater ponds, salvage yard and a portion of the Dry Stack TSF will be constructed. The rest of the general waste will be managed in line with current operations and existing facilities.

The Phase 1, 770 ktpa concentrator plant tailings will be filtered at the process plant, conveyed and stockpiled, from where it will be hauled to temporary dry stack tailing storage facility in the north-eastern corner of the mining area.

Associated with the flotation process is a tailings disposal sump, constructed during Phase 2 which permits mineral sludge to be transferred via the tailings delivery line to the Rietfontien TSF.

Certain of the above facilities require a Waste Management Licence (WML) in terms of the National Environmental Management Act, 2008 (Act 59 of 2008) (as amended).

The required ESIA processes were completed and the WML issued on the 13 March 2015, following approval of conceptual designs of the waste facilities by the DWS (based on a review of the designs and subsequent amendments made to the designs).

Since Ivanplats wish to use the site of the waste rock dump (WRD) as Dry Stack Tailings Storage Facility (TSF) the amendment of the approved WML and WUL is required. The ESIA process and public participation required for these amendments commenced in September 2021 and is proceeding in accordance with the National Environmental Management Waste Act (NEMWA), which regulates mine residue stockpiles and deposits.

18.14.2 Regulations and Principles Applied

The key waste legislation, principles and policies that were applied in the development of the Waste Management Plan and waste facility designs for the project are listed below:

- Constitution of the Republic of South Africa, 1996,
- National Environmental Management Act (NEM Act)
- Environmental and Social Impact Assessment (ESIA) Regulations,
- National Environmental Management: Waste Act,
- List of waste management activities that have, or are likely to have a detrimental effect on the environment (GN R.718 of 03 July 2009),
- Waste Classification and Management Regulations,
- National Norms and Standards for the Assessment of Waste for Landfill Disposal,
- National Norms and Standards for Disposal of Waste to Landfill,
- National Waste Information Regulations,
- Draft Health Care Risk Waste Regulations,
- National Water Act,
- Waste Tyre Regulations,
- Mineral and Petroleum Resources Development Act,
- Explosives Act, and
- Principles:
  - Integrated Waste Management Hierarchy
  - Extended Producer Responsibility
  - Duty of Care
  - Cradle to Grave

18.14.3 Description of Waste Facilities

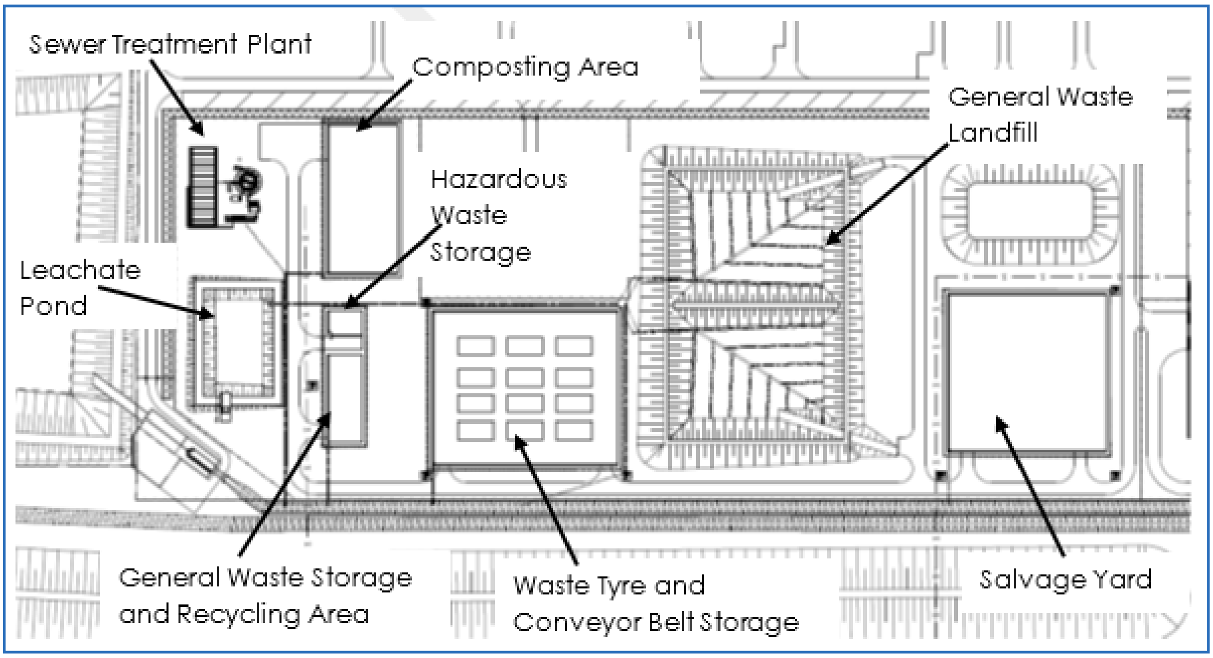
In the waste area allowance was made for the following waste facilities:

- General Waste Landfill
- Composting Plant
- General Waste Storage and Recycling Area
- Hazardous Waste Storage Area
- Salvage Yard
- Waste Tyre and Conveyor Belt Storage
- Oil Traps
- Sewage Treatment Plant

- Leachate Pond
- Waste Collection Bins

Figure 18.29 below indicates the position of the different type waste facilities within the waste area.

Figure 18.29 Waste Facilities Area



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18.14.3.1 General Waste Landfill

It has been established that Platreef will construct an on-site landfill with a designed lifetime of 36-years from the start of shaft sinking. The landfill model considers three rates of deposition:

- Bulk shaft rate of deposition (3-years) = 672 tpa (1.84 t/d)
- Construction rate of deposition (3-years) = 1,530 tpa (4.19 t/d)
- Operation phase rate of deposition (30-years) = 708 tpa (1.94 t/d)

The landfill will be dedicated to the disposal of general waste from the mine and mine-related activities only, with the removal of recyclable materials (tins, glass, plastic and paper) as well as organic material for composting before disposal of the residual waste.

The landfill site is expected to receive general waste, packaging waste and garden waste that are regarded as residual after source separation and further sorting at the materials recovery facility.

Due to low levels of organics in the landfill, odour and vermin are considered to be a minor issue. Cover material has been assumed to be provided in a 1:5 ratio, which may be excessive due to the reduced organic composition of disposed waste.

The design is of a set of two cells with a combined airspace of 80,000 m^3. The cells will be developed with base at 3 m below ground level and reaching a maximum height of 5 m above ground level.

18.14.3.2 Composting Plant

A composting plant is to be built in the vicinity of the landfill for the processing of organic wastes into a usable product for various landscaping functions around the mine. The composition of material from the general waste stream at the mine site which will enter the composting system is estimated using the following conservative figures:

- Potential 30% of domestic waste (rubbish) if organics are separated.
- Sewage sludge in operation phase after drying beds.
- 80% of wood and garden waste.
- Potential 80% of packaging.
- Sludge from an on-site potable water treatment plant (3–6% solids) if a provision for tankage and pumping is made at the composting facility.

The plant design will be based on a total capacity of 4,800 m^3 per year. Material will be supplied from the general waste holding yard or material recovery facility where separation, identification and quantities delivered to composting will be recorded.

18.14.3.3 General Waste Storage and Recycling Area

All of the general waste, including recyclables and organic material, that is disposed into the various bins on site, will be taken to a centralised storage and sorting area for further sorting and consolidation before the various types of materials are dispatched to their respective destinations.

The general waste storage area will have an adjoined recycling area also referred to as a Materials Recovery Facility (MRF) as this will be the source of all recyclable materials that are sent off site. The general waste storage and recycling area should comprise the following elements:

- Recyclable skips and cages for recyclable material accumulation by type.
- Area for separate disposal of packaging waste for subsequent separation to recyclable, composting and general landfill portions.
- Area for processing of wood and garden waste for use in composting via a shredder or for direct recycling where appropriate and, for disposal of residual to landfill.
- Area for receiving the fraction of domestic waste (rubbish) not source separated as recyclables, for processing of organics for use in composting, for direct recycling of further recyclables from the mixed waste where appropriate and for disposal of residual to landfill.

18.14.3.4 Hazardous Waste Storage Area

Hazardous waste that is generated at various locations around the mine will be taken to a centralised hazardous waste storage area. The hazardous waste storage area will be accommodated within the general waste storage area, but with separate security and entrance control as a stand-alone facility.

The hazardous waste yard will include appropriate covered storage area and will have a liner protected concrete pad with surface water containment and provision for final disposal of collected surface water. Each hazardous waste will be separately stored in the appropriate container, as discussed and provided by the hazardous waste removal contractor.

The hazardous waste storage area should be accessible to the hazardous waste collection contractors, and volumes of any hazardous waste material leaving the storage area should be recorded and put into the mine-wide waste information system.

18.14.3.5 Salvage Yard

A salvage yard is an area dedicated to the collection and sorting of re-usable parts, particularly metal parts. It is expected that a salvage yard will be alongside the scrap metal yard in the integrated waste management area (in the south-western corner of the site).

The salvage material should be identified by persons appropriately qualified to do so and should be stored in a way that does not hamper their re-usability, and protects the salvageable parts according to their value to the mine.

A detailed record of all salvageable parts in the salvage yard should be kept up to date and in the possession of the procurement department, in order to avoid ordering new parts, while the parts exist in the salvage yard. This list should also be available to the managers of the maintenance workshops.

Any used metal part which is beyond repair or may not be re-useable in any way on-site, should be sent to the designated area in the salvage yard for scrap storage before it is sold or auctioned off to scrap metal dealers. Platreef should ensure that scrap metal is not sold to unlicensed scrap dealers. A record of sale, recording price and volume, should be kept along with company registration documents of the purchasing scrap dealers.

18.14.3.6 Waste Tyre and Conveyor Belt Storage Area

Waste tyres and conveyor belts will be stockpiled until a downstream application is found and a waste transport contractor removes them. Tyres and conveyor belting will be stored in a separate facility with the required stacking and spacing arrangements to comply with the relevant legislation, particularly with respect to firebreaks.

Firefighting equipment will be required to be located in close proximity to the facility and the following site specifications should be met:

- Ground surface is cleared, levelled and compacted.
- Run off shall be diverted from upslope by a 1 m high diversion berm.

- A collector drain should be constructed on the boundaries to catch the runoff from the site. This should discharge through a silt trap to the contaminated water system.
- Appropriate demarcation and signage will be provided to indicate responsible person details and fire prohibition.
- The site is to be fenced with two gates for access.
- Internal firebreak access of 5 m between stockpiles and 6 m outside the stockpiles, i.e. outside the fence.
- Tyres to be neatly stacked, not exceeding 3 m height, 10 m width and row length to not exceed 20 m.

18.14.3.7 Oil Traps

Oil traps will be required to remove oil from surface water run-off, and thus a few of these facilities will be located on-site mainly at wash bays, workshops, maintenance areas and fuelling stations. While no manual labour is required to operate the oil traps, they will require cleaning at regular intervals, depending on the rate of oil capture. Most of the oil caught in the oil trap can be recycled and should be stored in a used oil storage tank for collection by a used oil collection contractor. An oily sludge is also produced and should be taken to a bioremediation plant off-site.

18.14.3.8 Leachate Pond

The leachate pond will receive leachate emanating from the landfill. The pond has been sized to have a volume of approximately 2,300 m^3. The pond will be excavated to a depth of 3 m below ground level, and side slopes will be 1:2.5. From the pond, leachate can be pumped and sprayed over the waste body if required, in order to reduce the level of leachate in the leachate pond.

18.14.3.9 Waste Collection Bins

It has been proposed that Platreef will install waste collection points at various localities around the site for the convenient separation and disposal of various waste materials by mine employees. These collection points should each consist of five 210l wheelie bins of different colours and with distinguishing labels for the separate disposal of the main groups of recyclable waste streams: paper and cardboard (together), tins and glass (together), plastic, and mixed domestic waste, while a fifth bin should contain hazardous wastes for safe disposal to hazardous landfill. Platreef should also consider having additional bins for the separate collection of certain hazardous wastes, such as batteries and oily rags. Extra bins should only be implemented in areas where significant volumes of the relevant waste streams are expected. Refer to Table 18.8 for details.

Table 18.8 Waste Bin Suggesting Colour Coding System

Waste Category	Colour of Containers
Hazardous waste	Red
Domestic	Black
Paper and cardboard	Blue
Tins and glass	Green
Plastic	Yellow

18.15 Construction Facilities

Where possible permanent infrastructure will be utilised to minimise temporary construction facilities. Temporary construction facilities and services has been allowed for during the construction phase of the project where permanent infrastructure has not been constructed or allowed for.

These facilities are a crucial part of construction management, as sites can be very demanding involving the co-ordination and movement of large quantities of materials, as well as high-value products, plant and people. Effectively and accurately laying out a site can help ensure that the construction works are undertaken safely and efficiently. Correct sizing and positioning of temporary facilities can help to reduce travel times, congestion and waiting times making the site a more effective workplace.

The following temporary utilities have been allowed for:

Construction Power

Construction power is to be provide from the current 8 MVA ESKOM supply. Local diesel generators will be provided by the contractors if required.

Provision has also been made to reticulate grid power to the (EPCM) construction team offices, the construction offices, the construction laydowns and the concentrator plant construction areas from a series of mini-substations. Remote areas, for example the tailings facility, will continue to be supplied by local diesel generators as required.

Construction Water

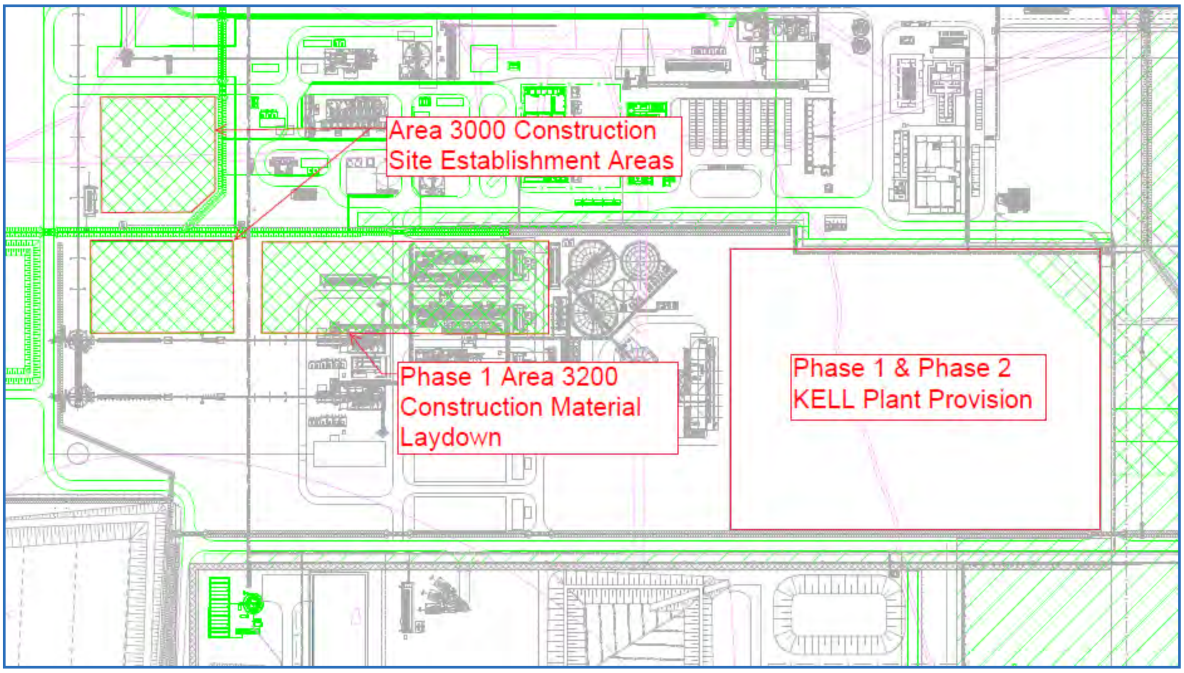
Construction water will be provided from the existing well field and distributed from the existing water tank on site. The water will be carted with water bowsers and discharged if required.

Construction Laydowns

Initially, a small allowance has been made for the preparation of construction site establishment and laydown areas. The preparation consists of site clearing, stripping topsoil as well as rip and compact of in situ material. It should be noted, that operating conditions on these areas will be difficult during the rainy seasons and frequent maintenance and improvements will be required.

Preliminary areas to the west of the concentrator plant area have been identified as construction site establishment areas for the concentrator plants. An area on the Phase 2 plant terrace has been made for material laydown during Phase 1 construction. Refer to Figure 18.30 for a preliminary layout of the area 3000 construction site establishment and laydown areas.

Figure 18.30 Phase 1 Construction Laydown Area



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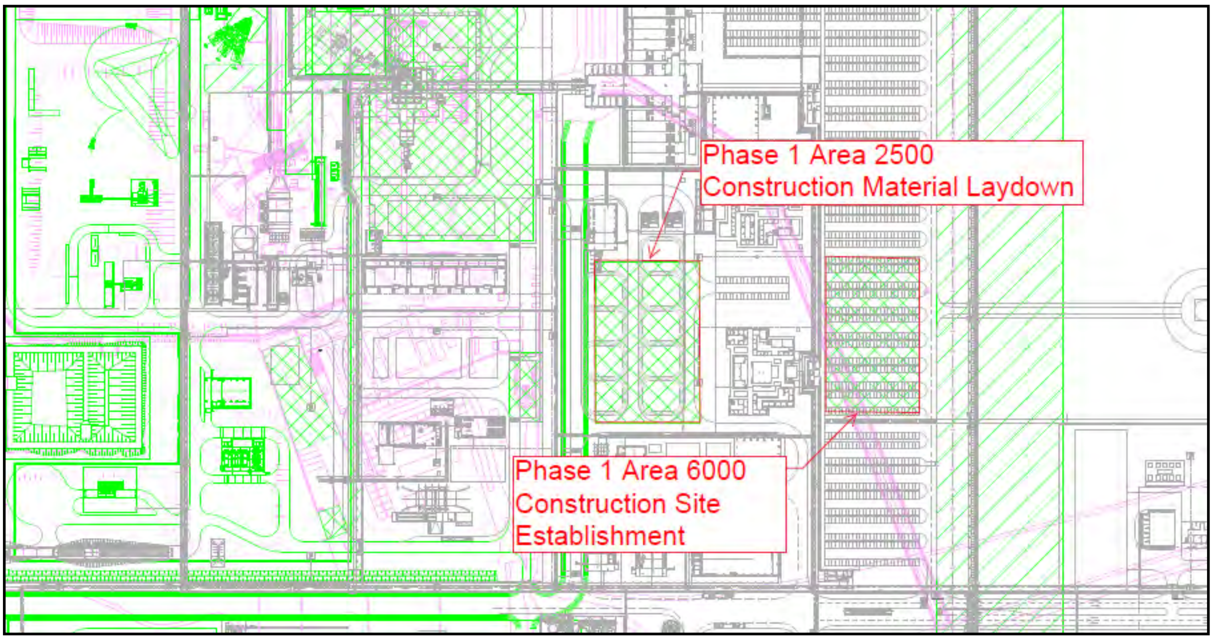
Refer to Figure 18.31 below for additional allowance that was made east of the existing Shaft Area concrete perimeter wall for Phase 1 construction material laydown and Area 6000 construction site establishment.

Figure 18.31 Phase 1 Material Laydown and Area 6000 Site Establishment



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Additional provision during Phase 2 construction was made for improvement and expansion of construction site establishment and laydown areas.

In the work areas, the terraces will be utilised where possible as hard stand for construction purposes, and in some areas, where required, smaller hard stands have been allowed for.

Construction Office

The Pre-fabricated parkhome office units have been allowed for Phase 1 EPCM construction offices. The temporary existing prefabricated shaft offices will also be utilised by the Owner's Team during Phase 1. During Phase 2, additional units will be established, if required, until such time permanent buildings have been constructed.

Contractors will be responsible for their own temporary site offices.

Construction Stores

During Phase 1, the construction team will utilise containerised storage solutions. Additionally, a small permanent store will be constructed for the Owner's Team shaft development operations.

The mining development contractor will utilise the existing shaft store, while all other construction contractors will be responsible for their own storage solutions.

Construction Communication

An independent IT network in the construction offices with Internet Data communications is catered for the EPCM project team.

Handheld mobile radios and two base stations are provided for site voice communications.

The area is covered by the mobile cellar networks.

Construction Vehicles

Construction vehicles have only been allowed for the EPCM Project team. Existing project vehicles will be utilised.

Construction SHEQ

Allowance were made for lighting detection equipment, breathalyser's, first aid equipment, permit and inspection books, construction office firefighting equipment, working at height rescue kit, PPE and environmental spill kits.

Paramedics and medical facilities have already been established on the site.

Allowance was made for standards signs required for a construction site.

Construction Ablution Facilities

The site has an existing establishment with sewerage being directed to a septic tank. During Phase 1, additional prefabricated ablution units have been allowed for the EPCM site establishment and laydown areas with septic tank systems until the permanent sewage treatment plant has been constructed. The temporary existing prefabricated ablution blocks, which are connected to the existing sewage treatment plant within the shaft area, will be utilised by the Owner's Team,

The development contractor, various construction contractors and installation contractors will be responsible for their own ablution facilities and sewage management systems. It will be advised that all contractors to have prefabricate ablution units with running water at their site establishment facilities and chemical toilets at remote working areas.

Construction Access and Security Facilities

Temporary container type turnstile units with card readers have been allowed for to assist with the access control of employees to the site. Vehicle access to site will be controlled by boom gates also linked to the card read system.

Construction Waste Facilities

Allowance was made for a construction salvage yard and waste skip laydown areas. A contractor will be appointed to clean and maintain the waste bin skips.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Platreef production is scheduled to come on stream at a time when the world will require additional PGM production to meet what many observers predict to be significant supply and demand deficits. Favourable positioning on the cost curve, base metal diversification and a natural South African Rand (ZAR) hedge should all conspire to make the concentrate attractive to South African toll smelters.

Ivanplats has prepared a number of marketing studies historically and maintains relationships with key smelters in the South African PGM space. Ivanplats recently secured offtake arrangements for Phase 1 concentrate and has a clearly defined development strategy to secure smelting and refining capacity in South Africa and has mapped a development path for placing Phase 2 concentrates and expects capacity to become available by the time that steady state production is achieved.

With the establishment of a number of smaller PGM mining firms, toll smelting and refining contracts and concentrate purchase agreements have become more prevalent in South Africa than in the past. The main PGM mining companies have some internal purchase contracts with their own mining / concentrating operations and external purchasing or toll contracts with independent or joint venture (JV) companies. Within the industry and along the value chain there are various possibilities for metal sales contracts: concentrates, furnace and converter mattes, Ni by-products, PGM residues or concentrates have all been sold or toll treated in the past.

PGM concentrate is sold within South Africa and into Europe under long-term contracts. The three major PGM producers have a full suite of process facilities to produce final PGM metal and hence tend to be purchasers rather than sellers of any PGM containing materials. Other PGM producers produce various intermediate products across the value chain ranging from flotation concentrate to high-grade PGE residue and nickel sulfate. The vast majority of these products are refined in South Africa, but some high-grade PGM residues are shipped overseas for final processing in Germany.

19.2 PGM Market Dynamics: Present and Forecast

PGM markets are influenced by a combination of fundamental supply and demand economics, global macro commodity drivers, commodity exchange futures, physical investment, over the counter (OTC) sales and purchase transactions, and movements of stockpiles across the value chain from mine to market. Platinum (Pt), palladium (Pd) and rhodium (Rh) are predominantly used in industrial applications, with Pt having a significant jewellery demand, and a higher price elastic component (approximately 35%).

Pt, Pd and Rh share unique catalytic and chemical properties which make them essential in the treatment of automotive emissions from both diesel and gasoline-powered vehicles. Sales volumes and strictly legislated global emission standards continue to underpin current and forecast demand. Catalytic converters on automobiles and trucks convert toxic pollutants in exhaust gas from internal combustion engines to fewer toxic emissions. Pt is particularly effective in diesel-based vehicles, while Pd and Rh tend to be favoured in the three-way catalysts in gasoline-powered vehicles. Rh in particular is effective in the removal of nitrogen oxide (NOx) from the exhaust gas. The PGM industry will continue to benefit from these unique technologies as the global automobile pool expands through the next decade, and more stringent emissions legislation is introduced globally. PGM demand is expected to benefit from the continued roll out of both hybrid and electric vehicles. The global car automobile pool constituents in the future, when Platreef reaches steady-state production, are likely to include fuel-cell electric, battery-powered electric, hybrid and the conventional internal combustion engine. Vehicle ownership is likely to increase significantly in the developing world where ownership at the moment significantly lags behind the developed world. LMC Automotive currently estimates that, in the US, ownership per 1,000 driving population is greater than 800 vehicles. In China, the largest vehicle market, ownership levels are currently less than 100, thus, growth prospects are expected to be strong.

Coupled with this vehicle volume effect on Pt, Pd and Rh demand, it is recognised that the ever increasingly tighter emissions legislation on conventional vehicles directly correlate to increased PGM usage from higher thresholds in the developed world and a steady roll out of legislation to the developing world where current standards can be significantly lower. Recent trends in the auto sector resulting from tightening emission regulations has driven demand for gasoline-powered vehicles over diesel-based vehicles, favouring palladium and rhodium use over platinum in catalytic converters. With more stringent emissions standards particularly in Europe and China following the announcement of the Euro 6 and China 6 emissions legislation, the increased demand for both gasoline-powered vehicles and the palladium loading in these vehicles has driven automotive demand for palladium to an all-time high of 9.7 million ounces in 2019 based on data from Johnson Matthey. Similarly, higher loading of rhodium in automobiles to reduce NOx emissions for real driving emissions (RDE) tests has resulted in significant tightness for the smaller rhodium market. Tighter NOx emissions legislation and the resulting demand form automobile manufacturers has driven the rhodium price to all-time highs topping $25,000/oz, over twenty times levels seen in 2017.

The gradual shift in the auto sector from diesel-based vehicles to gasoline-powered vehicles combined with a decline in jewellery demand has left the platinum market over-supplied, which has led to depressed prices. However, advancements in hydrogen fuel cell technology are driving a growing interest in fuel cell electric vehicles to play a role in the decarbonisation of the auto sector. Both the anode and the cathode in a fuel cell are platinum-based catalysts bringing the average platinum requirement for each fuel cell electric vehicle to approximately 10–20 g of platinum. With the typical diesel engine only requiring 5–6 g of platinum, the adoption of fuel cell electric vehicles has the potential to not only replace the loss in diesel demand but drive platinum demand growth from the auto sector beyond previous levels. These observations and comments are aimed at the passenger vehicle markets. There is currently a concerted effort to bring the global heavy-duty diesel market (trucks) into a similar legislative framework which will result in a further increase in Pt, Pd and Rh demand.

In addition to the auto sector, PGMs have many industry-specific applications. Many of these applications will have to expand to accommodate environmental and technological issues in the future. Demand for platinum in particular is closely tied to jewellery demand, which accounts for approximately one third of total demand, with China being the dominant market and India the successful growth story. Promotional campaigns in these markets are funded by the producers. Recycling of jewellery in China can have a short-term effect on demand but continued rolling out of the campaigns by the industry should allow for steady, if modest, growth going forward.

On the supply side, South Africa is pivotal to PGM production. The majority of mined Pt production comes from South Africa, while for Pd, the country's production is close to that of Russia, which is dominated by the production of Norilsk (as a by-product of Ni). South Africa's traditional PGM mines in the Western Bushveld are challenged by high costs, diminishing grades and labour issues around mining methods. However, the declining output from the deep mines in the Western Bushveld due to the closure of high-cost shafts is expected to be offset by incremental supply as a result of sustained high palladium and rhodium prices, influencing the long-term strategic decisions made by producers. However, low global stocks of palladium, in particular those stocks in Russia, and rhodium are expected to provide medium to long-term price support for the two metals. Without sustained high prices incentivising the construction of new mines and the expansion of existing mines, it is envisaged that South African production will remain high cost and decline over the next five to ten years. It is forecast that there will be a need for replacement ounces from new sources, including the Northern Bushveld where Platreef is located.

19.3 Trade in Flotation Concentrates

The sale of PGM flotation concentrates from various Merensky and UG2 mining operations in Southern Africa has increased over the past decades as producers seek to optimise existing metallurgical facilities and offer surplus capacity to smaller mines developed outside of the traditional larger mining lease areas. These agreements can be quite variable in net payment for metal contained but will have some of the following elements: smelter charges; refining charges; metal accountabilities; pipelines; penalties; delivery terms; assay charges; metal accounting provisions; dispute resolution mechanisms; length of contract periods; renewal conditions; other considerations. Most significant of these in economic terms are the metal accountabilities, any treatment charges, penalties and the pipelines or delays between concentrate delivery and payment. This pipeline for some metals is as long as nine months (i.e. rhodium), such that these terms can have an impact on the sellers' cash flow although platinum, palladium and the base metals are generally paid within eight to ten weeks. Flotation concentrates, because of their bulk and the South African government's focus on beneficiation, do not find a market offshore and are all sold locally.

19.4 Trade in Intermediate Products

The sale of PGM-rich intermediate products from further down the value chain, such as furnace mattes or converter mattes, is less developed as a market and tends to be shorter term in nature, often to debottleneck capacity. There is, however, significant competitive interest in these products should they become available due to the mix of both base metal and PGM components in the feed.

Some companies have long-term contracts for the sale of their nickel sulfates in place; on some occasions, there have been sales of nickel-copper matte after magnetic removal of the PGM-containing alloys, and there is a long-standing contract for very low PGM nickel concentrates in Botswana. Nickel mattes from Botswana are sent to Europe. PGM terms from offshore buyers are normally worse than those offered by local smelters, but base metal terms are usually better.

19.5 Trade in High-Grade PGE Concentrates

The high-grade PGE concentrates (>40% PGE) that are produced either as a base metal refining residue or as a residue from the leached magnetic concentrate would have a ready market with any one of the major PGE refiners and fabricators globally. The amphoteric elements (Sb, Bi, Te, Se, As, etc.) are seen as deleterious, and penalties are a possibility for unusual amounts of these contaminants. There is currently only one long-term contract in place for treatment of base metal refining residues with an overseas refinery, although international buyers do not routinely purchase such concentrates due to adequate available local PGE refining capacities.

19.6 Treatment Capacity in Southern Africa

It is sensible to examine capacity at the following key points in the process sequence: furnace capacity, converter and acid plant capacity, base metal refining capacity and precious metal refining capacity.

The Platreef concentrates will place a high demand on the smelting facilities because of their relatively high sulfur and iron content, and the furnace, converting and acid plant capacities may become a constraint in individual smelters.

The conventional furnace operations have limited capacity for UG2 concentrates as the level of chromite is a concern in the traditional electric furnaces. Hence high penalties are applied to chromite levels of more than 1% whilst generally concentrates with chromite levels in excess of 3% are rejected as feeds to conventional smelters. With the concentrates expected from Platreef, Ivanplats does not expect this to be a concern, and Platreef concentrate could potentially be used to dilute high chromite containing concentrates.

Further afield in Southern Africa, there is some idle capacity in Zimbabwe that has been unused since 2009 and which would require capital investment. In Botswana, it is unlikely that Platreef concentrate would be treatable in the flash furnace without significant blending to reduce the MgO levels. Metal recoveries, however, would be suboptimal for PGEs.

Base metal refining capacity requires further comment. Total nickel refining capacity in South Africa has been estimated to be 52 ktpa Ni in 2019. As Platreef ramps up from commissioning to steady state, nickel refining capacity is likely to be stretched, unless there are significant shut-down before then.

The availability of nickel refining capacity could increase if the PGE producers in Zimbabwe build a smelter, base metal refinery and precious metal refinery in Zimbabwe. The Zimbabwean government continues to discuss and consult on a proposal to heavily tax or ban the export of raw materials containing PGEs. PGE concentrates and mattes produced in Zimbabwe are currently processed in South Africa. Should this come into effect, then additional base metal refining capacity could become available in South Africa.

Investing in a dedicated furnace, converter and acid-plant facility at Mokopane to produce a converter matte opens up various other possibilities for Platreef, as converter mattes would have a ready market. Slow-cooling of the mattes could be considered, such that the PGEs are refined locally but Ni-Cu mattes are sent offshore. This would give Platreef competitive base metal terms for the mattes while allowing the high-grade PGE concentrates to be marketed locally. Use of a local smelter provides sufficient concentrate capacity for the long-term requirements, such that only base metal refining capacity is needed. This could be built locally in partnership or as a Platreef resources facility.

19.7 Treatment Contract and Cost Structures

There are many ways of structuring concentrate purchase or tolling contracts, and there can be many combinations of commercial terms providing the negotiated return between buyer and seller. These will also vary depending on the contract duration. The largest cost driver in the smelting complex is the tonnage of concentrate treated, and there can typically be a charge per dry metric tonne of concentrate to cover drying, smelting, converting and acid production costs. Longer-term deals will see these costs escalated. There are a range of terms offered in the market depending on several factors; type of concentrate, available capacity, opportunity costs associated with filling available capacity and the prevailing toll treatment market dynamics. Payables for PGEs can range as high as 85% whilst base metals payables would range from 75%.

Metal losses in the smelting and refining complex as well as base metal and PGE refining costs can be absorbed in the value of the metal retained by the toll treatment facility; this can be expressed as a metal gain representing the difference between the recoverable metal values and the accountable percentage of that recoverable value credited back to the concentrate seller. Alternatively, the toll treater can offer percentage returns closer to actual metallurgical recoveries but accompanied by higher refining charges. In reality, a combination of both systems is utilised, thus ensuring that neither party is overly exposed to wide market price variations over the course of a long-term contract.

Base metal refining costs are significant when separated from the metal recovery offered, and cost drivers in this operation are essentially the tonnes of nickel and copper to be refined, although the cost of removing sulfur is significant. Often cobalt is not a payable metal locally but overseas producers will pay some small amount for cobalt content in mattes and charge a cobalt refining fee. There is further expense in base metal refining through the slow-cooling route for matte treatment which gives rise to a magnetic separation plant and a separate leach circuit for the magnetic fraction, all operated under high security. However, this route can offer significant base metal refining and capacity flexibilities for both buyer and seller.

19.8 Metal Recoveries

Actual metal recoveries by the major PGE company process divisions (smelting, base metal refining and refining) tend to be quite high as is to be expected in light of the value of the metals concerned.

There are losses to smelter dust and slag, base metal refinery products and effluents, and to precious metal refinery effluents. The major losses are within the smelting operations and largely to furnace slags, as the bulk of all other refining losses are precipitated, collected and eventually recycled to the smelters.

Typical payable metal percentages to the concentrate suppliers take account of the downstream recoveries, the costs of refining the base and precious metals and the cost of capital to provide for smelting and refining capacity. Most often a fixed percentage of metal value (the metal accountability) is offered; these payable metals are nickel, copper, platinum, palladium, gold and rhodium. Payment is usually not made for ruthenium, iridium or osmium, although these metals are generally recovered and sold. Cobalt is also recovered and sold by the majors but is seen as a potential nickel contaminant rather than a profitable metal in its own right, thus payment for this metal is not often offered.

19.9 Payment Pipelines

The process required to produce pure metals from both refining operations takes significant time, and thus a large inventory of metal is held within the process. Metals that enter the smelting operations only appear as refined metal for sale some months later. Each metal flows through the process circuits with a different time distribution, and most refiners simply apply a single fixed period to each of the metal values to cover the cost of holding each metal in process. Various residue streams that are recycled to the smelting operations or within the refining operations add significantly to the 'pipeline' effect and impact on the operations cash flow and hence the business returns from a tolling contract. Pricing of accountable metals therefore reflects these time distributions, and the seller of concentrate has the option to use the terminal markets to hedge the final price if required. Advance payments reflecting a percentage of the accountable metal value less related treatment and refining charges can be negotiated.

19.10 Penalties

Penalties can be levied against the seller of concentrates for high moistures, low PGE grades and high chromite levels. Within the smelting and refining circuits, elements such as Fe, As, Bi, Sb, Se, Te, Pb, Zn, and SiO_2 can be problematic, such that buyers may levy penalties for some of these elements. There can be strict absolute acceptable limits of these elements, above which material may be rejected.

19.11 Terminal Pure Metal Sale Agreements

The final refiners of the base metals and PGEs use the established markets as pricing references. Base metals are priced on the London Metal Exchange (LME) with discounts or premiums applied depending on quality or end use application. Poorer quality metal is discounted while high-grade nickel could attract a premium in the battery, magnet and electroplating markets. Non-spot platinum and palladium are priced based on the London Fix mechanism which is quoted twice daily. Long-term contracts would typically use a monthly average quotation. There is a fluid discount / premium for metal sponge which varies according to, and is indicative of, real industrial demand. Rhodium, ruthenium and iridium are extremely illiquid and are usually priced based on a fabricator reference price or other New York dealer quotations. The concentrate buyer may or may not pass any of these price adjustments to the seller, and they are always subject to negotiation.

19.12 Platreef 2022 FS Smelter Terms and Price Assumptions

The prices in the economic analysis for the Platreef 2022 FS are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. The economic analysis uses price assumptions of $1,100/oz Pt, $1,450/oz Pd, $1,600/oz Au, $5,000/oz Rh, $8.00/lb Ni, and $3.50/lb Cu. For the Platreef 2022 FS the average transport distance for the concentrate has been assumed to be a distance of 277 km.

19.13 Conclusions and Development Plan

The Platreef 2022 FS will produce significantly larger volumes of concentrate once steady-stage production for Phase 2 is achieved in 2028 and 2030, with ramp up commencing 2028.

The studies by Ivanplats indicate that there will ultimately be sufficient smelting capacity in South Africa, but the degree to which smelters can commit capacity and the timing of commercial discussions will be determined by their respective captive mine schedules and differing views on the opportunity costs associated with making capacity available for Platreef and the significant lead time to steady state production.

In general, firm commitments from third party toll smelters will not be given until closer to first production, as it is not industry practice to allocate capacities with both these production lead times and the status of the current development work. Notwithstanding, discussions are underway with South African smelters and the possibility is recognised that concentrate offtake may be placed with more than one customer depending on interest, capacities and commercial terms.

The Ivanplats strategy is to seek firmer indications of interest for Phase 2 concentrate production and to continue the current technical discussions with a view to finalising offtake agreements.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Impact Assessment and Environmental Management Plan

In 2013, Ivanplats undertook the Environmental and Social Impact Assessment (ESIA) and Environmental Management Programme (EMP) for the Platreef Mine in support of a Mining Right, Environmental Authorisation (EA) and Waste Management Licence (WML) application. Since the approval of these applications, Digby Wells has provided ongoing environmental advisory support on the Platreef Project and undertaken further regulatory applications, including an ESIA/EMP Addendum (2016) in support of proposed amendments to the approved EA.

The key environmental and social licences and permits submitted for the Platreef Project are:

- Mining Right
- Environmental Authorisation
- Integrated Waste Management Licence
- Water Use Licence
- Heritage Permit

The possible future applications that Ivanplats may need to undertake based on the nature of the Platreef Project and/or amendments of approved activities include the following:

- Further amendments to the infrastructure plan, such as the inclusion the Masodi WWTW pipeline, the re-alignment of the TSF pipeline route and the change of the upstream TSF to a dry stacking facility, may require an amendment application process to be undertaken on the EA. Additional specialist studies may be required such as air quality studies, to inform the potential impacts and mitigation measures of the dry stacking of tailings.
- Ivanplats is considering options regarding the construction and operation of a Solar Photovoltaic Plant. Power generation that exceeds 10 MW requires environmental approval and triggers a Basic Assessment Process. Further to this, backfilling of residue material into the mine voids is being proposed. This operation does require a full ESIA process which will need to be granted prior to commencing with the activity of backfilling. This will require additional specialist studies as well as associated risk assessments and designs.

Various Public Participation Processes (PPPs) have been undertaken for the Platreef Project from the initial ESIA process and the subsequent EA amendment processes. Comments and issues raised by stakeholders were incorporated in a Comments and Response Report (CRR) as part of all ESIA processes undertaken. The key issues and concerns which were raised during the various PPP included:

- Impact of the Platreef Project on both ground and surface water (reduction in water quality and quantity).
- The increase in dust due to mining activities such as hauling on dirt roads and dust from tailings storage facilities.

- Potential damage to houses and infrastructure of surrounding communities as a result of blasting.
- Mistrust in Ivanplats management.
- Unmet expectations regarding benefits from the mine to the community.
- Surface lease agreements and negotiations.
- Enquiries as to how the mine will benefit people and communities.
- Employment of unskilled labour, disabled, women and local persons as first priority.
- Skills training and requirements for employment.
- The absence of government representation and traditional leadership at meetings.
- Following protocol before Public Meetings.
- Additional meetings for stakeholders who live in town and on farms.

The key environmental and social sensitivities that have been identified for the Platreef Project are:

- Surface Water
- Groundwater
- Wetlands
- Cultural Heritage
- Communities
- Noise
- Visual
- Dust

The Platreef Project will contribute to the local economy through both direct and indirect employment opportunities and will result in a substantial injection of cash into the local economy of the Mogalakwena local municipal area. In addition, there will be an increase in opportunities for local suppliers of goods and services to the operation. In general, the socio-economic conditions in the area will be uplifted through better infrastructure, Local Economic Development (LED) projects, Enterprise Development (ED), Broad Based Black Economic Empowerment (B-BBEE) ownership and projects and other company Corporate Social Responsibility (CSR) initiatives.

The development of entrepreneurs is one of the most effective ways of stimulating economic growth, transformation and the creation of jobs in the communities. Ivanplats' Economic and Enterprise Development function was established to ensure focused and integrated delivery of programmes aimed at contributing to the socio-economic development of the communities and the small, medium and micro enterprise (SMME) sector.

Development will focus on sustainability and job creation. Enterprise and supplier development aims to nurture, grow and sustain SMMEs by providing technical and business development support, through mentoring and coaching. In addition, loan funding will be provided to SMME suppliers through the Ivanplats Lefa Trust ("Lefa Trust"). Economic inclusion and development will be a joint effort between Ivanplats, contracting companies, preferred suppliers and government agencies.

The Local Economic Development (LED) projects in the Social and Labour Plan (SLP), as well as some additional projects, aimed in the first five years to construct infrastructure at strategic points in the host villages and provide appropriate support and training in an effort to make these projects sustainable. The infrastructure addressed urgent issues such as sanitation, a need for educational facilities, a need for pre-school facilities and access to a variety of services including information services, social services, financial services, training and entrepreneurial development. The second five-year plan changed its focus somewhat to include education infrastructure, access to water in the communities, support of health facilities and critical municipal infrastructure projects.

The potential impacts associated with the Platreef Project, including their pre-mitigation and post-mitigation significance, as well as mitigation management measures were identified. A monitoring programme has been developed to monitor various environmental aspects associated with the Platreef Project. The main potential impacts associated with the Platreef Project include, but are not limited to:

- Increased sediment and salts into drainage channels and streams.
- Increased fugitive dust generation.
- Loss of flora and fauna Species of Special Concern (SSC).
- Soil erosion and soil compaction.
- Increased surface water runoff resulting in decreased infiltration which will affect downstream users.
- Decreased surface water quality.
- Dewatering in upper aquifer resulting in negative groundwater quantity impacts.
- Groundwater quality impacts as a result of seepage from TSF, waste rock dumps, stockpile areas and hydrocarbon spills.
- Construction activities causing potential disturbances in wetlands will result in the loss of ecological services in these areas.
- Negative visual impacts due to site clearance and construction of noticeable infrastructure.
- Physical changes to burial grounds and graves due to site clearing.
- Noise impacts emanating from machinery and vehicles.

The findings of the ESIA and subsequent assessments undertaken have shown that the Platreef Project may result in certain negative impacts to the environment; however, adequate mitigation measures have been included into the EMP Report to reduce the significance of all the identified negative impacts. Most negative impacts (minor and moderate) can be reduced through the implementation of mitigation and management measures.

The main potential social impacts associated with the Platreef 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. Further to this, there are social risks due to the social environment under which the Platreef 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. The various stakeholder engagement processes revealed a reoccurrence of issues raised by stakeholders regarding the Platreef Project. Stakeholder engagement is an ongoing process, and a grievance mechanism has been developed to manage stakeholder concerns.

Continuous monitoring according to the EMP will be undertaken throughout the Life of Mine (LOM) to ensure correct implementation of the mitigation measures. Furthermore, internal and external audits of compliance to the EA, WML and WUL conditions will be undertaken in accordance with the authorisations and submitted to the relevant authorities.

20.2 Water Use Licence (WUL)

20.2.1 General Authorisation, Integrated Water Use Licence Application

An assessment of the General Authorisation (GA) from four properties (1,077 ha) on the Uitloop 3 KS farm was prepared and submitted to the DWS Limpopo Region (21 October 2013) and an acknowledgement of receipt was obtained. A request from DWS for Ivanplats to submit copies of agreements for the taking of water from two privately-owned properties on the Uitloop 3 KS farm was subsequently received. Agreements were drafted for each of the two landowners. These were submitted to Ivanplats for signature by the landowners and returned. Subsequently the signed agreements have been submitted to DWS and the registration of the water use on the WRMS data base finalised.

The IWULA and IWWMP for bulk sampling were submitted to DWS, Limpopo Office on 6 November 2013. Receipts were obtained and copies provided to Ivanplats.

The Bulk Sampling Shaft (BSS) IWUL was signed on the 9 March 2017, IWUL Licence Number: 01/A61F/AGJ/5021.

20.2.2 Integrated Water Use Licence (IWUL) for Main Mine

An engineering design review meeting took place with Mr Kelvin Legge and his team from the National DWS Department. All the engineering designs including those for the Tailings Storage Facility (TSF) were reviewed. The Platreef 2022 FS design, like the 2017 FS, included all the requirements of the regulator. The DWS issued a letter of acceptance of the TSF design, as well as the other water and waste management facilities on the site.

Designs currently in progress, including the dry stacking facility design, will inform the required amendments to the current Water Use Licence. The designs have been submitted to the DWS for review and comment. The existing Ivanplats Water Use Licence was approved by DWS on 22 January 2019. An administrative amendment of the licence was done and approved by DWS on 2 September 2021.

Ivanplats Water Use License was amended and approved by the Department of Water and Sanitation on the 7 September 2021, License Number: 07/A61G/GCJAIBF/6975.

20.2.3 Integrated Waste Management Licence

Platreef was issued with a Waste Management Licence (Ref. No. 12/9/11/L1224/5) in terms of the National Environmental Waste Management Act on 13 March 2015. An amendment to the Waste Management Licence is currently under way in order to approve the PDP activities such as the dry stacking of tailings.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating costs have been divided into functional cost areas based on the project work breakdown structure (WBS) as follows:

- Area 1000 – Geology
- Area 2000 – Mining
- Area 3000 – Concentrator
- Area 6000 – Infrastructure, Utilities and Ancillaries
- Area 7000 – Site Costs
- Area 8000 – Owners Costs and General and Administration

DRA compiled the overall capital and operating cost estimates. The estimates were prepared by the following for the areas shown above:

- DRA Projects SA (Pty) Ltd
- OreWin Pty Ltd
- Murray and Roberts Cementation (Pty) Ltd
- SRK (Pty) Ltd
- Golder Associates Africa (Pty) Ltd
- Digby Wells Environmental (Pty) Ltd
- Paterson and Cooke (Pty) Ltd
- Ivanplats

A standard engineering design process was followed to develop designs and drawings which were used for purposes of the estimating process. A detailed project execution schedule was developed for the overall mine life cycle, to determine the respective phases of capital cost requirements, from the Project start date in 2022, first concentrate in 2024, ramp-up to full production steady state of 5.2 Mtpa in 2030 and finally mine closure requirements in 2053.

21.2 Discipline Specific Estimates

The method applied to the discipline specific estimates was:

- Earthworks quantities were measured from completed drawings and a BOQ, in standard SANS 1200 format was produced. Formal requests for quotations were issued to the marketplace.
- A significant portion of the earthworks estimate is the use of G4 materials for terracing. Preference was given to using material produced by the crushing of waste rock on site, rather than using commercially sourced materials, however a significant amount of commercial material is required for the first phase.

- The surface fleet associated with the waste rock dump management was estimated on an owner supply and operating model. Building Works - Building works quantities were measured from the completed drawings to estimate a rate per m^2 that was applied to smaller buildings and structures. Applicable internal equipment, i.e. furniture, water heating systems and lockers, was grouped into procurement packages. Quotations were received for these items, adjudicated and applied to the overall estimate.
- Civil Works – Concrete quantities were measured from the completed drawings.
- Structural Steel – Plate work and Mechanical Installation Quantities were measured from completed drawings, and mechanical equipment lists. Key mechanical equipment elevations and weights were also derived from the designs, and a BOQ was produced.
- Mechanical – Based on the Mechanical equipment list, mechanical equipment was grouped into procurement packages, and a request for quotations issued accordingly. Quotations were received, adjudicated and applied to develop the estimate. Larger mechanical equipment such as Winders, Mills, Filter Presses and Underground Mobile Fleet was enquired as formal tenders, with specific terms and conditions. Detailed adjudication was performed and pricing for estimating purposes was selected on this basis.
- Piping – Piping within the footprint of the 770 ktpa concentrator could be estimated by factorisation. Overland piping and the piping for the 4.4 Mtpa plant was estimated by applying market rates to an estimated BOQ. quantities were measured from the material take-offs and line lists. A formal request for quotation to obtain rates was issued to the marketplace.
- Electrical, Instrumentation and Control (EC&I) – The EC&I estimate was derived using a combination of database rates, factorisation and quotations received.

For each discipline, a formal request for quotations was issued to the marketplace complete with:

- Detailed scope of work
- Standard FIDIC terms and Conditions
- BOQ
- High Level Schedule

A rates adjudication was performed, inclusive of current execution project rates as a benchmark. Suitable rates were selected and applied to the final quantities in developing the overall estimate.

21.3 Labour

Ivanplats has put in place a recruitment and selection process to be fair and transparent and to ensure that South Africans will be employed as part of the National Development Plan and local potential employees trained through skills transfer from expatriate labour. Ivanplats will deploy compensation and benefits that are of international standard and compensate employees within benchmarked labour rates with additional appropriate and transparent target-driven incentive schemes.

The community skills audit has identified that there is a shortage of mining skills required for the operation in the labour sending areas. This requires Ivanplats to set the objective to develop and reskill community members to ensure that the required skills will be available. This process will occur through skills transfer and by implementing several other development plans. The mine will be developed and mined by contractor model in its initial stages and be taken over by the owner at steady state in phases.

The mine will employ in total 2,000 employees at steady state with an emphasis on high technical skill levels. To train and maintain the workforce, integrated Human Capital Development Programmes will be set focussing specifically on the hard-to-fill positions but not neglecting others. Training will be conducted in a Mine Qualification Authority accredited centre that will have skills programmes in relation to the mine's operation. This will be done by ensuring that each job profile is well developed and that each employee will have a development plan on how he or she can progress through the relevant career path. The training will be conducted in an environment where a student can comply with the lead times but still achieve the correct level of competency by means of the appropriate mix of theoretical and practical training. This will be realised through e-learning, simulation and virtual training.

Employee relations will be a key factor in making a success of this mega mine. Organised labour will be managed inside set and agreed-upon recognition agreements and in an environment of mutual respect. By employing a well-defined joint communication strategy, the associated threats of an uninformed workforce can be addressed and if a dispute arises it can be mitigated through the set strategy and guidelines. The community as stakeholder will be managed by a well-developed engagement strategy supported by a fully staffed department. At the end of the mine life the project will downscale and close. It has already planned and made financial provision for this event in the SLP.

21.3.1 Salary and Wage Structure

The structure of salaries will be based on the current Total Cost Package (TCP) principle in the Paterson D to E levels. There is currently no collective agreement with a registered trade union. The C and B Paterson levels exercise their rights to collective bargaining through an employee representative forum. During 2021, a three-year wage agreement was concluded between the company and the employee representative forum. Remuneration packages are taxed as per the South African Revenue Services (SARS) tables per individual income and statutory deductions, i.e., Unemployment Fund (UIF) and Pay as You Earn (PAYE). Employees will contribute to an additional funeral policy at minimal cost. Permanent employees will partake in the company provident scheme currently at 14% of the Total Cost Package. The provident deduction includes a free funeral policy. Permanent employees will be covered by the group Disability and Life Cover related to the specific mining risk; the rates will be deducted from the Total Cost Package and paid over by the company on behalf of each employee.

Paterson C, and B employees will have the following allowances included in their TCP:

- Housing allowance,
- Paterson C, a medical subsidy equal to a hospital plan for one person, Paterson B, medical insurance,

- Transport allowance (dependent on own or company transport), and where applicable,
- Shift allowance (when on shift cycle),
- Underground allowance (when working under the surface),
- Standby allowance and applicable call-out.

The Company improved on all statutory requirements as per the Basic Conditions of Employment Act on other benefits i.e., leave.

21.3.2 Shift Cycles

The Project will operate on a continuous operation model-in planning the most cost-effective and optimised shift cycles the company did a trade-off study of several shift cycles. The optimised cycle decided on was a four-team 11.5 hour four-day shift on four-night shift and four days off. There will always be two teams off. This cycle was chosen as it supports the fatigue management programme to ensure effective handover between shifts in the mining cycle.

21.3.3 Labour Plan

The Labour Plan for the mine has been designed for the mine steady state of 700 ktpa projected at 2024 and of 5.2 Mtpa in 2030. The construction and development phase of the mine for the initial production of 700 ktpa and later the plant will take place consecutively within the first three years up until steady state of 770 ktpa is reached and then the expansion phase up until steady state of 5.2 Mtpa will occur within the next five years. Labour will increase as the phases of construction and development progress until mining commences and progresses until the 5.2 Mtpa production target is reached. Employment of workers will take place in such a way that the right people are in the right place at the right time when the mine is operationally ready. In period from 2022–2024, all construction, development, mining, and processing will take place by a contractor model.

21.4 WBS Areas Cost Estimates

The costs included in WBS cost areas are described below. WBS costs 4000 and 5000 were not used in the study.

21.4.1 Area 1000 – Geology

The capital cost associated with future surface and underground geotechnical works, associated equipment and software, and applicable requirements with all geology work over LOM.

Operating costs include the following:

- Mining Geology
- Mine Technical Services
- Geotechnical Engineering (underground (UG))

- Laboratory, Sampling, and Assaying

21.4.2 Area 2000 – Mining

The capital costs associated with:

- Shaft 1:
  - Surface infrastructure facilities.
- Shaft 2:
  - Construction of headgear.
  - Sinking of shaft.
  - Equipping of shaft and associated infrastructure as required for production.
- Ventilation Raises:
  - Raise boring and supporting of raises.
  - Establishing a second egress on Vent Raise 1.
  - Installation of infrastructure to establish each of the three ventilation raises into up-cast ventilation shafts.
- Shaft Surface Infrastructure:
  - Establishment of all surface infrastructure, required for steady state mine operation which include but is not limited to:
    - Surface backfill plants.
    - Surface building and Earthworks.
    - Electrical reticulations.
    - Rock handling and associated stockpiling.
    - Air Cooling.
- Underground Mining:
  - Underground Development.
  - Establishment of all underground infrastructure required to achieve steady state mining operations, which include but is not limited to:
    - Underground backfill systems.
    - Underground Fleet (Primary and Secondary).
    - Settling and Dewatering.
    - Crushing.
    - Underground Workshops.
- Commission and spares.
- EPCM.

The mining operating costs include the following:

- Labour – All mining related labour, including technical services.
- Power – Bulk Eksom power supply, excluding generator power.
- Water – Bulk water supply applicable to Area 2000, obtained from the site wide water balance.
- Ore development – Ore drift development including permanent materials, mobile equipment running costs and consumables.
- Vertical development – Contractor raise boring and in-house drop raising.
- Stoping – Permanent materials, mobile equipment running cost and consumables, for Drift-and-Fill stoping and longhole stoping.
- Consumables – maintenance consumables on engineering infrastructure.
- Construction – ongoing construction of underground engineering infrastructure, such as pumps, ventilation controls, escape routes, tips, chutes and refuge bays.
- Backfill – All costs related to backfill manufacturing and distribution.

Operating costs for mining include all associated costs for ore development, longhole stoping, and Drift-and-Fill stoping, labour, power, water, fleet running costs, backfill.

21.4.3 Area 3000 – Concentrator

The capital cost required to establish the concentrator plant in Phase 1 (770 ktpa) and then the concentrator plant in Phase 2 (4.4 Mtpa) and associated infrastructure to treat 770 ktpa at steady state ore production in Phase 1 and 5.2 Mtpa at steady state ore production in Phase 2, producing concentrate to the correct volumes and specification. EPCM and spares are also included.

The costs consider plant direct and indirect costs, and includes in-plant infrastructure, but excludes mine-wide infrastructure and TSF costing. In order to compare these costs, the US dollar component was escalated by the annual United States Consumer Price Index and the ZAR portion escalated by annual ZAR CPI estimates. Estimates derived for the study compared well with other projects successfully concluded during this period.

21.4.4 Area 6000 – Infrastructure, Utilities and Ancillaries

Key assumptions relating to the plant and to the infrastructure capital to establish bulk, external and common infrastructure supporting mine and concentrator steady state production are:

Bulk Power

The bulk water estimate includes the construction of the Masodi WWTW as per the offtake agreement with Mokopane LDM, Municipalities' Masodi water supply, and no allowance has been made for the Olifants River Water Resource Development Project (ORWRDP) water.

Bulk Water

The bulk water estimate includes the offtake agreement with Mokopane Municipalities' Masodi water supply, and no allowance has been made for Olifants River Water Resource Development Project (ORWRDP) water.

Preliminary and General

The P&G estimates were developed both as factorised and first principles estimates. Due to the level of detail associated with Shaft 2 designs and schedule, associated P&G estimates were developed from first principles.

The remainder of the P&G estimates were all factored based, from quotations provided by the contractors.

Tailings Storage Facility

The capital cost for the TSFs were estimated by Golder Associates. The cost of the earthworks was estimated by applying the selected earthworks rates and liner costs to a to SANS 1200 BOQ. The cost on the dewatering plant was based on quotes received from the market. Platreef 2017 FS assumed a hybrid paddock deposition TSF. A FS for a stacked tailings storage facility for Platreef Mine was undertaken by Golder Associates (December 2016). Dry Stacking TSF costs were sourced from the Golder report, and were to Feasibility Study level of accuracy.

Project Construction Facilities

The project philosophy is to ensure as much as possible of the permanent infrastructure is built early and used as construction facilities.

Note: By building infrastructure early, not all facilities could be catered for. In these instances, associated costs were estimated using historical information and analogue estimation. Costs from previous projects have been included as part of the estimate.

Spares

Spares were collated into three groups, comprising capital spares, operating spares and commissioning spares. Capital spares is provided by the project capital, and therefore included in this estimate. Large capital spares were included as individual items in the procurement packages and priced accordingly. For the smaller capital spares, a 3.5% over and above allowance was made on the capital costs of each package. Operating spares was included as part of the operating cost estimate, and commissioning spares was priced as part of the 3.5% allowance.

EPCM

For the Platreef 2022 FS the EPCM estimate was developed as a factored estimate related to the direct field costs, and not based on any labour hour schedules.

21.4.5 Area 7000 – Site Costs

Site based capital costs required during building the project:

- Site Safety, Health, and Environment,
- Ivanplats Project Development team,
- Project Development Fleet, and
- Site security.
- COVID Testing
- Hygiene sampling
- Software licenses
- Office running costs (not covered in other areas of the estimate

21.4.6 Area 8000 – Owners Costs, Administration and Overheads

The Capital costs associated with:

Project Services

Project execution services other than EPCM. This includes but not limited to:

- Commissioning
- Project consultant fee's (other than EPCM)
- Financing fees
- Logistics and transport

Owners Team Administration and Overheads

Capital costs associated with owner's team administration and overheads; this includes:

- Traveling costs
- Compliance and government fees
- Mine IT costs
- Legal and Joint Venture (JV) fees
- Management fees

Environmental and Social

Capital costs commitments as required by the environmental management plan and implementation, together with all future enterprise development and social and labour plan commitments, as required by legislation.

Closure Costs

The Guideline Document for Evaluation of the Quantum of Closure-Related Financial Provision Provided by a Mine was used to assess the applicant's environmental liability at the feasibility stage. The Department of Mineral Resources (DMR) Guideline format makes use of a set template, for which defined rates and multiplication factors are used. Multiplication and weighting factors which ultimately define the rate to be used, are determined by topography, classification of the mine according to mineral mined, the risk class of the mine and its proximity to build up, or urban areas.

21.4.7 Area 9000 – Contingency

The estimate is presented in real terms based on Q4'20 pricing received. This includes all capital estimate contingency allowed for. A line-by-line approach was applied to estimating the contingency. Contingency was applied onto Capital Items under Area 2000, 3000 and 6000 in the estimate with a nett outcome of 5.26% on the items in these areas.

21.5 Cost Basis and Exchange Rates

The base date for the capital estimate was end Q4'20August 2021. In the Platreef 2020 FS each item was escalated according to the Escalation Factors. Each line item in the cost estimate was recorded in its base currency and converted to ZAR based using the exchange rates shown in Table 21.1.

Table 21.1 Platreef 2022 Exchange Rates

Exchange Rate	Rate
ZAR/USD	16.00
EUR/USD	0.71
AUD/USD	1.18
CNY/USD	5.70
GBP/USD	0.63
JPY/USD	100.00
NOK/USD	7.00
SEK/USD	7.00

21.6 Project Execution and Life-of-Mine Schedule

Each phase of the project has a critical path both of which exist within the project schedule. Critical path of phase one leads to the first concentrate date and the critical path of phase two leads to the steady state mining production date. Any slippage on the first path will delay first concentrate and hence first revenue. Slippage on the second critical path will result in delay in meeting the 5.17 Mtpa steady state production target. The key dates are shown in Table 21.2.

Table 21.2 Platreef 2022 FS Key Dates

Activity Name	Start	Finish
Shaft 1 commissioning		Q1'22
Restart Development from Shaft 1	Q2'22	
Ventilation Raise 1 (750 m level to Surface)	Q1'23	Q4'23
Ventilation Raise 1 (950 m Level to 750 m Level)	Q4'23	Q1'24
Shaft 2 Sinking to –60 m Level	Q3'23	Q4'24
First Concentrator	Q3'24	
Shaft 2 Sinking to –114 m Level	Q4'24	Q4'24
Shaft 2 Sinking to –750 m Level	Q4'24	Q4'25
Shaft 2 Sinking to –850 m Level	Q4'25	Q1'26
Shaft 2 Sinking to –950 m Level	Q1'26	Q1'26
Shaft 2 Sinking to –1,050 m Level	Q1'26	Q2'26
Shaft 2 Sinking to –1,100 m level	Q2'26	Q3'26
Shaft 2 Equipping Complete		Q3'27
Start of mining ramp up for first 2.2 Mtpa concentrator	Q1'28	
Start of mining ramp up for second 2.2 Mtpa concentrator	Q1'30	
Mine Production Steady State (5.2 Mtpa)	Q4'30	

21.7 Cost Estimation Method

The following techniques were used in preparation of the capital and operating cost estimates of the Platreef 2022 FS:

- Vendor or Contractor Bid Analysis: The primary approach used in developing the capital estimate. Owing to the high level of design detail available, acquisition of competitive bids from the marketplace as an estimating method was used successfully.
- Bottom-up and first principle estimating: Certain portions of the estimate were developed from first principles, where smaller portions of the work was estimated and rolled up to develop sections of the estimate.
- Expert Judgement with historical information: Estimates that were made using experience and historical information from projects, with a similar environment or geographical area of South Africa, that were updated for the Platreef conditions.
- Analogous estimating: Used to estimate applicable portions of the scope not included in the Vendor or contractor Bid Analysis. Costs from previous projects of similar scope and items were used to develop applicable sections of the estimate.
- Contingency Reserve analyses: Used to develop contingency allowances.

The estimate has been based on the assumptions, that the Project will be executed on an Engineering, Procurement and Construction Management (EPCM) basis.

A detailed procurement operating plan was developed, identifying various procurement packages and proposed vendors or contractors for each package. These vendors and contractors were approved by Ivanplats, not only from a technical perspective, but also in terms of their adherence to Social and Labour Plan requirements.

Depending on the complexity and anticipated value of the packages, terms and conditions were assigned to each package. In most instances, typical Fédération Internationale des Ingénieurs-Conseils terms and conditions were referred to. For each package, a detailed scope of work, datasheets and or bill of quantities were developed, to complete an enquiry package to be issued to the marketplace. These were adjudicated accordingly and applied to the overall estimate.

For the mining estimate of underground mining and development requirements, enquiries were issued to both local and international contractors with detailed scopes of work, a full set of terms and conditions, detailed mine schedule and various supporting documents. Tenders received were evaluated by Ivanplats on cost, and the capability of the contractors to execute the development and mining rates in the mine schedule.

P&G estimates were developed both as factorised and first principles estimates. Owing to the level of detail associated with the designs and schedule of Shaft 2, associated P&G estimates were developed from first principles. The remainder of the P&G estimates were all factored from quotations received.

Spare parts estimates were accounted for in three principal groups: (i) capital, (ii) operating and (iii) commissioning. The high value and long lead time capital spares were included as individual items in the procurement packages and priced accordingly. An allowance of 3.5% of the capital value was included as an allowance for other capital spares.

Operating spares are included as part of the operating cost estimate and commissioning spares are priced as part of the 3.5% allowance.

Mine closure capital costs are based on the Department of Mineral Resources guideline entitled Evaluation of the Quantum of Closure-Related Financial Provision Provided by a Mine.

The cost estimate is a project basis costing, it does not include escalation and the cost of project financing.

All expenditure prior to January 2022 were classified as sunk costs and have not been included in the capital estimate.

A number of review sessions were held for purposes of quantifying the overall estimate quality, including detailed reviews throughout the development of the Platreef 2022 FS.

Costs estimated in ZAR have been converted to US dollars at an exchange rate of 16 ZAR/USD. A comparison between the exchange rates used in the Platreef 2022 FS and the 2017 FS is shown in Table 21.3.

Table 21.3 Exchange Rates

Exchange Rates	Forex Amount	
	2022 FS	2017 FS
ZAR	16.00	13.00
EUR	0.71	0.79
AUD	1.18	1.18
CNY	5.70	5.91
GBP	0.63	0.67
JPY	100.00	92.86
NOK	7.00	7.39
SEK	7.00	7.56

21.8 Capital Cost Summary

The total capital cost includes initial (pre-production) capital, expansion capital and sustaining capital. Pre production capital includes all direct and indirect mine development and construction costs prior to the first concentrate. Owner's costs have been allowed for, which includes drilling campaigns, sampling, assaying, salaries and wages, community, office administration costs, Health, Safety and Environmental (HSE), and site office allowance up to operations phase. The capital expenditure summary is shown in Table 21.4.

Table 21.4 Capital Expenditure Summaries

Description	Initial ($M)	Expansion ($M)	Sustaining ($M)	Total ($M)
Mining				
Geology	9	31	32	72
Mining	187	697	861	1,744
Capitalised Operating Costs	–	–	–	–
Subtotal	195	728	893	1,816
Concentrator and Tailings				
Concentrator	73	273	2	349
Capitalised Operating Costs	–	–	–	–
Subtotal	73	273	2	349
Infrastructure				
Infrastructure	87	251	25	363
Site Costs	7	0	0	7
Capitalised Operating Costs	–	–	–	–
Subtotal	95	251	25	371
Owners Cost				
Owners Cost	93	126	2	222
Closure Cost	–	–	11	11
Capitalised Operating Costs	–	–	–	–
Subtotal	93	126	13	233
Capex Before Contingency	456	1,378	933	2,768
Contingency	32	101	1	134
Capex After Contingency	488	1,480	934	2,902

1. Initial Capital for the preproduction time including $50M in Shaft 2.
2. Totals vary due to rounding.

21.9 Operating Cost Summary

The operating costs are summarised in Table 21.5.

Table 21.5 Operating Costs

Description	LOM Total ($M)	Milled ($/t)		
		Years 1–4 Average	Years 5–9 Average	LOM Average
Concentrate Transport	195	2.12	1.55	1.56
Site Operating Costs				
Mining	4,005	72.12	31.22	31.98
Processing and Tailings	1,593	21.38	12.56	12.72
Infrastructure	289	9.44	2.17	2.30
Site Cost	160	3.94	1.23	1.28
General and Administration	447	26.11	3.15	3.57
Escalation and Contingency	–	–	–	–
Total	6,493	132.99	50.32	51.86

Totals may vary due to rounding.

22 ECONOMIC ANALYSIS

22.1 Summary of Financial Results

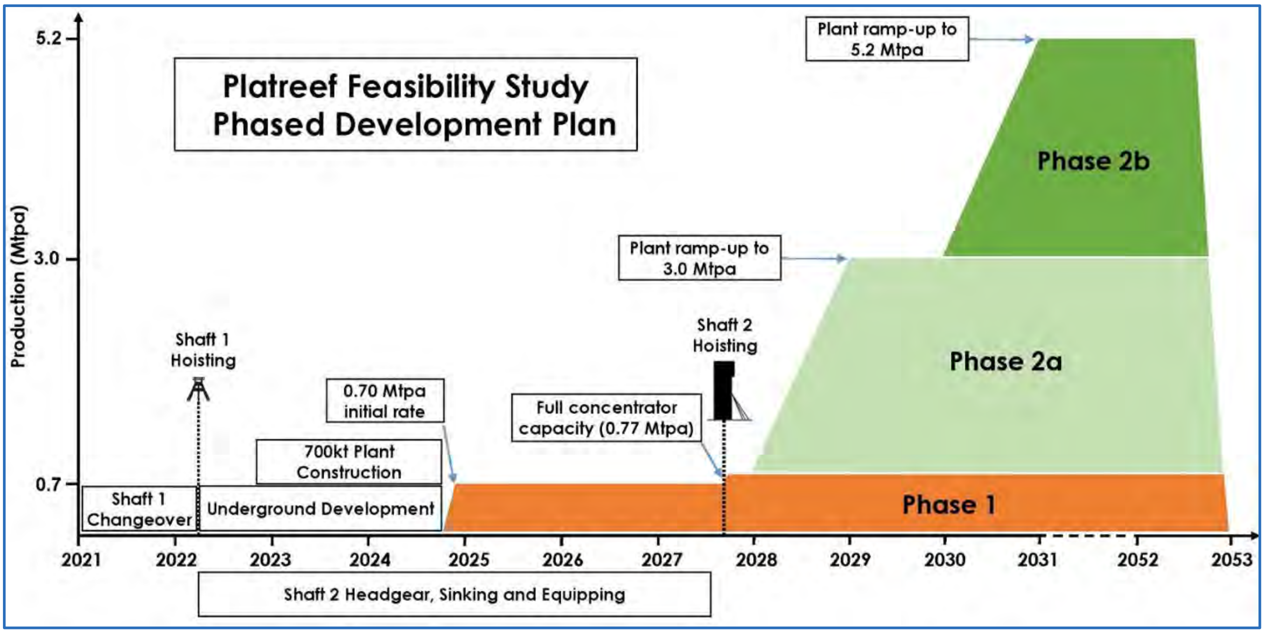
The key features of the Platreef 2022 FS include:

- Development of a large, mechanised, underground mine is planned at an initial 700 ktpa and expansion to 5.2 Mtpa.
- Planned average annual production rate of 522 koz of platinum, palladium, rhodium and gold (3PE+Au)
- Estimated pre-production capital requirement of approximately $488M.
- After-tax NPV of $1,690M, at an 8% discount rate.
- After-tax IRR of 18.48%.
- The Platreef 2022 FS maintains options available to accelerate expansions, to the 8 Mtpa or the 12 Mtpa scenarios, as the market dictates.

Mine production is shown in Figure 22.2, and the after-tax cash flow is shown in Figure 22.3. The key production and financial results including Net Present Value at 8% Discount Rate (NPV8%) and Internal Rate of Return (IRR) of the Platreef 2022 FS are shown in Table 22.1.

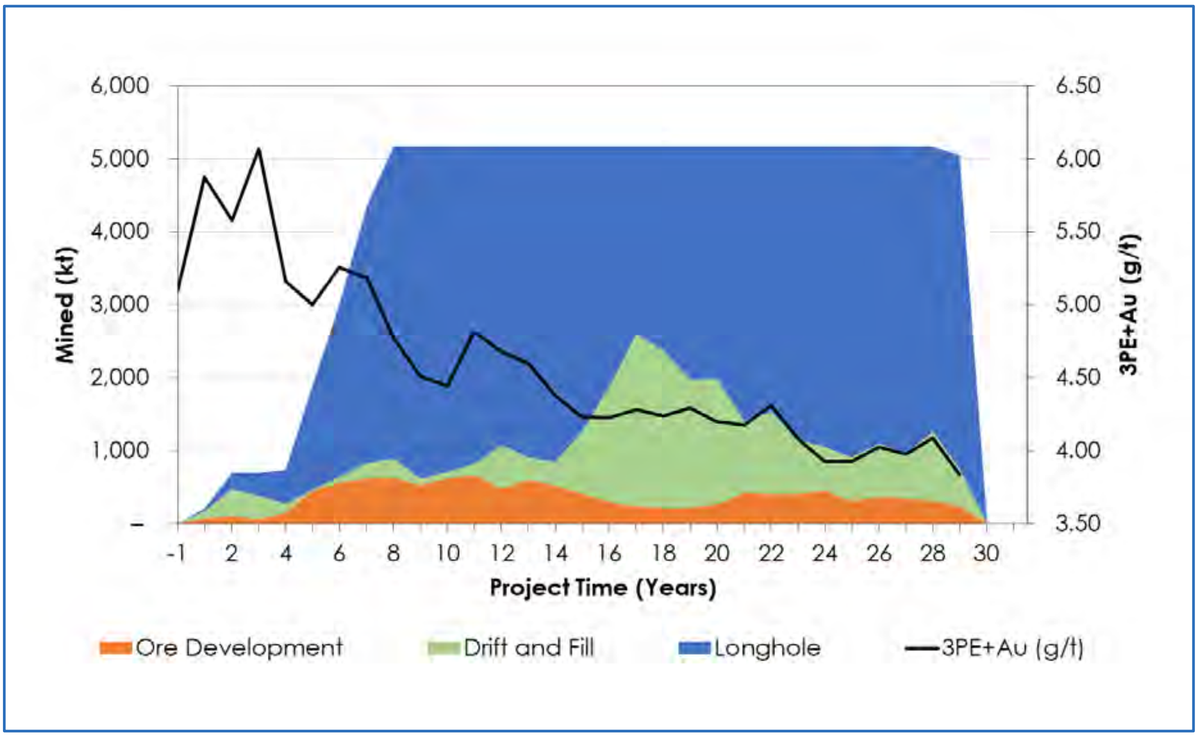
Figure 22.1 shows the Development and Production Timeline for the Platreef 2022 FS.

Figure 22.1 Platreef 2022 FS Development and Production Timeline



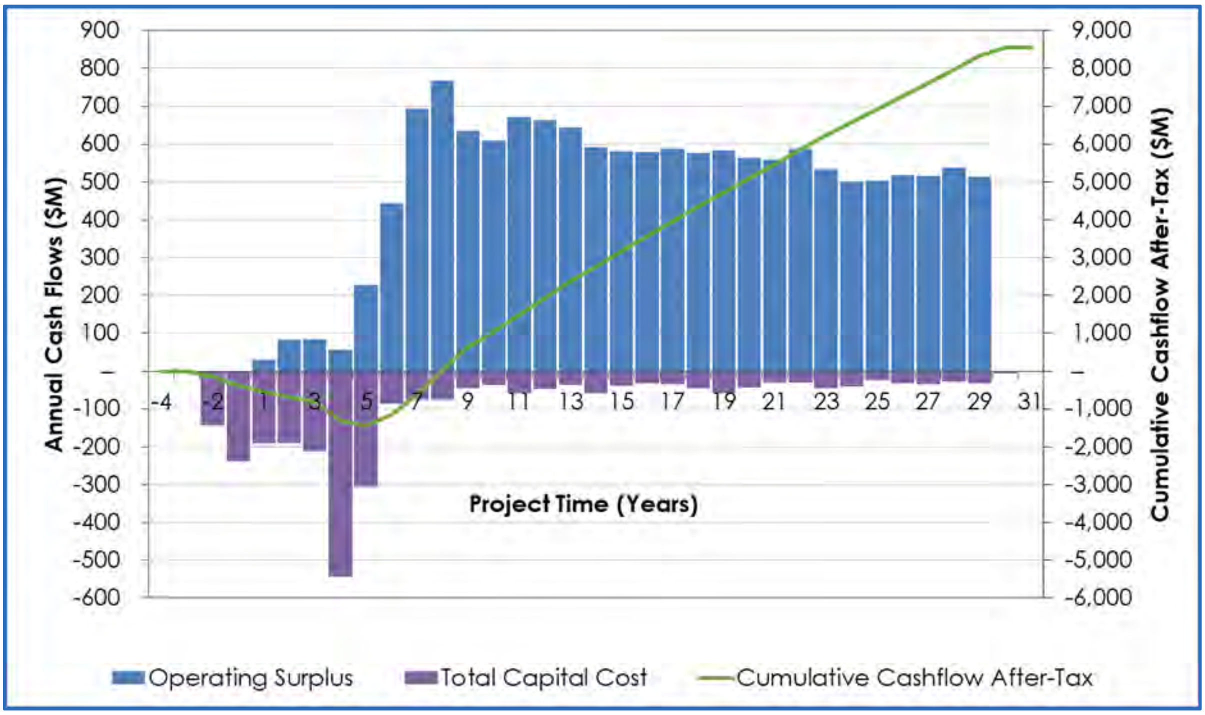
OreWin, 2021

Figure 22.2 Platreef 2022 FS Mining Production



OreWin, 2021

Figure 22.3 Platreef 2022 FS Cumulative Cash Flow After-Tax



OreWin, 2022

Table 22.1 Platreef 2022 FS Key Production and Financial Results

Item	Unit	Total
Mined and Processed	Mt	125
Platinum	g/t	1.94
Palladium	g/t	1.99
Gold	g/t	0.30
Rhodium	g/t	0.13
3PE+Au	g/t	4.37
Copper	% Cu	0.16
Nickel	% Ni	0.34
Concentrate Produced	kt	5,545
Platinum	g/t	38.2
Palladium	g/t	39.0
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE+Au	g/t	85.0
Copper	% Cu	3.3
Nickel	% Ni	5.4
Recovered Metal		
Platinum	koz	6,813
Palladium	koz	6,954
Gold	koz	948
Rhodium	koz	433
3PE+Au	koz	15,149
Copper	Mlb	399
Nickel	Mlb	665
Key Financial Results		
Life-of-Mine	Years	29
Initial (Pre-Production) Capital	$M	488
Expansion Capital	$M	1,480
Sustaining Capital	$M	934
Mine-Site Cash Cost	$/oz Rec. 3PE+Au	429
Total Cash Costs After Credits	$/oz Rec. 3PE+Au	452
Site Operating Costs	$/t Milled	52
After-Tax NPV8%	$M	1,690
After-Tax IRR	%	18.48
Project Payback Period	Years	7.93

1. Initial Capital including $50M in Shaft 2 and $32M in contingencies
2. Totals may not add due to rounding.
3. 3PE+Au = platinum, palladium, rhodium and gold.
4. Economic analysis metal price assumptions: $1,100/oz platinum, $1,450/oz palladium, $1,600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

22.2 Model Assumptions

22.2.1 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2022. It is presented in 2022 constant dollars, cash flows are assumed to occur evenly during each year, and a mid-year discounting approach is taken. The base case real discount factor applied to the analyses is 8%. No allowance for inflation has been made in the analyses.

The prices in the economic analysis for the Platreef 2022 FS are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. The commodity price assumptions for the Platreef 2022 FS are shown in Table 22.2.

Table 22.2 Platreef 2022 FS Commodity Price Assumptions

Parameter	Unit	Financial Analysis Assumptions
Platinum	$/oz	1,100
Palladium	$/oz	1,450
Gold	$/oz	1,600
Rhodium	$/oz	5,000
Copper	$/lb	3.50
Nickel	$/lb	8.00

22.2.2 Treatment Charges and Refining Charges

In the Platreef 2022 FS, payables have been assumed on the basis of the two offtake arrangements and expectations for the life-of-mine concentrate production. Refining charges are shown in the Table 22.3.

Table 22.3 Platreef 2022 FS Refining Charges (% Gross Sales)

	Refining Charges
Copper	27.0%
Nickel	30.0%
Platinum	16.5%
Palladium	16.5%
Gold	16.5%
Rhodium	17.5%

22.2.3 Concentrate Transport Costs

In the Platreef 2022 FS, concentrate transport cost based on the distance of 270 km is ZAR 1.23/t/km and the average distance of the smelters is 277 km. The transport cost applied to the financial model is $0.08 per wet tonne concentrate per km.

22.2.4 Royalties and Taxes

The majority of taxes and fees payable to the government under Republic of South Africa legislation are the Corporate Income Tax (28%) and a production royalty. The royalty rate for refined minerals is a percentage determined as per Section 4 of the Republic of South Africa Royalty Act 28 (2008; Government Gazette No. 31635), and the Mineral and Petroleum Resources Royalty (Administration) Act No. 29 (2008; Government Gazette No. 31642).

Royalty % = 0.5 + [EBIT/ (Gross Sales x 9)] x 100, with a maximum of 7%, for production of unrefined minerals.

Assumptions for the royalties and taxes are shown in Table 22.4.

Table 22.4 Platreef 2022 FS Royalties and Taxes

Royalties		
Base Factor	%	0.50
Unrefined Mineral Factor		9.00
Pct Factor Not to be Exceeded	%	7.00
Taxes		
Corporate Income Tax Rate	%	28.00
Opening Tax Losses	million ZAR	305
Opening Depreciation	million ZAR	7,468
Working Capital		
Receivables	weeks	15.00
Payables	weeks	4.00

22.2.5 Exchange Rates

Costs estimated in ZAR have been converted to US dollars at an exchange rate of 16 ZAR/USD. A comparison between the exchange rates used in the Platreef 2022 FS and the 2017 FS is shown in Table 22.5.

Table 22.5 Exchange Rates

Exchange Rates	Forex Amount	
	2022 FS	2017 FS
ZAR	16.00	13.00
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GBP	0.63	0.67
JPY	100.00	92.86
NOK	7.00	7.39
SEK	7.00	7.56

22.3 Project Results

The results of the financial analysis show an After-Tax NPV8% of $1,690M. The Platreef 2022 FS exhibits an after-tax IRR of 18.48% and a payback period of approximately eight years. The estimates of cash flows have been prepared on a real basis as at 1 January 2022 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 22.6. The mining production statistics are shown in Table 22.7.

Table 22.6 Platreef 2022 FS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	11,535	8,543
	5.0%	4,242	3,098
	8.0%	2,369	1,690
	10.0%	1,594	1,104
	12.0%	1,051	692
	15.0%	513	283
	20.0%	33	-83
Internal Rate of Return	–	20.54%	18.48%
Project Payback Period (Years)	–	7.93	7.93

Concentrator feed and estimated concentrator produced along with grades for the life of mine are depicted in Figure 22.4 and Figure 22.5.

Table 22.7 Platreef 2022 FS Mining Production Statistics

Item	Unit	Total LOM	Years 1–4 Average	Years 5-29 Average	LOM Average
Mined and Processed	Mt	125	0.70	4.92	4.42
Platinum	g/t	1.94	2.53	1.93	1.94
Palladium	g/t	1.99	2.54	1.98	1.99
Gold	g/t	0.30	0.38	0.30	0.30
Rhodium	g/t	0.13	0.17	0.13	0.13
3PE+Au	g/t	4.37	5.63	4.34	4.37
Copper	% Cu	0.16	0.19	0.16	0.16
Nickel	% Ni	0.34	0.40	0.34	0.34
Concentrator Recoveries					
Platinum	%	87.23	90.39	87.15	87.2
Palladium	%	86.76	90.18	86.68	86.8
Gold	%	78.54	80.39	78.49	78.5
Rhodium	%	80.28	84.37	80.18	80.3
3PE+Au	%	86.21	89.43	85.99	86.2
Copper	%	87.70	90.04	87.65	87.7
Nickel	%	71.58	77.54	71.44	71.6
Concentrate Produced	kt (dry)	5,545	35	216	196
Platinum	g/t	38.21	38.11	38.22	38.2
Palladium	g/t	39.00	38.10	39.03	39.0
Gold	g/t	5.32	5.11	5.33	5.3
Rhodium	g/t	2.43	2.45	2.43	2.4
3PE+Au	g/t	84.97	83.76	85.00	85.0
Copper	% Cu	3.26	2.82	3.28	3.3
Nickel	% Ni	5.44	5.11	5.45	5.4
Recovered Metal					
Platinum	koz	6,813	51	266	240
Palladium	koz	6,954	51	271	245
Gold	koz	948	7	37	33
Rhodium	koz	433	3	17	15
3PE+Au	koz	15,149	113	591	535
Copper	Mlb	399	3	16	14
Nickel	Mlb	665	5	26	23

Figure 22.4 Platreef 2022 FS Concentrator Production

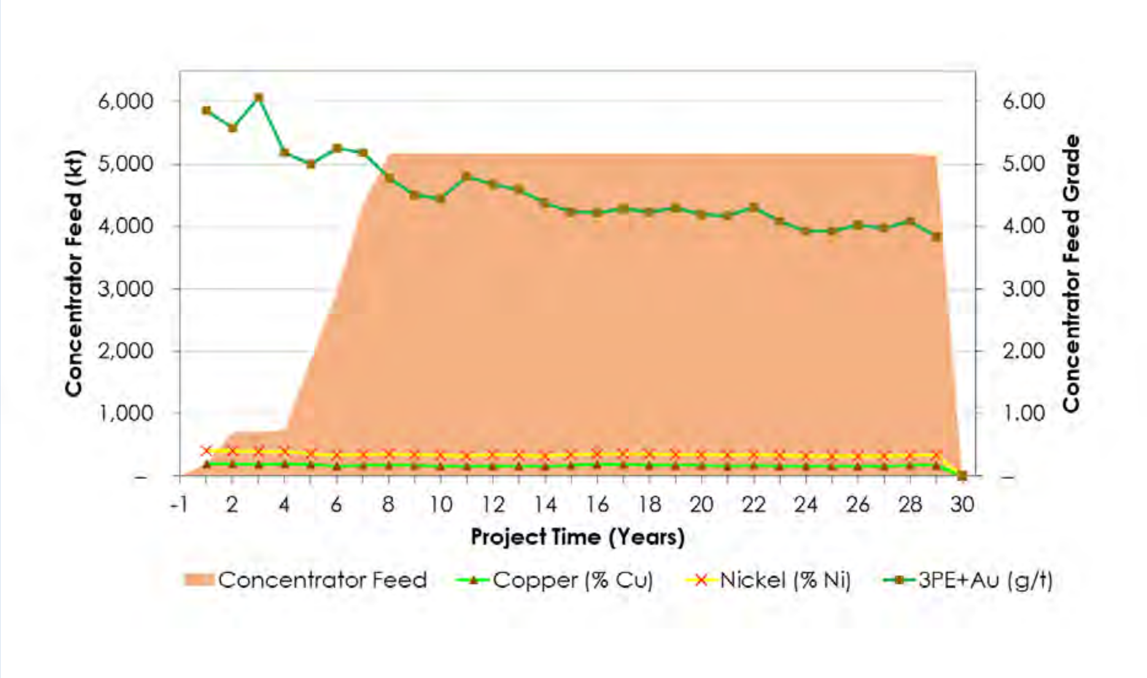

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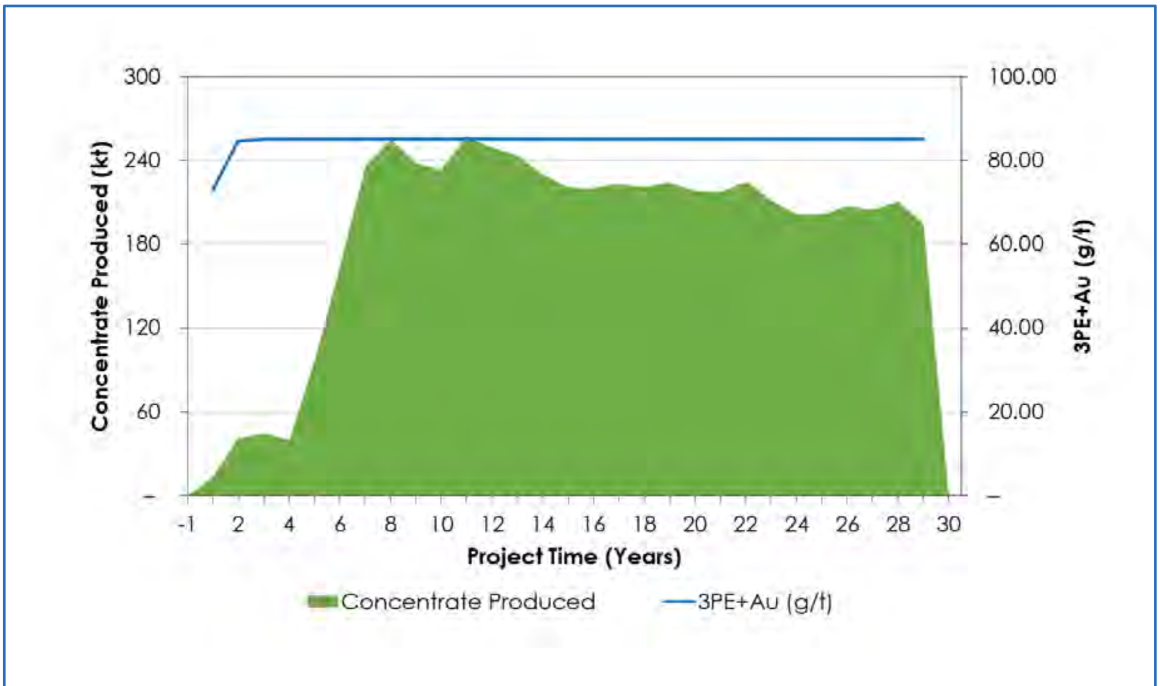
Figure 22.5 Platreef 2022 FS Estimated Concentrate Produced and 3PE+Au Grade



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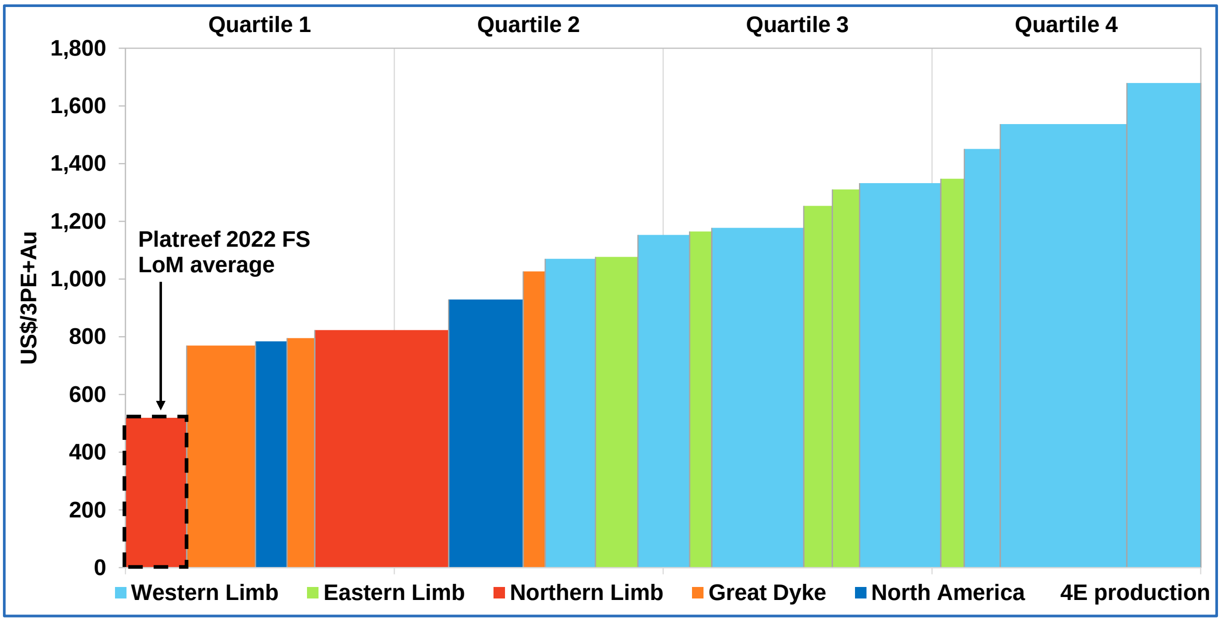
22.4 Capital and Operating Cost Summary

Mine site cash costs are summarised in Table 22.8. The revenues and operating costs are presented in Table 22.9. Comparisons to global primary producers and ranking of selected global primary PGM producers shown in Figure 22.6 and Figure 22.7.

Table 22.8 Platreef 2022 FS Cash Costs After Credits

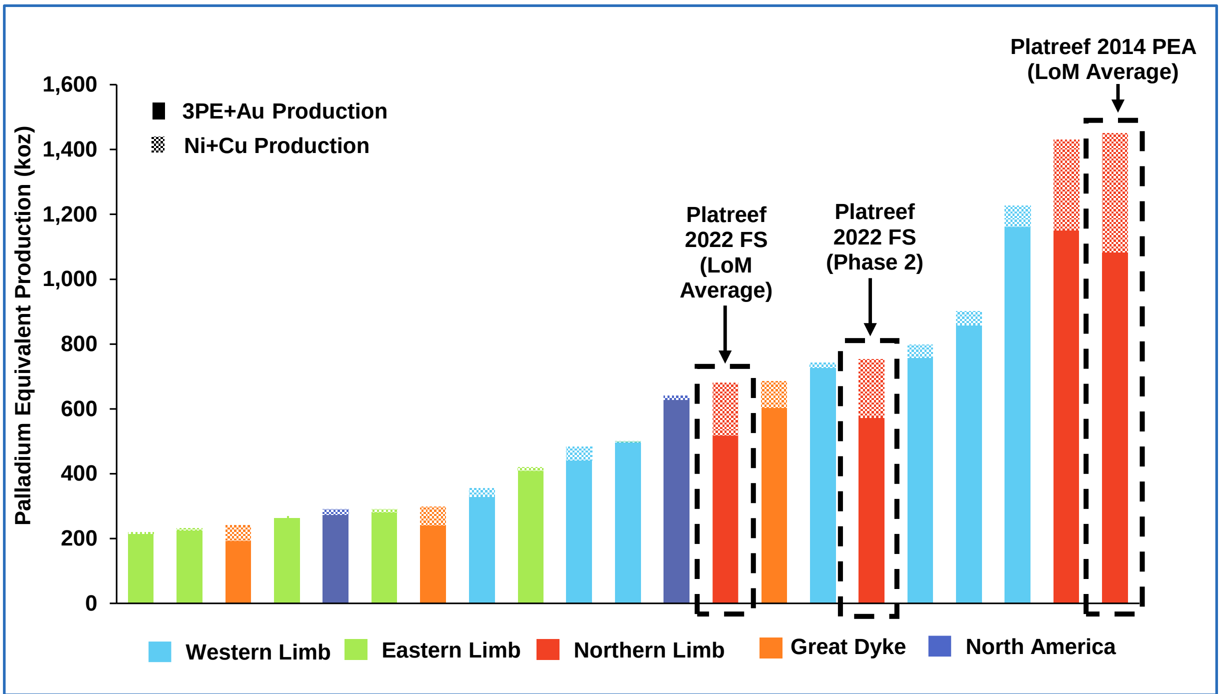
Item	Recovered 3PE+Au ($/oz)		
	Years 1–4 Average	Years 5-29 Average	LOM Average
Mine Site cash costs	822	419	429
Transport	13	13	13
Treatment & Refining	369	366	366
Royalties	8	90	88
Total Cash Costs Before Credits	1,212	887	895
Nickel Credits	334	351	351
Copper Credits	84	92	92
Total Cash Costs After Credits	794	443	452
Sustaining Capital Costs	–	63	62
Total Cash Costs After Credits & Sustaining Capital	794	506	514

Figure 22.6 Global Primary Producers' Net Total Cash Cost + Sustaining Capital (2021E), US$/3PE+Au oz.



SFA (Oxford), Ivanplats. Notes: Cost and production data for the Platreef project is based on the Platreef 2022 FS parameters, applying payabilities and smelting and refining charges as agreed with purchase of concentrate partners for Platreef concentrate (this is not representative of SFA's standard methodology). SFA's peer group cost and production data follows a methodology to provide a level playing field for smelting and refining costs on a pro-rata basis from the producer processing entity. Net total cash costs have been calculated using Ivanplats' long term price assumptions of 16:1 ZAR:USD, US$1,100/oz platinum, US$1,450/oz palladium, US$5,000/oz rhodium, US$1,600/oz gold, US$8.00/lb nickel and US$3.50/lb copper.

Figure 22.7 Ranking of Selected Global Primary PGM Producers, Based on 2021E Palladium Equivalent Production.



Source: SFA (Oxford), Ivanplats. Notes: Chart excludes by-product PGM producers. Nornickel (by-product PGM producer) is the largest producer on a palladium equivalent basis. Cost and production data for the Platreef project is based on the Platreef 2022 FS and 2014 PEA parameters. Production data for the peer group is provided by SFA (Oxford). Equivalent palladium production has been calculated using Ivanplats' long term price assumptions of 16:1 ZAR:USD, US$1,100/oz platinum, US$1,450/oz palladium, US$5,000/oz rhodium, US$1,600/oz gold, US$8.00/lb nickel and US$3.50/lb copper.

Table 22.9 Platreef 2022 FS Operating Costs and Revenues

Item	LOM Total ($M)	Milled ($/t)		
		Years 1–4 Average	Years 5–29 Average	LOM Average
Gross Sales Revenue	28,002	305.48	222.09	223.64
Less: Realisation Costs				
Transport	195	2.12	1.55	1.56
Treatment and Refining	5,540	59.76	43.95	44.25
Royalties	1,337	1.23	10.85	10.67
Total Realisation Costs	7,072	63.10	56.36	56.48
Net Sales Revenue	20,930	242.37	165.74	167.16
Site Operating Costs				
Mining	4,005	72.12	31.22	31.98
Processing and Tailings	1,593	21.38	12.56	12.72
Infrastructure	289	9.44	2.17	2.30
Site Cost	160	3.94	1.23	1.28
General and Administration	447	26.11	3.15	3.57
Escalation and Contingency	–	–	–	–
Total	6,493	132.99	50.32	51.86
Operating Margin	14,437	109.38	115.41	115.30
Operating Margin	52%	36%	52%	52%

Totals may vary due to rounding.

The total initial (pre-production), expansion and sustaining capital costs required are shown in Table 22.10.

Table 22.10 Platreef 2022 FS Total Project Capital Cost

Description	Initial ($M)	Expansion ($)	Sustaining ($M)	Total ($)
Mining				
Geology	9	31	32	72
Mining	187	697	861	1,744
Capitalised Operating Costs	–	–	–	–
Subtotal	195	728	893	1,816
Concentrator and Tailings				
Concentrator	73	273	2	349
Capitalised Operating Costs	–	–	–	–
Subtotal	73	273	2	349
Infrastructure				
Infrastructure	87	251	25	363
Site Costs	7	0	0	7
Capitalised Operating Costs	–	–	–	–
Subtotal	95	251	25	371
Owners Cost				
Owners Cost	93	126	2	222
Closure Cost	–	–	11	11
Subtotal	93	126	13	233
Capex Before Contingency	456	1,378	933	2,768
Contingency	32	101	1	134
Capex After Contingency	488	1,480	934	2,902

1. Initial Capital for the preproduction time including $50M in Shaft 2.
2. Totals may vary due to rounding.

22.5 Project Production and Cash Flows

Cumulative cash flow after-tax is depicted in Figure 22.8. Year minus 4 is 2020 and Year 1 is 2024. Year 1 is the commencement of production and the plant is only in operation for part of the year commencing in Q3'24. The details of mine and process production and the project cash flow are shown in Table 22.11 to Table 22.16. Year -2 is 2022.

Figure 22.8 Platreef 2022 FS Cumulative Cash Flow After-Tax

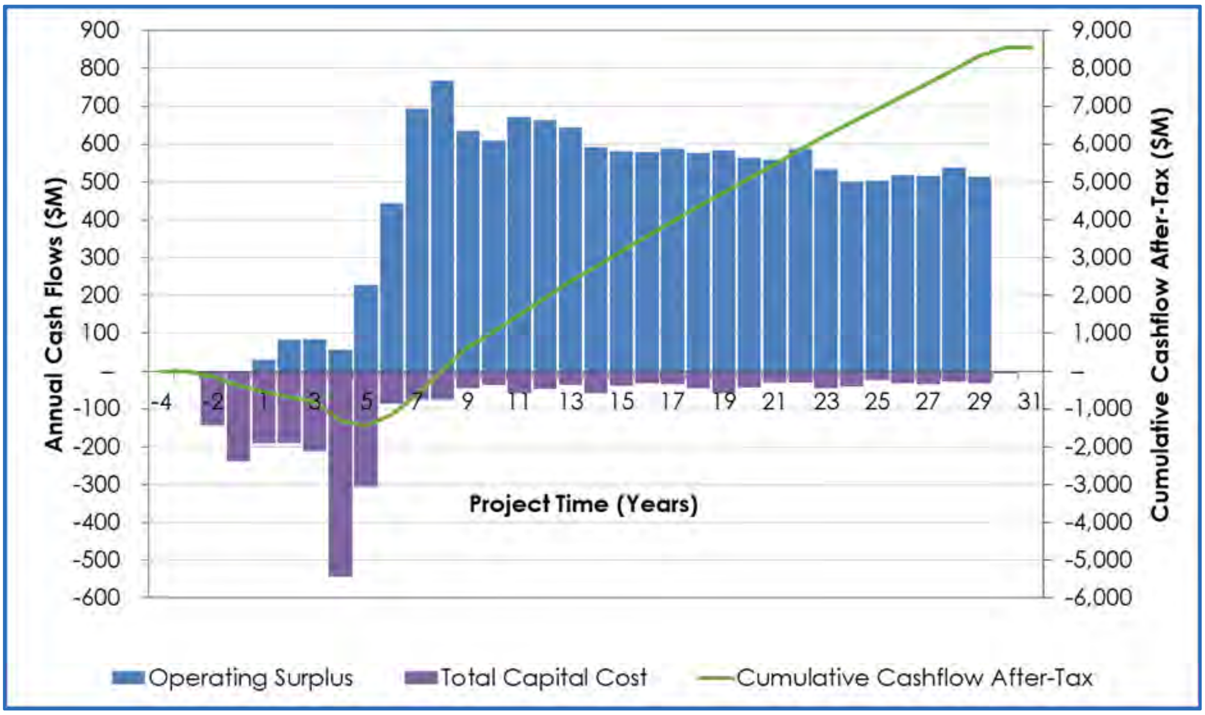


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Table 22.11 Platreef 2022 FS Mine Production

Item	Year Number	Total	-1	1	2	3	4	5	6	7	8	9	10	11	21
	Year To													20	LOM
Total Ore Mined	kt	125,212	2	209	700	700	735	1,885	3,000	4,356	5,170	5,170	5,170	51,703	46,411
NSR BDT20	$/t	156	200	211	200	215	189	178	181	180	169	160	157	158	146
Platinum	g/t	1.94	2.23	2.64	2.43	2.80	2.34	2.26	2.34	2.26	2.08	1.99	1.99	1.95	1.81
Palladium	g/t	1.99	2.37	2.65	2.63	2.67	2.28	2.25	2.43	2.44	2.24	2.09	2.03	2.01	1.81
Gold	g/t	0.30	0.36	0.41	0.35	0.41	0.39	0.34	0.32	0.32	0.31	0.29	0.28	0.30	0.30
Rhodium	g/t	0.13	0.14	0.17	0.18	0.20	0.15	0.15	0.17	0.17	0.15	0.14	0.14	0.13	0.12
Copper	% Cu	0.16	0.23	0.19	0.19	0.19	0.19	0.18	0.16	0.17	0.17	0.16	0.16	0.17	0.16
Nickel	% Ni	0.34	0.49	0.41	0.40	0.39	0.39	0.36	0.34	0.35	0.35	0.34	0.33	0.34	0.33
Sulfur	% S	0.82	1.21	1.01	1.00	0.95	0.96	0.88	0.79	0.86	0.87	0.85	0.81	0.82	0.80
3PE+Au	g/t	4.37	5.10	5.88	5.58	6.07	5.16	5.00	5.25	5.19	4.77	4.51	4.44	4.39	4.04

Note: NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

Table 22.12 Platreef 2022 FS Process Production Plant Feed

Item	Year Number	Total	-1	1	2	3	4	5	6	7	8	9	10	11	21
	Year To													20	LOM
Concentrator Feed	kt	125,212	–	190	700	700	735	1,870	2,970	4,345	5,170	5,170	5,170	51,700	46,493
NSR BDT20	$/t	156	–	210	200	215	190	178	181	180	170	160	157	158	146
Platinum	g/t	1.94	–	2.63	2.43	2.80	2.35	2.26	2.34	2.26	2.08	1.99	1.99	1.95	1.81
Palladium	g/t	1.99	–	2.65	2.63	2.68	2.29	2.25	2.43	2.45	2.24	2.09	2.03	2.01	1.81
Gold	g/t	0.30	–	0.40	0.35	0.40	0.39	0.34	0.32	0.32	0.31	0.29	0.28	0.30	0.30
Rhodium	g/t	0.13	–	0.17	0.18	0.20	0.15	0.15	0.17	0.17	0.15	0.14	0.14	0.13	0.12
Copper	% Cu	0.16	–	0.19	0.19	0.19	0.19	0.18	0.16	0.17	0.17	0.16	0.16	0.17	0.16
Nickel	% Ni	0.34	–	0.41	0.40	0.39	0.39	0.36	0.34	0.35	0.35	0.34	0.33	0.34	0.33
Sulfur	% S	0.82	–	1.01	1.00	0.95	0.96	0.88	0.79	0.86	0.87	0.85	0.81	0.81	0.80
3PE+Au	g/t	4.37	–	5.86	5.58	6.08	5.18	5.00	5.25	5.19	4.78	4.51	4.44	4.39	4.04

Note NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: $1,600/oz platinum, $815/oz palladium, $1,300/oz gold, $1,500/oz rhodium, $8.90/lb nickel and $3.00/lb copper. Metal-price assumptions used for the Platreef 2022 FS economic analysis are as follows: $1,100/oz platinum, $1,450/oz palladium, $1600/oz gold, $5,000/oz rhodium, $8.00/lb nickel and $3.50/lb copper.

Table 22.13 Platreef 2022 FS Concentrator Recoveries

Item	Year Number	Total	-1	1	2	3	4	5	6	7	8	9	10	11	21
	Year To													20	LOM
Concentrate Produced	kt	5,545	–	13	41	45	40	97	164	236	255	238	234	2,308	1,874
Concentrator Recoveries															
Platinum	%	87.23	–	88.90	90.50	90.90	90.13	89.00	90.46	89.86	88.71	87.73	87.47	87.28	85.94
Palladium	%	86.76	–	88.70	90.30	90.70	89.91	88.78	90.23	89.63	88.47	87.39	87.09	86.85	85.21
Gold	%	78.54	–	78.80	80.43	80.80	80.38	79.47	80.58	80.09	79.56	78.95	78.78	78.66	77.77
Rhodium	%	80.28	–	82.90	84.51	84.90	84.00	82.78	84.41	83.76	82.26	80.97	80.62	80.35	78.45
Copper	%	87.70	–	88.87	90.01	90.09	90.34	88.65	87.47	88.23	88.72	87.65	86.71	87.79	87.31
Nickel	%	71.58	–	77.11	77.82	77.37	77.54	73.39	71.56	72.40	73.38	71.93	70.73	71.89	70.48
Sulfur	%	70.79	–	73.94	75.24	74.40	74.64	71.99	69.89	71.52	72.22	71.77	70.58	70.64	70.28
3PE+Au	%	86.21	–	87.93	89.59	89.95	89.12	88.07	89.56	88.97	87.81	86.80	86.54	86.29	84.78
Mass Pull	%	–	–	7.06	5.90	6.43	5.44	5.18	5.54	5.43	4.94	4.61	4.52	4.46	4.03

Table 22.14 Platreef 2022 FS Concentrate Production

Item	Year Number	Total	-1	1	2	3	4	5	6	7	8	9	10	11	21
	Year To													20	LOM
Concentrate Produced	kt	5,545	–	13	41	45	40	97	164	236	255	238	234	2,308	1,874
Concentrator Grades															
Platinum	g/t	38.21	–	33.14	37.26	39.58	38.99	38.86	38.26	37.44	37.36	37.84	38.57	38.12	38.52
Palladium	g/t	39.00	–	33.21	40.21	37.76	37.93	38.57	39.56	40.36	40.20	39.68	39.07	39.23	38.34
Gold	g/t	5.32	–	4.52	4.75	5.06	5.73	5.21	4.66	4.65	4.94	4.99	4.85	5.23	5.75
Rhodium	g/t	2.43	–	2.04	2.52	2.59	2.34	2.36	2.53	2.56	2.50	2.49	2.52	2.42	2.39
Copper	% Cu	3.26	–	2.44	2.88	2.61	3.14	3.05	2.57	2.80	3.12	3.13	3.00	3.26	3.50
Nickel	% Ni	5.44	–	4.46	5.27	4.70	5.61	5.07	4.35	4.63	5.25	5.31	5.14	5.47	5.71
Sulfur	% S	13.09	–	10.62	12.75	11.01	13.22	12.25	9.94	11.29	12.75	13.30	12.67	12.91	13.99
3PE+Au	g/t	84.97	–	72.91	84.74	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00

Table 22.15 Platreef 2022 FS Metal Production

Item	Year Number	Total	-1	1	2	3	4	5	6	7	8	9	10	11	21
	Year To													20	LOM
Recovered Metal															
Platinum	koz	6,813	–	14	49	57	50	121	202	284	307	290	290	2,828	2,320
Palladium	koz	6,954	–	14	53	55	49	120	209	306	330	304	294	2,910	2,310
Gold	koz	948	–	2	6	7	7	16	25	35	41	38	36	388	346
Rhodium	koz	433	–	1	3	4	3	7	13	19	20	19	19	180	144
Copper	klb	399,108	–	721	2,622	2,590	2,765	6,505	9,325	14,593	17,576	16,429	15,466	165,951	144,567
Nickel	klb	664,740	–	1,317	4,794	4,667	4,940	10,823	15,761	24,103	29,553	27,866	26,514	278,340	236,060
Sulfur	klb	1,599,885	–	3,136	11,604	10,932	11,647	26,131	36,047	58,740	71,706	69,885	65,313	656,786	577,957
3PE+Au	koz	15,149	–	31	112	123	109	264	449	645	697	651	639	6,306	5,120

Table 22.16 Platreef 2022 FS Cash Flow

Cash Flow Statement ($M)		Year													
Year Number	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	20
Year To														20	LOM
Gross Revenue	28,002	–	–	64	217	227	202	479	791	1,154	1,281	1,196	1,164	11,658	9,570
Realisation Costs	7,072	–	–	13	45	46	42	99	162	237	266	307	302	3,044	2,508
Net Sales Revenue	20,930	–	–	51	172	181	159	380	629	917	1,014	890	862	8,614	7,062
Less: Site Operating Costs															
Mining	4,005	–	–	8	49	52	58	86	108	131	150	157	155	1,609	1,440
Processing & Tailings	1,593	–	–	4	15	15	16	37	45	60	64	64	64	637	573
Infrastructure	289	–	–	2	6	7	8	8	9	10	11	11	11	109	98
Site Cost	160	–	–	1	3	3	3	3	4	5	6	7	7	66	53
General & Administration	447	–	–	6	18	18	18	18	18	18	16	16	16	155	131
Subtotal	6,493	–	–	22	90	95	102	153	185	224	246	254	253	2,575	2,295
Operating Surplus / (Deficit)	14,437	–	–	29	82	85	57	227	444	693	768	636	609	6,039	4,767
Capital Costs															
Initial Capital Expenditure	488	143	229	116	–	–	–	–	–	–	–	–	–	–	–
Expansion Capital Expenditure	1,480	–	9	74	190	211	544	306	87	58	–	–	–	–	–
Sustaining Capital Expenditure	934	–	–	–	–	–	–	–	–	19	76	45	36	455	303
Working Capital	–	–	–	13	30	2	-7	60	70	80	27	-20	-7	-12	-236
Subtotal	2,902	143	238	204	220	213	537	366	157	157	102	25	29	443	67
Net Cashflow Before Tax	11,535	-143	-238	-174	-138	-128	-480	-140	288	536	666	611	580	5,596	4,700
Depreciation	-2,902	-143	-238	-191	-190	-211	-544	-306	-87	-77	-76	-45	-36	-455	-303
Income Tax Expense	2,992	–	–	–	–	–	–	–	–	–	–	18	159	1,564	1,252
Net Cashflow After-Tax	8,543	-143	-238	-174	-138	-128	-480	-140	288	536	666	593	421	4,033	3,448

22.6 Price Sensitivity Analysis

A price sensitivity to metal prices analysis of the NPV8% were performed on the financial model and the results are shown in Table 22.17 to Table 22.28 and Figure 22.9 to Figure 22.20. palladium, platinum, and nickel are the major contributors to project revenue.

Table 22.17 After-Tax NPV8% Platinum and Nickel Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Nickel Price - $/lb	550	700	850	1,100	1,250	1,400	1,550
3.00	726	877	1,029	1,286	1,439	1,592	1,744
4.50	846	998	1,152	1,407	1,560	1,712	1,867
6.00	966	1,120	1,274	1,529	1,681	1,836	1,988
8.00	1,129	1,283	1,436	1,690	1,845	1,997	2,149
10.00	1,292	1,445	1,598	1,854	2,006	2,158	2,310
12.50	1,494	1,647	1,800	2,055	2,207	2,359	2,510
15.00	1,696	1,850	2,003	2,256	2,408	2,559	2,710

Table 22.18 After-Tax NPV8% Sensitivity to Platinum and Nickel Prices

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Nickel Price - $/lb	550	700	850	1,100	1,250	1,400	1,550
3.00	-57%	-48%	-39%	-24%	-15%	-6%	3%
4.50	-50%	-41%	-32%	-17%	-8%	1%	10%
6.00	-43%	-34%	-25%	-10%	-1%	9%	18%
8.00	-33%	-24%	-15%	–	9%	18%	27%
10.00	-24%	-15%	-5%	10%	19%	28%	37%
12.50	-12%	-3%	6%	22%	31%	40%	49%
15.00	0%	9%	19%	34%	42%	51%	60%

Figure 22.9 Platinum and Nickel Price Sensitivity

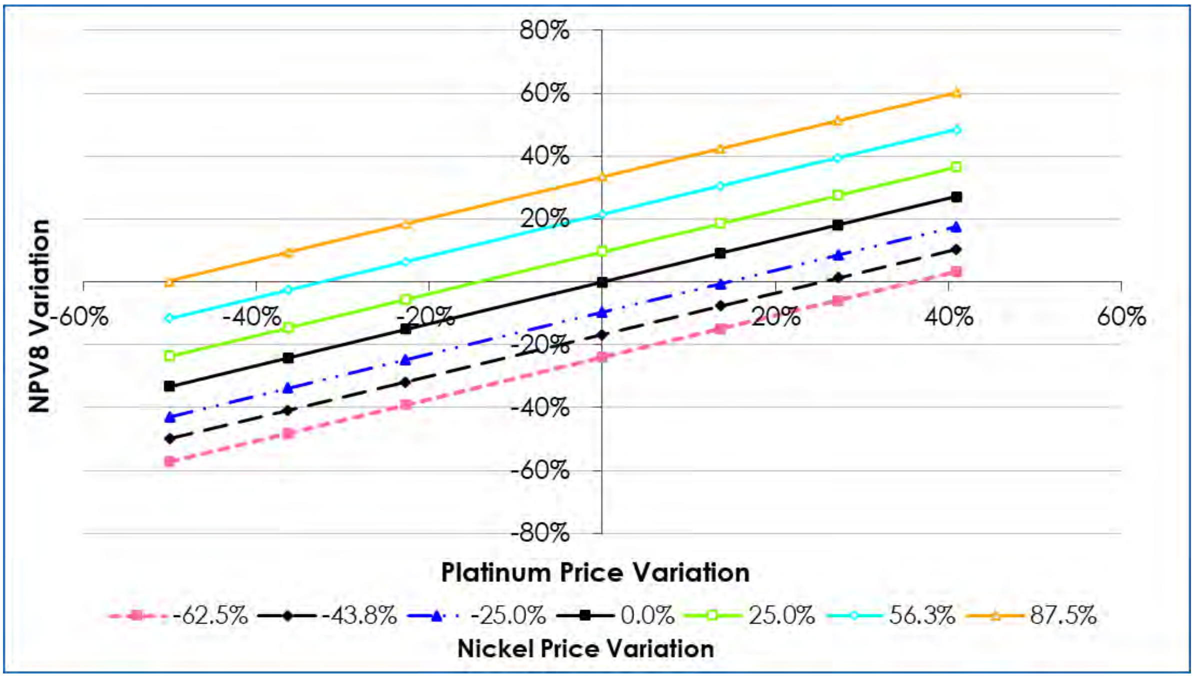

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Figure 22.10 Nickel and Platinum Price Sensitivity

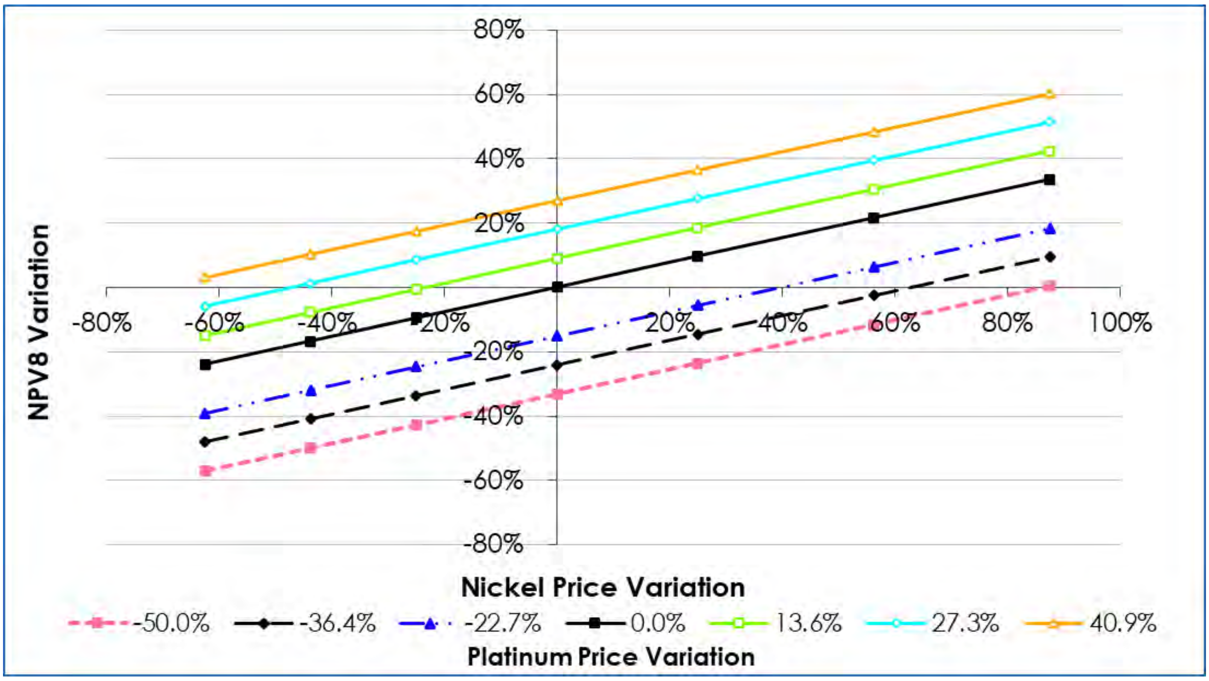

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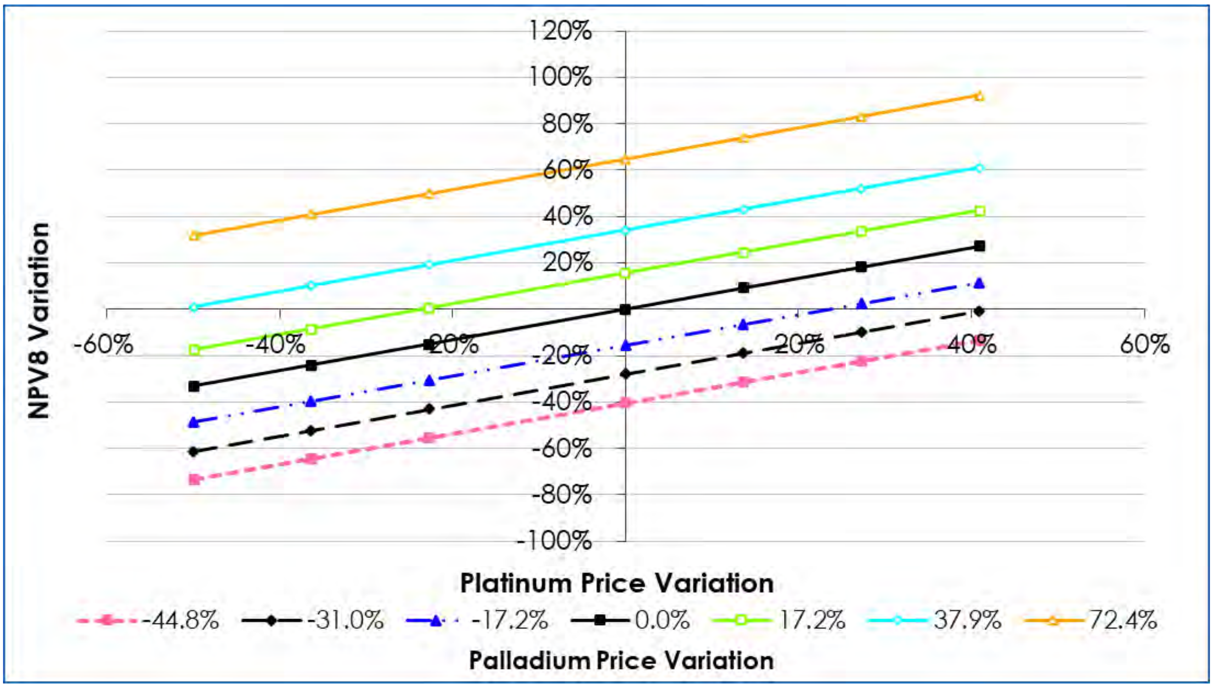
Table 22.19 After-Tax NPV8% Platinum and Palladium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Palladium Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
800	447	600	752	1,006	1,159	1,313	1,466
1,000	657	809	961	1,218	1,371	1,523	1,676
1,200	866	1,018	1,173	1,428	1,581	1,733	1,888
1,450	1,129	1,283	1,436	1,690	1,845	1,997	2,149
1,700	1,393	1,546	1,698	1,954	2,106	2,258	2,410
2,000	1,707	1,862	2,014	2,267	2,419	2,570	2,720
2,500	2,232	2,384	2,535	2,786	2,940	3,095	3,249

Table 22.20 After-Tax NPV8% Sensitivity to Platinum and Palladium Prices

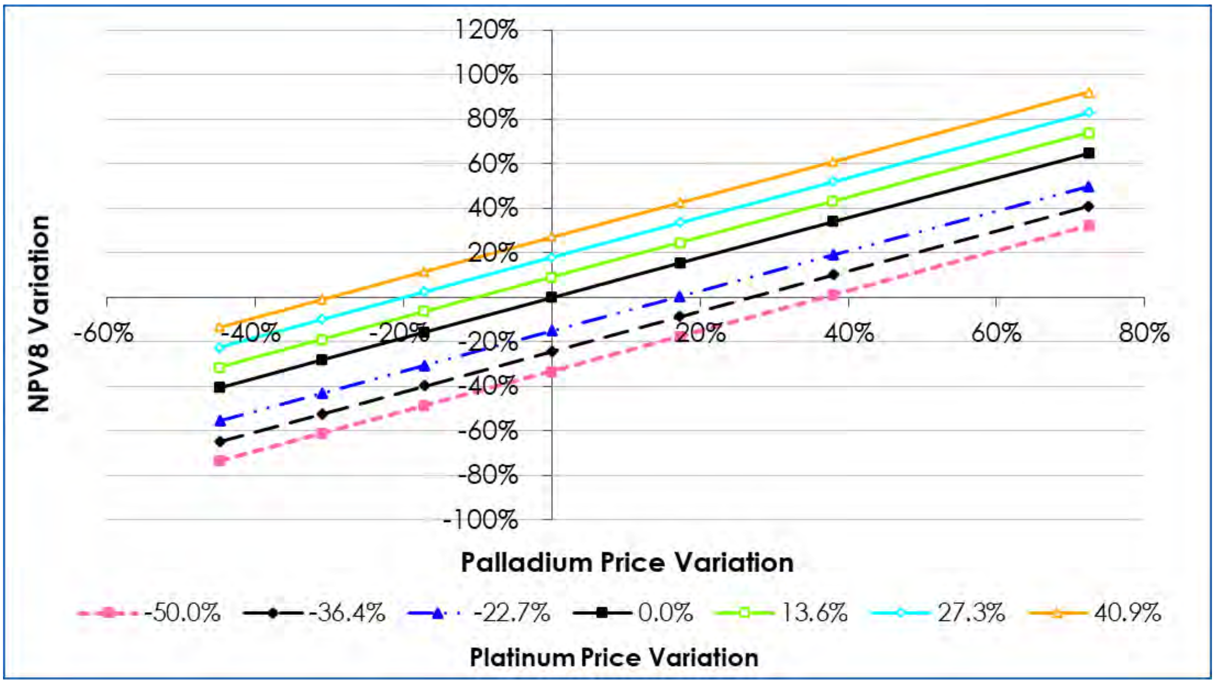
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Palladium Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
800	-74%	-65%	-56%	-41%	-31%	-22%	-13%
1,000	-61%	-52%	-43%	-28%	-19%	-10%	-1%
1,200	-49%	-40%	-31%	-16%	-6%	3%	12%
1,450	-33%	-24%	-15%	–	9%	18%	27%
1,700	-18%	-9%	0%	16%	25%	34%	43%
2,000	1%	10%	19%	34%	43%	52%	61%
2,500	32%	41%	50%	65%	74%	83%	92%

Figure 22.11 Platinum and Palladium Price Sensitivity



OreWin, 2022

Figure 22.12 Palladium and Platinum Price Sensitivity



OreWin, 2022

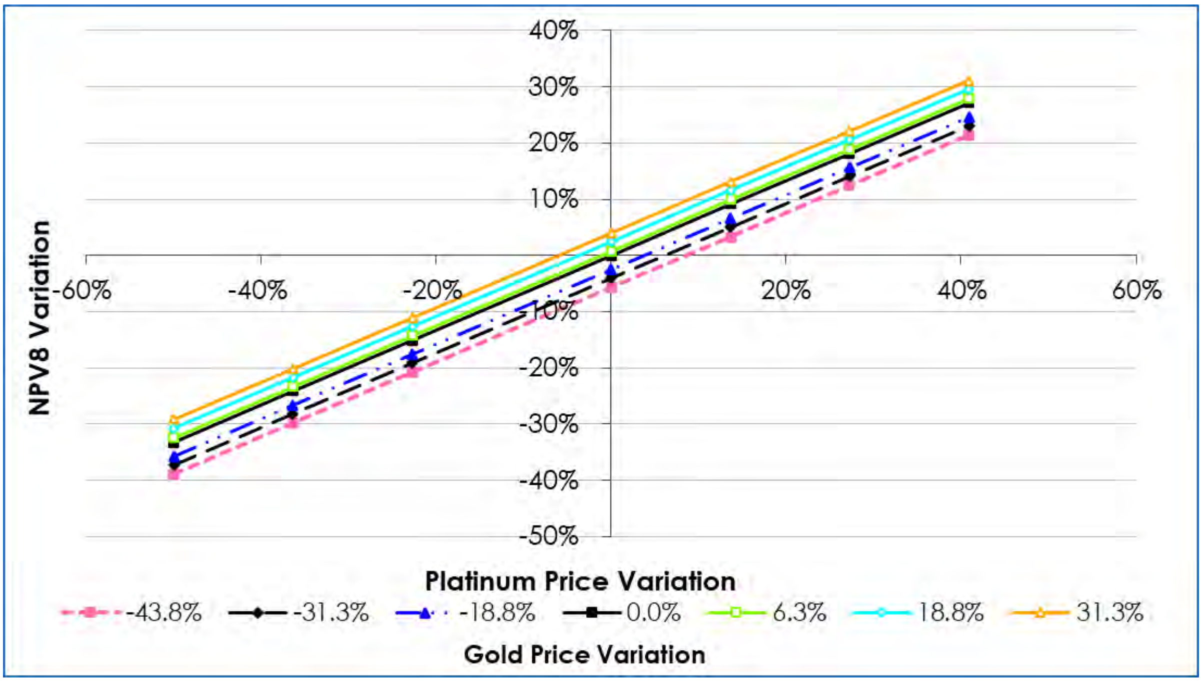
Table 22.21 After-Tax NPV8% Platinum and Gold Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Gold Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
900	1,032	1,187	1,339	1,595	1,747	1,901	2,053
1,100	1,059	1,214	1,367	1,622	1,775	1,929	2,080
1,300	1,087	1,242	1,394	1,649	1,803	1,956	2,107
1,600	1,129	1,283	1,436	1,690	1,845	1,997	2,149
1,700	1,143	1,296	1,449	1,704	1,859	2,010	2,163
1,900	1,171	1,324	1,477	1,731	1,886	2,038	2,190
2,100	1,199	1,351	1,504	1,759	1,913	2,065	2,217

Table 22.22 After-Tax NPV8% Sensitivity to Platinum and Gold Prices

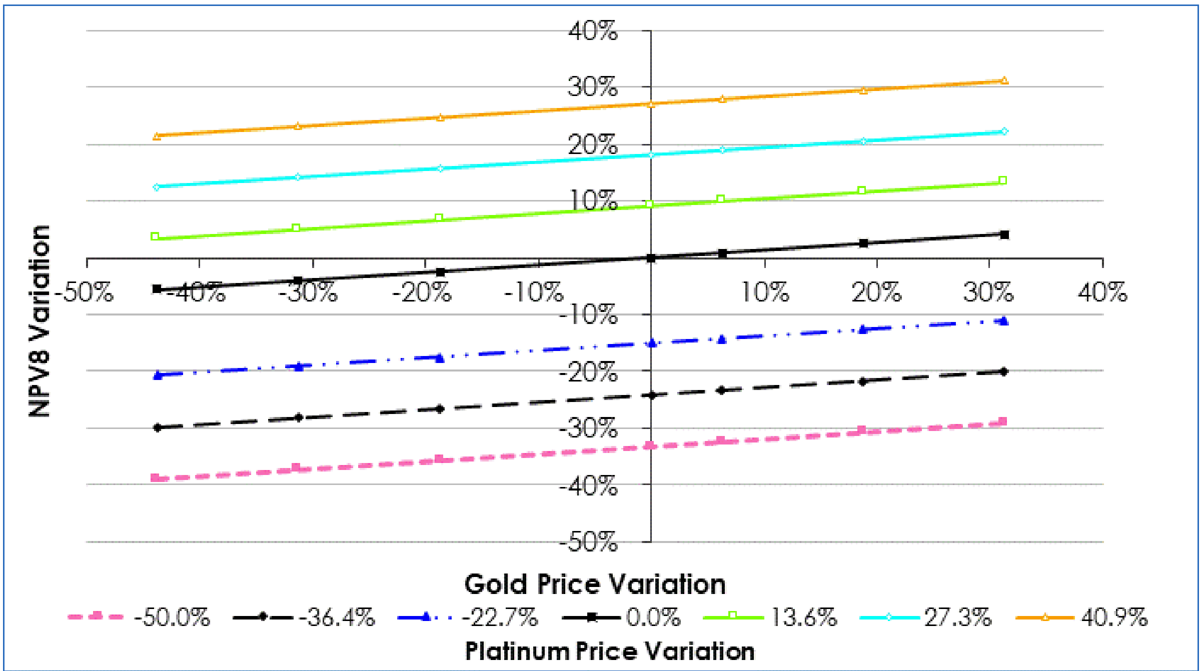
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Gold Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
900	-39%	-30%	-21%	-6%	3%	12%	21%
1,100	-37%	-28%	-19%	-4%	5%	14%	23%
1,300	-36%	-27%	-17%	-2%	7%	16%	25%
1,600	-33%	-24%	-15%	–	9%	18%	27%
1,700	-32%	-23%	-14%	1%	10%	19%	28%
1,900	-31%	-22%	-13%	2%	12%	21%	30%
2,100	-29%	-20%	-11%	4%	13%	22%	31%

Figure 22.13 Platinum and Gold Price Sensitivity



OreWin, 2022

Figure 22.14 Gold and Platinum Price Sensitivity



OreWin, 2022

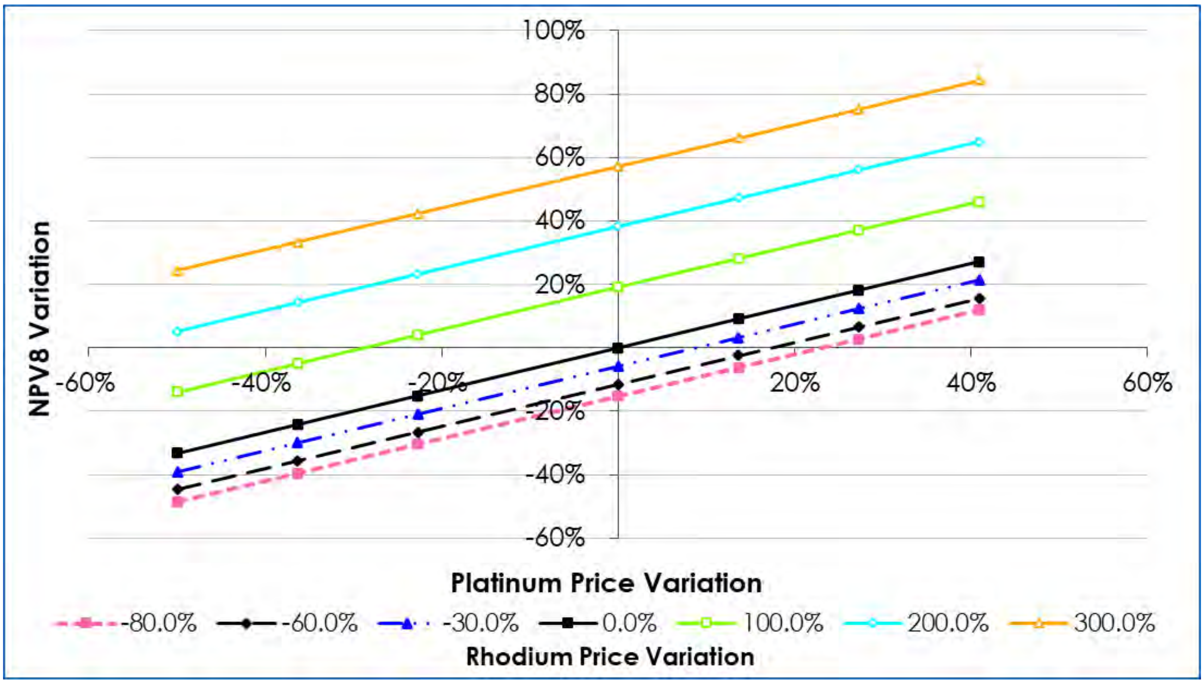
Table 22.23 After-Tax NPV8% Platinum and Rhodium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Rhodium Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
1,000	869	1,021	1,176	1,431	1,584	1,736	1,891
2,000	934	1,086	1,241	1,496	1,649	1,802	1,955
3,500	1,031	1,186	1,338	1,593	1,745	1,900	2,052
5,000	1,129	1,283	1,436	1,690	1,845	1,997	2,149
10,000	1,454	1,607	1,760	2,015	2,167	2,319	2,470
15,000	1,779	1,932	2,084	2,338	2,489	2,639	2,790
20,000	2,103	2,255	2,406	2,658	2,809	2,963	3,118

Table 22.24 After-Tax NPV8% Sensitivity to Platinum and Rhodium Prices

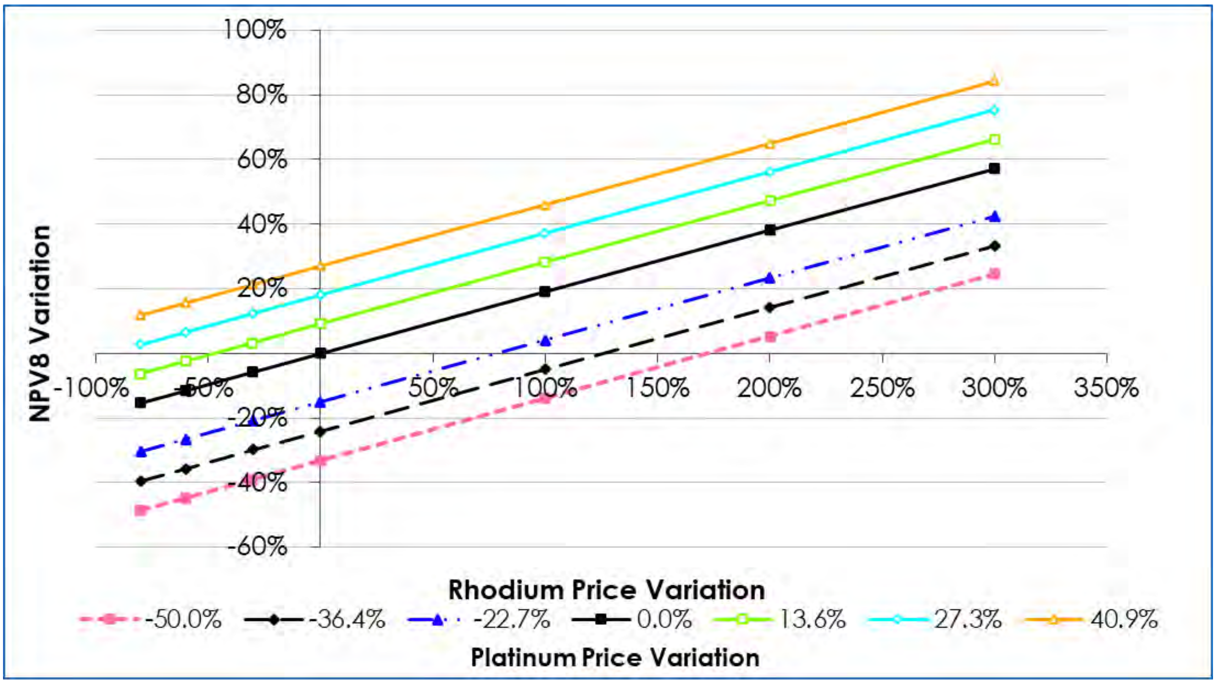
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Rhodium Price - $/oz	550	700	850	1,100	1,250	1,400	1,550
1,000	-49%	-40%	-30%	-15%	-6%	3%	12%
2,000	-45%	-36%	-27%	-11%	-2%	7%	16%
3,500	-39%	-30%	-21%	-6%	3%	12%	21%
5,000	-33%	-24%	-15%	–	9%	18%	27%
10,000	-14%	-5%	4%	19%	28%	37%	46%
15,000	5%	14%	23%	38%	47%	56%	65%
20,000	24%	33%	42%	57%	66%	75%	84%

Figure 22.15 Platinum and Rhodium Price Sensitivity



OreWin, 2022

Figure 22.16 Rhodium and Platinum Price Sensitivity



OreWin, 2022

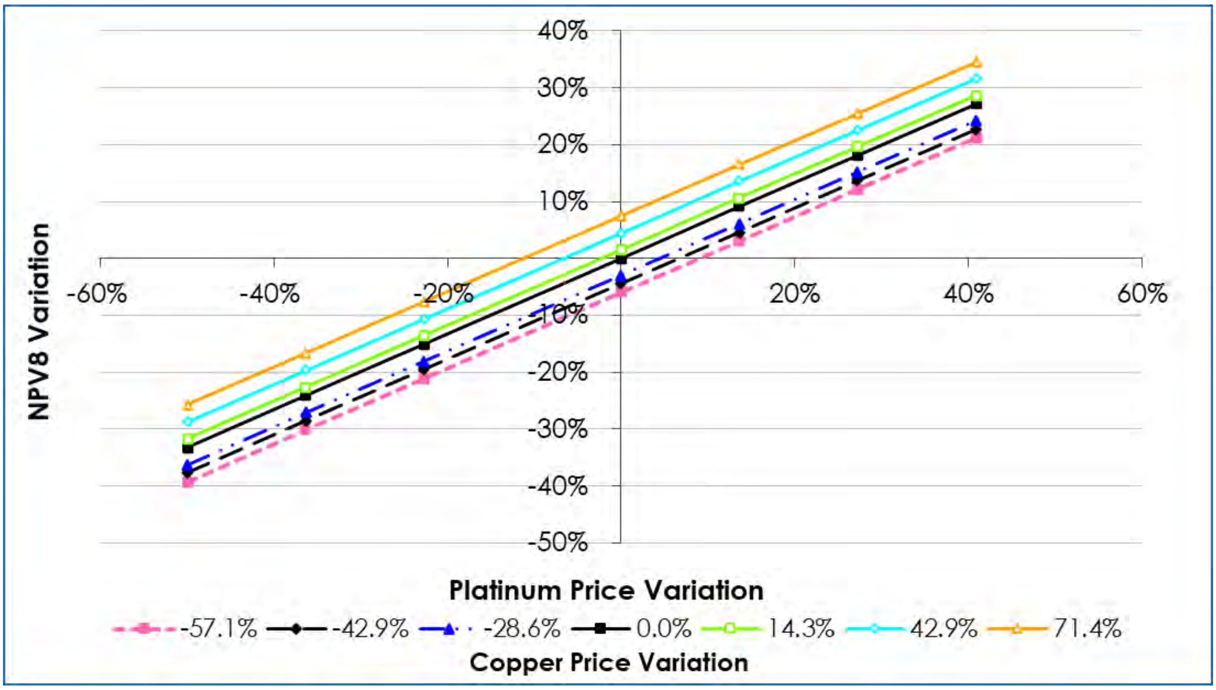
Table 22.25 After-Tax NPV8% Platinum and Copper Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Copper Price - $/lb	550	700	850	1,100	1,250	1,400	1,550
1.50	1,027	1,182	1,335	1,590	1,742	1,897	2,048
2.00	1,053	1,207	1,360	1,615	1,768	1,922	2,073
2.50	1,078	1,232	1,385	1,640	1,793	1,947	2,098
3.50	1,129	1,283	1,436	1,690	1,845	1,997	2,149
4.00	1,154	1,308	1,461	1,715	1,870	2,022	2,174
5.00	1,205	1,358	1,511	1,766	1,920	2,072	2,224
6.00	1,256	1,409	1,562	1,817	1,970	2,122	2,274

Table 22.26 After-Tax NPV8% Sensitivity to Platinum and Copper Prices

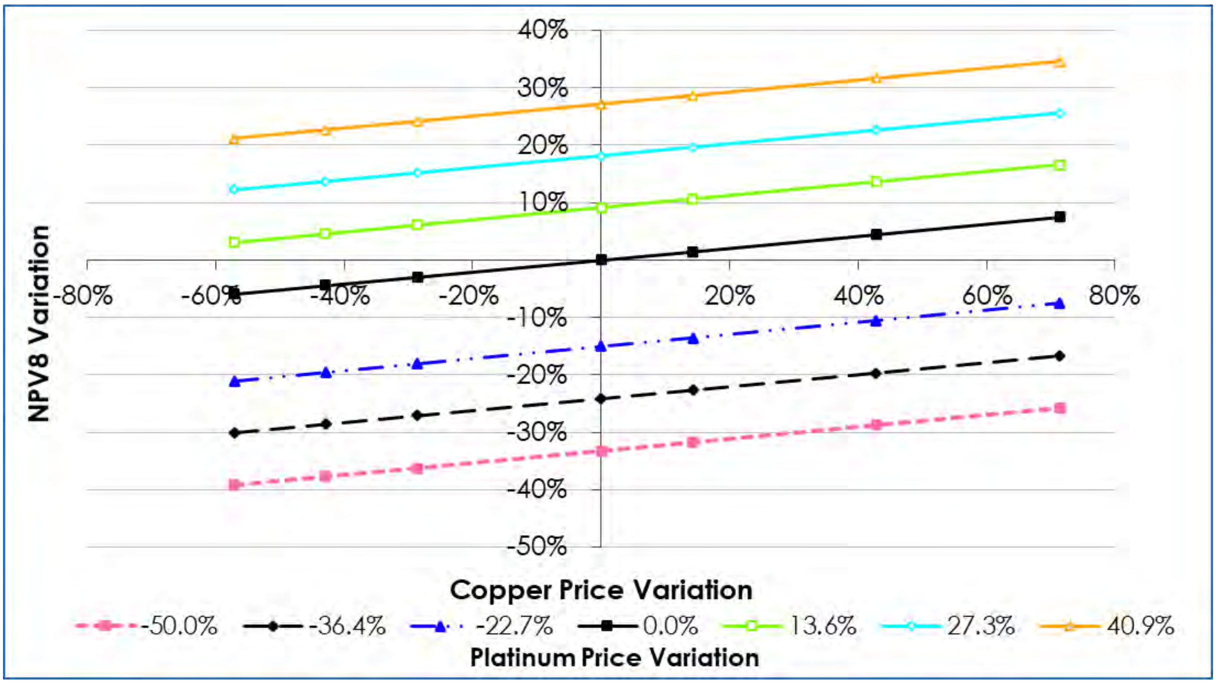
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Copper Price - $/lb	550	700	850	1,100	1,250	1,400	1,550
1.50	-39%	-30%	-21%	-6%	3%	12%	21%
2.00	-38%	-29%	-20%	-4%	5%	14%	23%
2.50	-36%	-27%	-18%	-3%	6%	15%	24%
3.50	-33%	-24%	-15%	–	9%	18%	27%
4.00	-32%	-23%	-14%	1%	11%	20%	29%
5.00	-29%	-20%	-11%	4%	14%	23%	32%
6.00	-26%	-17%	-8%	8%	17%	26%	35%

Figure 22.17 Platinum and Copper Price Sensitivity



OreWin, 2022

Figure 22.18 Copper and Platinum Price Sensitivity



OreWin, 2022

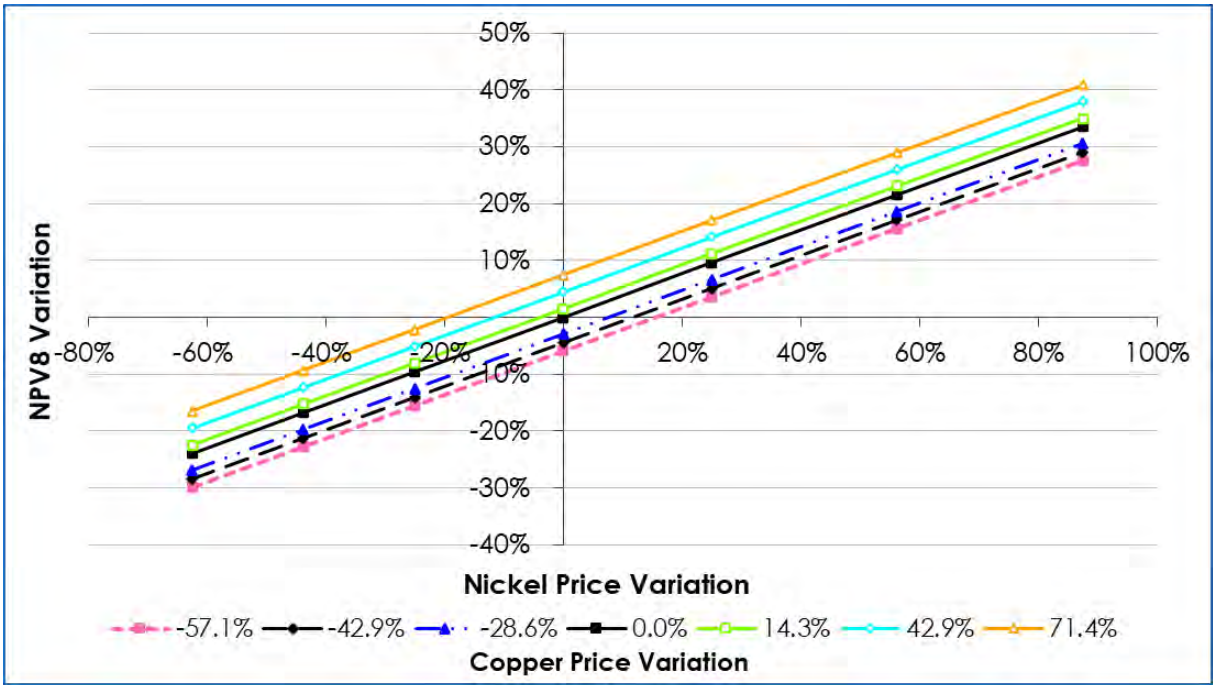
Table 22.27 After-Tax NPV8% Nickel and Copper Price Sensitivity

After-Tax NPV8% ($M)	Nickel Price ($/lb)						
Copper Price - $/lb	3.00	4.50	6.00	8.00	10.00	12.50	15.00
1.50	1,185	1,306	1,428	1,590	1,751	1,955	2,156
2.00	1,210	1,332	1,453	1,615	1,777	1,980	2,181
2.50	1,235	1,357	1,478	1,640	1,802	2,005	2,206
3.50	1,286	1,407	1,529	1,690	1,854	2,055	2,256
4.00	1,311	1,433	1,554	1,715	1,879	2,080	2,281
5.00	1,361	1,483	1,604	1,766	1,929	2,130	2,331
6.00	1,412	1,533	1,654	1,817	1,979	2,180	2,381

Table 22.28 After-Tax NPV8% Sensitivity to Nickel and Copper Prices

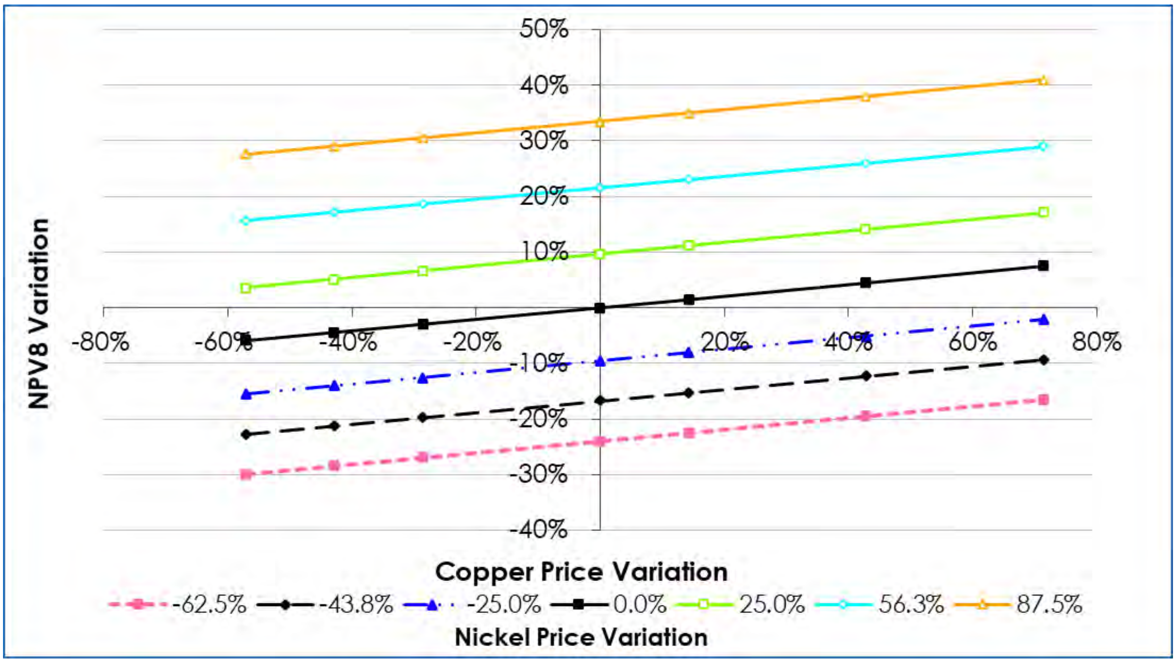
After-Tax NPV8% ($M)	Nickel Price ($/lb)						
Copper Price - $/lb	3.00	4.50	6.00	8.00	10.00	12.50	15.00
1.50	-30%	-23%	-16%	-6%	4%	16%	28%
2.00	-28%	-21%	-14%	-4%	5%	17%	29%
2.50	-27%	-20%	-13%	-3%	7%	19%	31%
3.50	-24%	-17%	-10%	–	10%	22%	34%
4.00	-22%	-15%	-8%	1%	11%	23%	35%
5.00	-19%	-12%	-5%	4%	14%	26%	38%
6.00	-16%	-9%	-2%	8%	17%	29%	41%

Figure 22.19 Nickel and Copper Price Sensitivity



OreWin, 2022

Figure 22.20 Copper and Nickel Price Sensitivity



OreWin, 2022

22.7 Cost Sensitivity Analysis

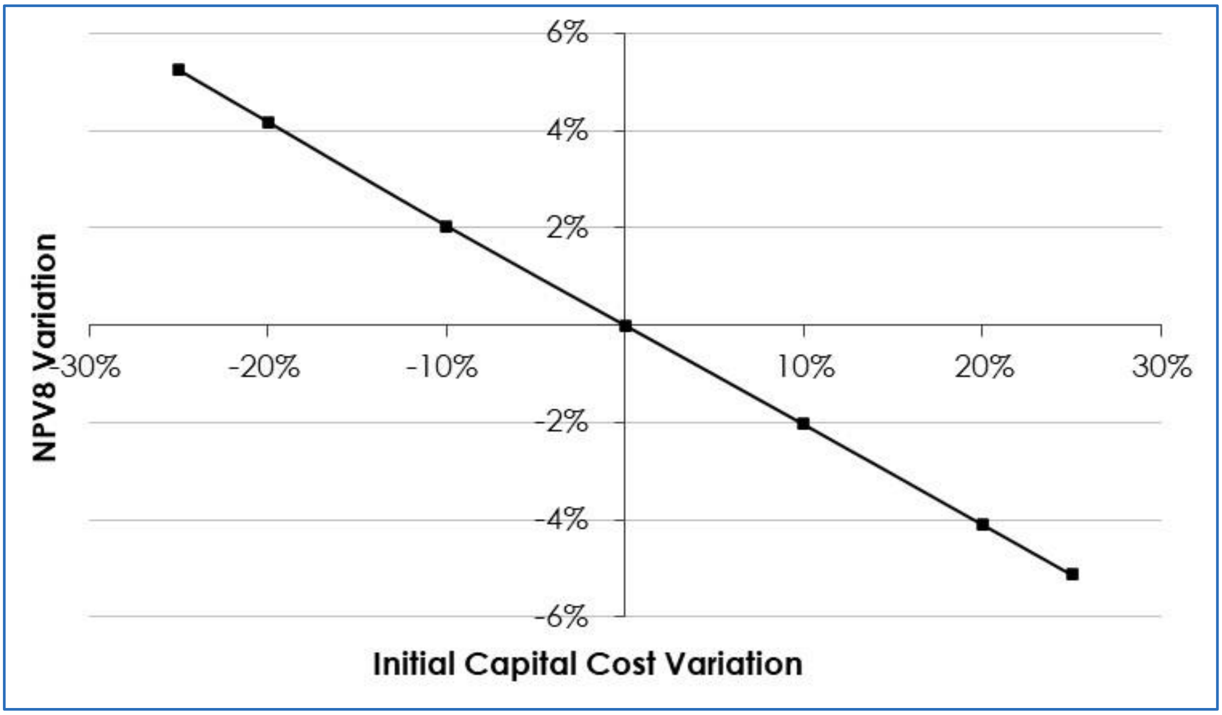
Cost sensitivity analyses of the After-Tax NPV8% were examined for the initial capital, expansion capital, sustaining capital, and operating costs.

The initial capital cost sensitivity analysis shows the After-Tax NPV8% of $1,779M at a 25% decrease in the initial capital cost and the After-Tax NPV8% of $1,604M at a 25% increase in the pre-production capital cost. The results of the pre-production capital cost sensitivity analysis are shown in Table 22.29 and Figure 22.21.

Table 22.29 Initial Capital Cost Sensitivity

Pre-Production Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	1,779	1,761	1,725	1,690	1,656	1,621	1,604
Difference ($M)	89	71	35	–	-34	-69	-87
Difference (%)	5%	4%	2%	0%	-2%	-4%	-5%

Figure 22.21 Initial Capital Cost Sensitivity



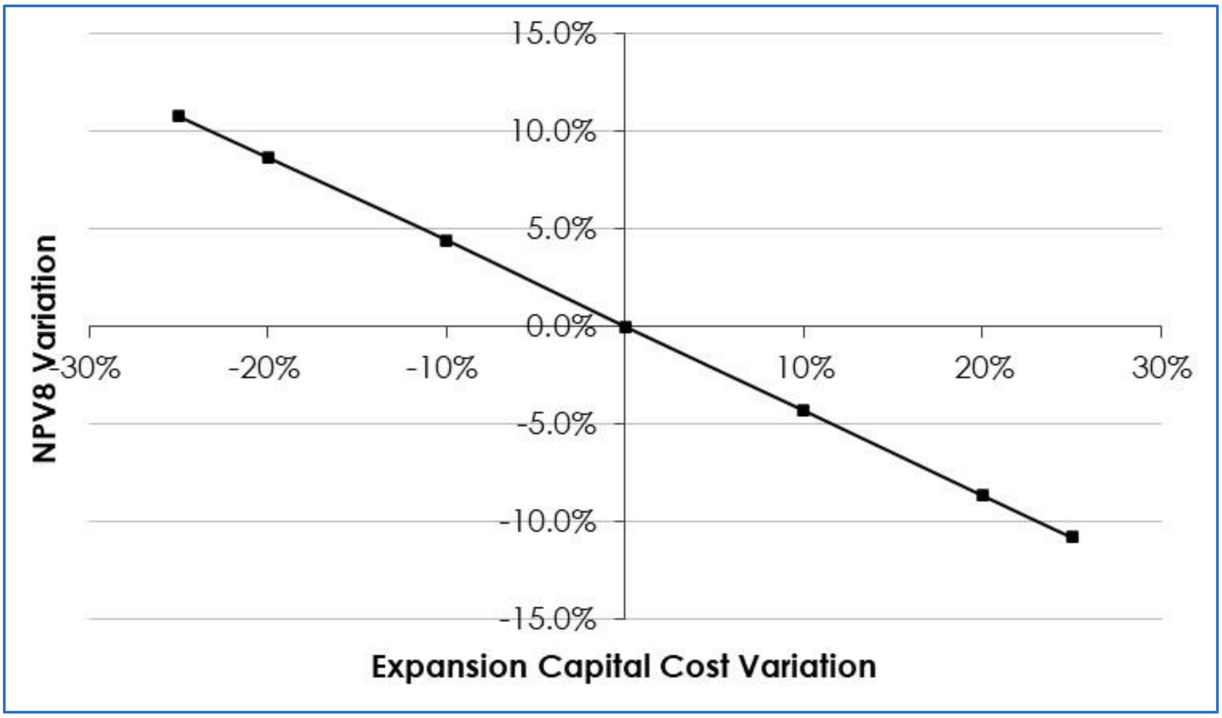
OreWin, 2021

The expansion capital cost sensitivity shows the After-Tax NPV8% of $1,872M at a 25% decrease in the expansion capital cost and the After-Tax NPV8% of $1,508M at a 25% increase in the expansion capital cost. The results of the expansion capital cost sensitivity analysis are shown in Table 22.30 and Figure 22.22.

Table 22.30 Expansion Capital Cost Sensitivity

Expansion Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	1,872	1,836	1,765	1,690	1,618	1,544	1,508
Difference ($M)	182	146	75	–	-73	-146	-182
Difference (%)	10.7%	8.6%	4.4%	0.0%	-4.3%	-8.6%	-10.8%

Figure 22.22 Expansion Capital Cost Sensitivity



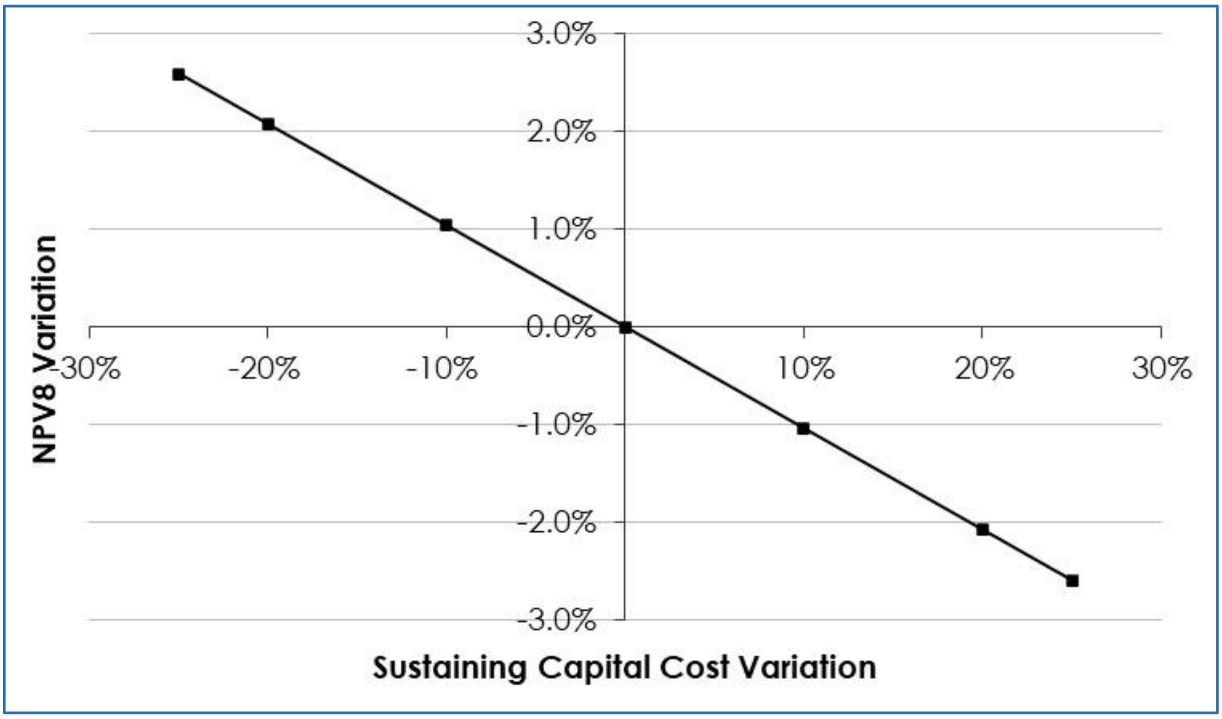
OreWin, 2021

The sustaining capital cost sensitivity analysis shows the After-Tax NPV8% of $1,734M at a 25% decrease in the sustaining capital cost and the After-Tax NPV8% of $1,646M at a 25% increase in the sustaining capital cost. The results of the sustaining capital cost sensitivity analysis are shown in Table 22.31 and Figure 22.23.

Table 22.31 Sustaining Capital Cost Sensitivity

Sustaining Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	1,734	1,725	1,708	1,690	1,673	1,655	1,646
Difference ($M)	44	35	17	–	-17	-35	-44
Difference (%)	2.6%	2.1%	1.0%	0.0%	-1.0%	-2.1%	-2.6%

Figure 22.23 Sustaining Capital Cost Sensitivity



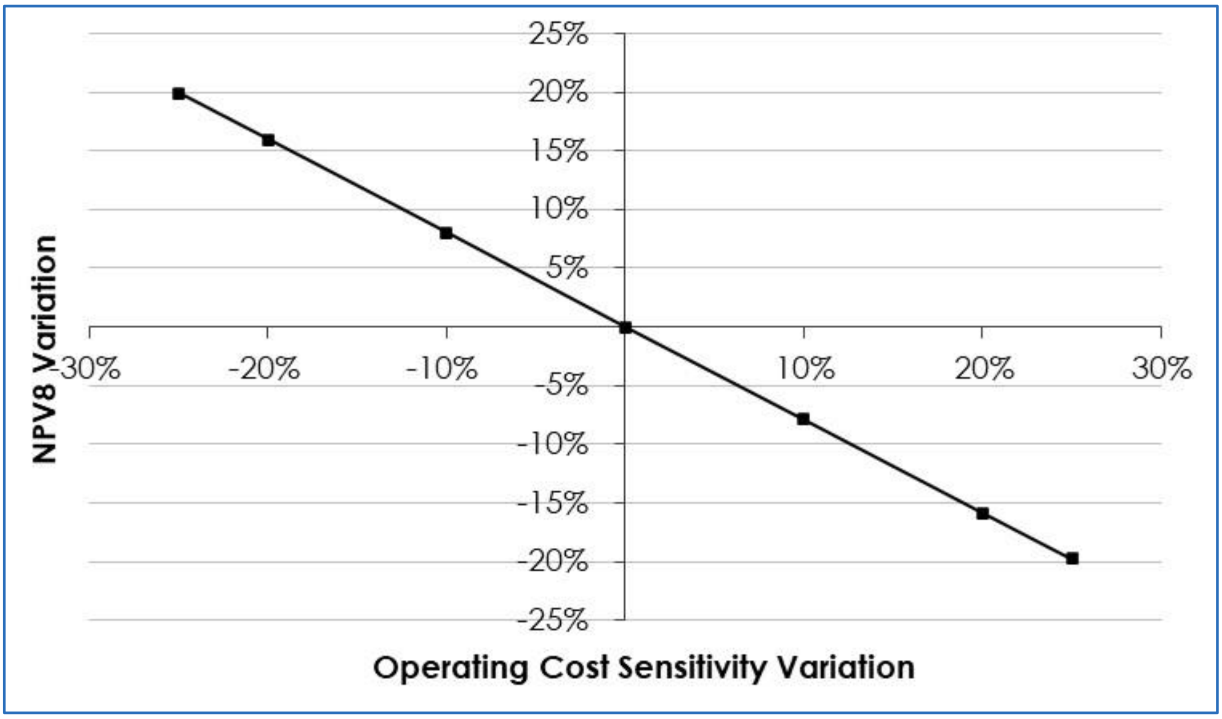
OreWin, 2021

The operating cost sensitivity analysis shows the After-Tax NPV8% of $2,026M at a 25% decrease in the operating cost and the After-Tax NPV8% of $1,356M at a 25% increase in the operating cost. The results of the operating cost sensitivity analysis are shown in Table 22.32 and Figure 22.24.

Table 22.32 Operating Cost Sensitivity

Sustaining Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	2,026	1,960	1,826	1,690	1,557	1,423	1,356
Difference ($M)	336	270	136	–	-133	-267	-334
Difference (%)	20%	16%	8%	0%	-8%	-16%	-20%

Figure 22.24 Operating Cost Sensitivity



OreWin, 2021

22.8 Exchange Rate Sensitivity Analysis

Exchange rate sensitivities to US dollars analysis of the After-Tax NPV8% were performed on the financial model. The exchange rate sensitivity analysis shows the After-Tax NPV8% is most sensitive to the ZAR. After-Tax NPV8% at a 25% decrease in ZAR exchange rate (12 ZAR/USD) is $875M and the After-Tax NPV8% at a 25% increase in ZAR exchange rate (20 ZAR/USD) is $2,178M. The results of the exchange rate sensitivity analysis are shown in Table 22.33.

Table 22.33 Exchange Rate Sensitivity

After-Tax NPV8% ($M)	Exchange Rate (+ / - %)						
Currency	-25%	-20%	-10%	–	10%	20%	25%
ZAR	875	1,077	1,418	1,690	1,914	2,098	2,178
EUR	1,674	1,678	1,685	1,690	1,694	1,698	1,700
AUD	1,685	1,686	1,688	1,690	1,692	1,693	1,693
CNY	1,676	1,679	1,685	1,690	1,694	1,697	1,699
JPY	1,690	1,690	1,690	1,690	1,690	1,690	1,690

22.9 Financial Results at Current Spot Metal Prices

At current spot metal prices, the Platreef 2022 FS shows an after-tax NPV8% of $4,116M and IRR of 29.35%. Financial results at base consensus (base case) and current spot metal prices are summarised in Table 22.34.

Table 22.34 Platreef 2022 FS Financial Results at Base Case and Current Spot Metal Prices

	Discount Rate	Long Term Prices[1]	Spot Prices[2]
Net Present Value ($M)	Undiscounted	8,543	17,130
	5.0%	3,098	6,815
	8.0%	1,690	4,116
	10.0%	1,104	2,979
	12.0%	692	2,169
Internal rate of return (IRR)		18.48%	29.35%
Project Payback Period (Years)		7.93	6.4

[1]Long Term metal price assumptions are as follows: $1,100/oz. platinum, $1,450/oz. palladium, $1,600/oz. gold, $5,000/oz. rhodium, $8.00/lb nickel and $3.50/lb copper.
[2]Spot metal prices (23 February 2022) are as follows: $1,095/oz. platinum, $2,480/oz. palladium, $1,909/oz. gold, $18,750/oz. rhodium, $11.31/lb nickel and $4.48/lb copper.

22.10 Platreef 2022 FS Comparison to Platreef 2017 FS

A comparison of the Platreef 2022 FS and Platreef 2017 FS financial models was carried out. The Platreef 2022 FS estimates of cash flows were prepared on a real basis as at 1 January 2022 to calculate NPV and the Platreef 2017 FS estimates cash flows were prepared on a real basis as at 1 January 2017. The after-tax financial results in 2022 FS and 2017 FS is shown in Table 22.35.

Table 22.35 After-Tax Financial Results Comparison

	Discount Rate	2022 FS	2017 FS
Net Present Value ($M)	Undiscounted	8,543	6,471
	5.0%	3,098	1,961
	8.0%	1,690	916
	10.0%	1,104	500
	12.0%	692	217
	15.0%	283	–57
	20.0%	-83	–291
Internal Rate of Return	–	18.48%	14.2%
Project Payback Period (Years)	–	7.9	5.3

The 2021 FS and 2017 FS metal prices are based on a review of consensus price forecasts from a financial institutions and similar studies. A comparison of the base and precious metal prices in 2022 FS and 2017 FS is presented in Table 22.36.

Table 22.36 Metal Prices Comparison

Parameter	Unit	2022 FS	2017 FS
Platinum	$/oz	1,100	1,250
Palladium	$/oz	1,450	825
Gold	$/oz	1,600	1,300
Rhodium	$/oz	5,000	1,000
Copper	$/lb	3.50	3.00
Nickel	$/lb	8.00	7.60

The key production summary comparison is shown in Table 22.37. A comparison of the Pre-Production Capital Costs, Sustaining Capital Costs and Total Capital Costs in 2021 FS and 2017 FS are shown in Table 22.38.

Table 22.37 Production Summary Comparison

Item	Unit	2022 FS	2017 FS
Mined and Processed	Mt	125	125
Platinum	g/t	1.94	1.95
Palladium	g/t	1.99	2.01
Gold	g/t	0.30	0.30
Rhodium	g/t	0.13	0.14
3PE+Au	g/t	4.37	4.40
Copper	% Cu	0.16	0.17
Nickel	% Ni	0.34	0.34
Concentrator Recoveries			
Platinum	%	87.2	87.4
Palladium	%	86.8	86.9
Gold	%	78.5	78.6
Rhodium	%	80.3	80.5
Copper	%	87.7	87.9
Nickel	%	71.6	71.9
Concentrate Produced	kt (dry)	5,545	5,568
Platinum	g/t	38.21	38.24
Palladium	g/t	39.00	39.07
Gold	g/t	5.32	5.32
Rhodium	g/t	2.43	2.43
3PE+Au	g/t	84.97	85.07
Copper	% Cu	3.26	3.27
Nickel	% Ni	5.44	5.46
Recovered Metal			
Platinum	koz	6,813	6,846
Palladium	koz	6,954	6,994
Gold	koz	948	952
Rhodium	koz	433	436
3PE+Au	koz	15,149	15,228
Copper	Mlb	399	402
Nickel	Mlb	665	670

Table 22.38 Total Capital Costs Comparison

Item	Unit	2022 FS	2017 FS
Mining			
Geology	$M	72	20
Mining	$M	1,744	1,129
Capitalised Operating Costs	$M	–	63
Subtotal	$M	1,816	1,213
Concentrator and Tailings			
Concentrator	$M	349	246
Capitalised Operating Costs	$M	–	0.05
Subtotal	$M	349	246
Infrastructure			
Infrastructure	$M	363	253
Site Costs	$M	7	11
Capitalised Operating Costs	$M	-	36
Subtotal	$M	371	300
Owners Cost			
Owners Cost	$M	222	52
Closure Cost	$M	11	17
Subtotal	$M	233	69
Capex Before Contingency	$M	2,768	1,827
Contingency	$M	134	135
Capex After Contingency	$M	2,902	1,962

23 ADJACENT PROPERTIES

This Section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This Section not used.

25 INTERPRETATION AND CONCLUSIONS

25.1 Platreef 2022 FS

The Platreef 2022 FS will provide the technical basis for Ivanplats to continue the project financing and to continue marketing negotiations that have been undertaken to date. Ivanplats should continue to prepare for the execution activities and to update the long-term development plans for Platreef. Continued development of Shaft 1 will progress the project and this can be used for further defining the execution plans.

25.2 Geology and Mineral Resources

The Platreef comprises a variably layered, composite norite–pyroxenite–harzburgite intrusion that lies at the base of the Northern Limb of the Bushveld Complex, in contact with metasedimentary and granitic floor rocks. The variability of lithology and thickness along strike is attributed to underlying structures and assimilation with local country rocks.

Five major cyclic units have been recognised which correlate well with the UCZ rock sequence described for the main Bushveld Complex.

The TCU is laterally continuous across large parts of the Platreef Project area. Mineralisation in the TCU shows generally good continuity and is mostly confined to pegmatoidal orthopyroxenite and harzburgite.

Pyrrhotite, pentlandite and chalcopyrite occur as interstitial sulfides in the TCU lithologies. Platinum group minerals are mainly present as PGE–sulfides, PGE–BiTe and PGE–As alloys, that are fine-grained (<10 µm) and may occur within base metal sulfides, on their rims, or encapsulated in silicates.

Mr Kuhl is of the opinion that knowledge of the deposit settings, lithologies, mineralisation style and setting, and structural and alteration controls on mineralisation within the AMK, ATS, and UMT deposits are sufficient to support Mineral Resource estimation. For the Platreef 2017 FS only the underground-mineable UMT deposit is considered. The mineralisation delineated at the Turfspruit 241 KR, Macalacaskop 243 KR, and Rietfontein 2 KS farms is typical of Platreef-style mineralisation within the Northern Limb of the Bushveld Complex. Exploration programmes developed using the Merensky-reef analogue are appropriate to the deposit style.

The Mineral Resources are limited to areas that have been sufficiently drilled to support geological interpretation and grade estimation. There is approximately 9.4 km^2 of exploration targets and an additional 48 km^2 of prospective ground on the Turfspruit and Macalacaskop farms to the southwest of the Mineral Resources.

25.2.1 Drilling, Sampling and Data Verification

The database (closed 24 July 2015) includes 578 drillholes (196,213 m) from Phase 1 (including all redrills and deflections). The Phase 1 drilling was completed in support of open-pit resources (See Section 6).

The database also includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totalling 501,638 m completed by 11 February 2015. Depths for deflections are calculated based on point of defection and do not include the mother or pilot hole portion. This includes 33 drillholes and deflections (9,181 m) completed for geotechnical purposes and 62 drillholes and deflections (23,001 m) completed for metallurgical sampling purposes.

Standardised geological core logging conventions were used to capture information from drill core. Collar surveys were conducted by a licensed land surveyor on all completed holes. The majority of drillholes have been down-hole surveyed. Recovery data indicate a substantial decrease within faulted and sheared zones.

Sample preparation and analyses were performed by accredited independent laboratories and have followed similar protocols since 2001. The preparation and analytical procedures are in line with industry-standard methods for PGE–Au–Ni–Cu deposits. Drill programmes included insertion of blank, duplicate and SRM/CRM samples.

The QA/QC programme results do not indicate any problems with the analytical programmes that would preclude use of the data.

Sample security has been demonstrated by the fact that the samples were always attended or locked in the on-site sample preparation facility.

Mr Kuhl is of the opinion that the data collection procedures and QA/QC control are acceptable to support Mineral Resource estimation.

The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programmes are sufficient to support Mineral Resource estimation.

The sample preparation, sample analyses, data entry and security have been done to industry-standards for large exploration and development projects.

The quality of the Pt, Pd, Au, Rh, Cu and Ni analytical data are sufficiently reliable to support Mineral Resource estimation.

25.2.2 Mineral Resource Estimates

Mr Kuhl is of the opinion that the Mineral Resources for the Platreef Project, which have been estimated using core-drill data, have been performed to industry best practices (CIM, 2019), and conform to the requirements of the 2014 CIM Definition Standards.

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction including:
  - Long-term commodity price assumptions.
  - Long-term exchange rate assumptions.

  - Assumed mining method.
  - Availability of water and power.
  - Operating and capital cost assumptions.
  - Metal recovery assumptions.
  - Concentrate grade and smelting and refining terms.
- Additional mini pilot plant testwork is planned. This work may result in changes to the metallurgical recoveries and smelter payables assumptions used to evaluate reasonable prospects of eventual economic extraction.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. The current practice of using grade shells in the area drilled in detail may underestimate the variability of the grades within and in the vicinity of the T1MZ and the T2MZ, and any stope boundaries that are laid out along the 2PE+Au grade shell surface will likely not, in practice, be able to follow the exact actual surface. The consequence would be that the effects of contact dilution and ore loss could be more than is currently projected. The data on positions of grade shell boundaries should be examined to the extent possible to estimate their short-scale variability; the likely accuracies of down-hole surveys should be taken into account, and it is recognised a definitive answer may have to await exposures in underground workings.

25.3 Mineral Reserve Estimate

The mineral reserve estimate for Platreef was based on proven methods, mining practices, and modelling techniques applied to a well-defined resource block model. The cost assumptions assigned are reasonable and support the cut-off grades for use in defining the reserve model and supporting mine plan. Based on this assessment, the Platreef Probable Mineral Reserve will support the Platreef 2022 FS Phased Development Plan with 125 Mt ore production for the life of the mine.

25.4 Mining Risks and Opportunities

25.4.1 Risks

The following is a list of potential risks for Platreef:

- Production and schedule constraints due to shaft pillar designs.
- Possible requirement of replacement ore and waste passes; the design provides for locations but they are not currently in the design.
- Uncertainty of raise boring to depths of 750 m and deeper in this location.
- The amount of lining required in the ventilation raises once they are reamed.
- Mining through the Tshukudu Fault.
- Timeliness of the definition drilling during the preproduction and early production stages.
- Cooling capacity during the summer peaks. This can be managed by a work rest regime and supplemental underground air conditioning units.

- Mining underneath cemented paste fill or rock fill in the sill pillars (although less than 1% of the tonnage is mined under a sill pillar).
- The amount of additional S3 support that may be required in the later years of the mine life if ground conditions are worse and mining impacts are more severe.
- Difficulty handling the low-grade material during development.
- The stability of the accesses in the secondary stopes during and after the mining of the primary stopes.

25.4.2 Opportunities and Recommendations

The following is a list of potential opportunities Platreef may experience:

- In the current design, Shaft 2 has a 6.19 Mtpa capacity. This, combined with the conversion of Shaft 1 to a production shaft with a 2.5 Mtpa capacity, may present an opportunity to achieve the anticipated production rate of 8 Mtpa in Phase 2 as described in the Platreef 2014 PEA (March 2014).
- Reducing the angle of the footwall and hanging wall in production stopes once the definition drilling programme better defines the block model.
- Automation of production LHDs and trucks.
- Remote operation of fixed rock breakers at ore and waste truck dumps.
- Potential to incorporate electric-powered mobile equipment in an effort to reduce ventilation and associated refrigeration requirements.
- Further analysis of equipment utilisation to reduce fleet size.
- Reducing the number of individuals supporting the development crews.
- Opportunity to use production ore passes as waste passes during the development phases.
- Storing and hoisting development ore during the Shaft 1 hoisting phase.
- Optimisation of the mine air cooling system.
- Further definition and delineation of FW Mineral Resources.

25.4.3 Mining Conclusions

The mine plan and expenditure schedule presented herein is reasonable. The plan is based on the currently available Platreef data and established mining practice. The resource model and geotechnical parameters provided appear reasonable and are a sound basis for the design of a large-scale and highly mechanised underground mine at a feasibility-level of confidence.

The proposed plan uses well-established mining technology. No unproven equipment or methods are contained in the plan; however, there is potential to take advantage of currently available and future technology gains.

25.5 Metallurgy

It is the opinion of the qualified person responsible for the metallurgical aspects of the Platreef 2022 FS, Mr Val Coetzee, that an acceptable metallurgical testwork programme was conducted on the samples provided. The range of samples tested appears to span the limits of the mineralised material from a grade perspective and includes the main domains identified by the geological team Sufficient; variability testwork has been conducted to delineate variability of recovery response and cost to grade, domain and spatial location in the deposit of the material to be processed.

Detailed mineralogical analysis of the selected ores have contributed to the understanding of the mode of occurrence and liberation characteristics of the valuable minerals.

The testwork programmes have been conducted by parties well versed in the processing of ores from the Bushveld Igneous Complex. The necessary checks and balances have been applied to ensure that the testwork and chemical analysis has been conducted with the necessary diligence and accuracy.

The proposed circuit is considered to be the preferred option for the concentrator. The use of a multi-stage crusher circuit followed by a single stage milling circuit is considered to be the option of least risk to the project and is recommended for this stage of the study. Preliminary assessments have indicated that the inclusion of an HPGR circuit as an alternative to the tertiary crushing circuit for the Phase 2 4.4 Mtpa concentrator could potentially provide additional operating cost saving opportunities.

A mini pilot plant campaign was conducted, primarily, to produce bulk concentrate samples for downstream hydrometallurgical refining test work and concentrate de-watering test work. The added objective of deriving additional design data from the pilot runs was only partially achieved due to of operational challenges at Mintek. These runs are thus considered to reflect preliminary commissioning results. These commissioning runs, successfully, allowed for generation of concentrate samples for Kell test work and concentrate de-watering test work but did not provide sufficient data to fully confirm the comprehensive design and metallurgical performance data. The locked cycle test results as derived during the 2017 FS are considered adequate for deriving metallurgical performance projections.

The proposed flotation circuit is based on interpretation of the results obtained from the bench-scale flotation testwork. The design and specification of the various flotation stages is considered adequate for this the level study and provides sufficient flexibility required during commercial production to optimise outputs.

25.6 Infrastructure

A number of mining projects are in the development phase on the Bushveld Igneous Complex that all require water, power and road access. This will place significant strain on the existing infrastructure, as well as further pressure on the approval and/or completion of major infrastructure projects.

The Platreef Project team has addressed the supply-demand requirements of bulk power and water to a sufficient level of detail for this study. Bulk water availability seems to be sufficient based on the level of accuracy of the study performed, however the timing when water will be available remains of concern, and Ivanplats should monitor progress of the development of the infrastructure feeding the waste water treatment works, that the mine draw water from. Consideration of drawing water from an alternative bulk water supply should be considered for redundancy.

The design of the overhead line is adequate for the transmission of the required power to the mine, and agreement has been secured from Eskom to supply the requisite amount of energy. Eskom should be kept apprised of any changes to the load build up to prevent incurring unnecessary costs.

It can be concluded that the availability of skilled labour resources, for both construction and operational phases, is limited and that the training and skills development programme will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Platreef Project.

25.6.1 Infrastructure Risks

Ivanplats has secured agreement for the supply of an adequate bulk water supply for the project. The ramp up of the supply of the grey water from a local water treatment works is however dependant on the construction of supporting infrastructure by the municipality. Matching the ramp up of the supply to the needs of the mine production, poses a risk to the Platreef Project, in that it can restrict the expansion and ramp up to steady state. Continuous engagement with stakeholders, the appointment of dedicated consultants, monitoring of progress and active investigation into alternative sources are some of the mitigation methods being implemented by Ivanplats.

Illegal plot cutting in the designated servitude could affect the placement of project infrastructure and hinder construction. Specifically, the placement of the bulk water supply pipeline and the Eskom 132 kV supply. Fencing of the pipeline servitude is a proposed mitigation.

25.6.2 Infrastructure Opportunities

Being on the forefront of mining development in the Bushveld Complex, the Platreef Project should investigate methods on how to use the new infrastructure upgrades in the area to the Platreef Project's advantage. One such opportunity being pursued is to include suitable changes to the N11 (off-ramps etc.) early and as part of the N11 upgrade project being developed by SANRAL.

Identification of other mining projects in the area and collaboration with such companies can mitigate risk and minimise costs for common infrastructure projects to be undertaken.

26 RECOMMENDATIONS

26.1 Platreef 2022 FS

The Platreef 2022 FS is the current development scenario for the Platreef project. It has advanced the development plan for the Platreef Project and increased the confidence in the Mineral Reserve to a feasibility study level of accuracy. This level of study will provide Ivanplats the information to further progress the financing and marketing negotiations that have been undertaken to date. The Platreef 2022 FS will provide the basis for detailed planning of execution activities and to update the long-term development plans for Platreef. Drill Programme.

26.2 Drill Programme

Ivanhoe is planning a long-term delineation drilling programme in support of detailed mine access and stope designs for the following reasons:

- Positioning of footwall access drives is dependent on the footwall contact of the mineralisation; thus, this information is based on the results of the delineation drilling and needs to be available prior to development.
- If these footwall access drives are developed too soon, they could be in the incorrect position (causing ore loss in the stopes).
- Numerous drillholes are planned from individual drill sites. The last of these drillholes must be completed prior to the footwall access drives being developed.

Ivanhoe has noted that many of these considerations have not been fully incorporated into the Feasibility Study mine plan. A revised plan will be necessary prior to implementation to address concerns related to sequencing of development and time necessary to process drillhole information.

The planned delineation drilling is separated by Ivanhoe into two periods:

- A 25 m drill grid is planned for the first 10-years (Year 1 to Year 10). This covers the initial stope establishment and allows the nature of the orebody to be tested on a close-spaced grid ahead of mine ramp-up.
- From Year 11 onwards, the delineation grid is widened to a 50 m grid. This is on the assumption that close-spaced variability in the position of the mineralisation, and localised variability in grade, will be better understood following the tighter drill grid and mining in previous years.

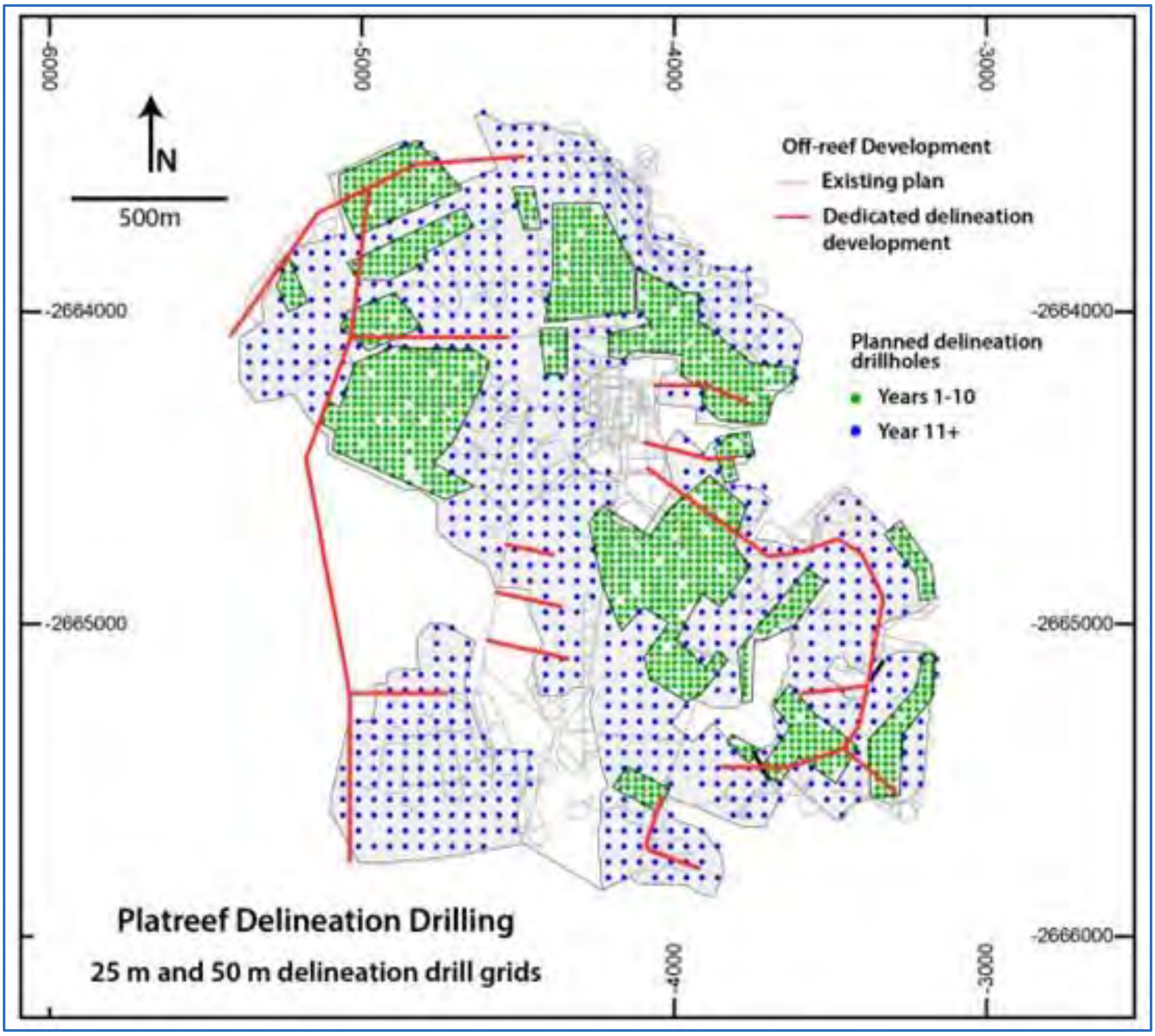
Ivanhoe has budgeted higher amounts of drilling up-front, as numerous mining areas are developed in quick succession. The drilling rate is assumed to stabilise as steady-state production is reached. From Year 11, mining is planned to continue for 25-years but the delineation drilling is estimated over only 15-years as in Ivanhoe's view, there should not be a need for delineation drilling in the final years of production.

It is anticipated that approximately 24,000 m of drilling per year on average will be necessary for Year 1 to Year 10, generating about 8,000 samples per year. The delineation drilling should stabilise at approximately 8,000 m per year (2,500 samples per year) thereafter.

Figure 26.1 and Figure 26.2 show the planned locations of the delineation drillholes and a perspective view. The estimated cost of the delineation drilling, including drilling, sampling and analysis, is $45–$50M ($127–$152/m) over the life of the mine and has been included in the costs of the Platreef 2022 FS.

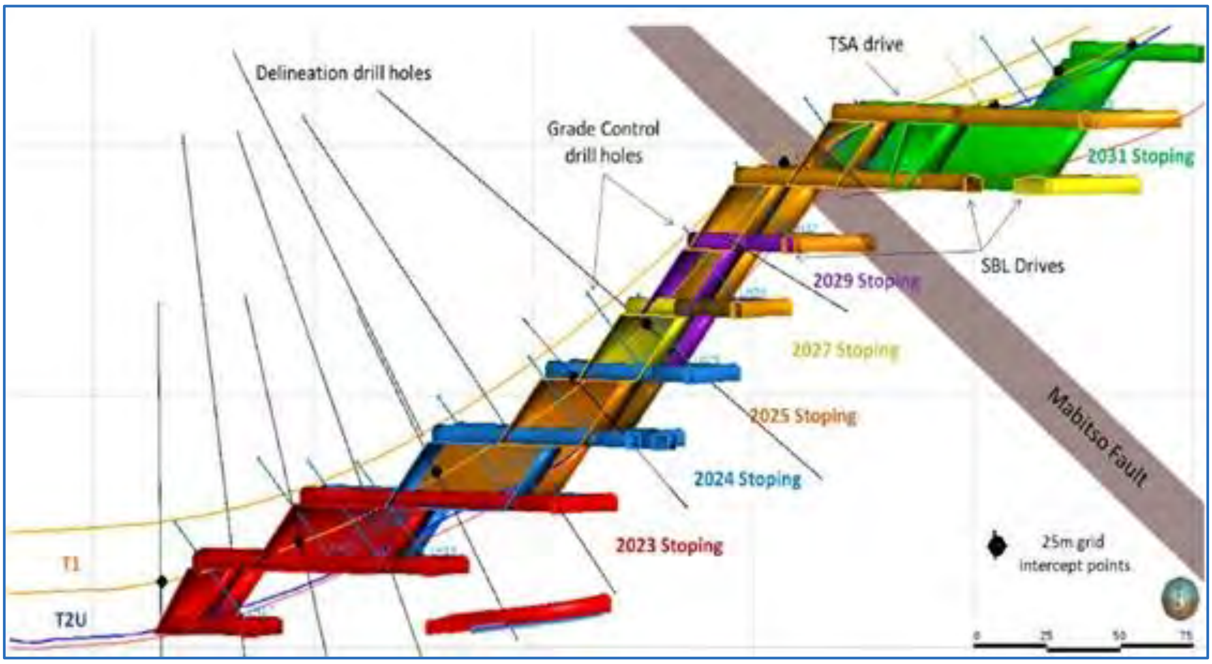
MTS has reviewed the estimates and given the assumptions in the proposed programme, considers the costs to be reasonable.

Figure 26.1 Proposed Delineation Drilling



Ivanhoe, 2022

Figure 26.2 Perspective View for Delineation Drilling



Ivanhoe, 2017

26.3 Resource Estimate

MTS recommends the assay tables for the ATS and, AMK drillhole campaigns in the acQuire database be validated against supporting documents. This is required before the AMK and ATS assay data can be used for Mineral Resource estimation. This is estimated depending whether this is done internally or by a consultant to cost approximately $15,000–$25,000.

26.4 Mineral Reserve Recommendations

The results and conclusions from the feasibility mining study indicate that further execution studies be undertaken.

26.5 Mining Recommendations

Recommendations regarding additional work and modifications to the current mine plans during the project phase:

- Optimise the definition drilling programme required for the initial mining areas.
- Determine stope sizes and footwall and hanging wall angles once the block sizes in the block model are reduced due to more detailed definition drilling.
- Monitor fragmentation during the development stage to eliminate the need for secondary breaking on development rock.
- Maximise the flow of ventilation from Shaft 1 (cooled air) to the deepest and warmest mine workings.

- Monitor and optimise the first development through the Tshukudu Fault.
- Develop an operating procedure to allow waste rock to go into primary Longhole Stopes and Drift-and-Fill areas.
- Set up a programme for ore pass monitoring to ensure the longevity of the passes.
- Set up a ground control observation programme to proactively recondition the ground support as needed during the mine life.

26.6 Metallurgical Recommendations

The metallurgical testwork programme has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes.

Preliminary open circuit bench scale flotation test work to evaluate the potential inclusion of Jameson cells in the cleaner flotation circuit showed reduced metallurgical performance but were deemed inconclusive due to the use of an outdated test procedure. It is recommended that these tests are repeated using the updated vendor procedure to confirm these findings.

Preliminary mini pilot plant test work was conducted during the Platreef 2022 FS, however, the plant was not adequately commissioned, stabilized and optimized. Additionally, the majority of the runs reflect commissioning runs on low grade samples with a 3PE+Au head grade of 2.9 to 3.8g/t. To further evaluate optimisation opportunities and confirm additional detail design parameters, additional pilot plant test work on high grade samples aligned to the early years of mining (> 5g/t 3PE+Au) is proposed as part of the project implementation phase.

The mini-pilot testwork included trials of an SIBX reagent suite with preliminary data indicating this to be a viable alternative to the copper collector reagent suite. Additional testwork should be conducted to confirm this result and the inclusion of an SIBX make-up and dosing system should be undertaken during project implementation.

Pilot scale column test work is recommended to confirm the additional concentrate upgrade potential in a column cell as aligned to the Platreef design flowsheet.

The potential for cost savings should be evaluated further during the project implementation phase as follows:

- The 0.77 Mtpa concentrate de-watering equipment, as sized based on the 2017 FS benchmarked flux information, is considered adequate for the required duty. It is however noted that there is the potential to reduce the size of the 0.77 Mtpa concentrate dewatering equipment based on the findings from concentrate dewatering test work conducted during the Platreef 2022 FS
- The installation of a tailings vacuum disk filter circuit to replace the vacuum belt filter as currently allowed for in the 0.77 Mtpa concentrator design for the Platreef 2022 FS
- The potential inclusion of an HPGR circuit as an alternative to the tertiary crushing circuit for the 4.4 Mtpa concentrator (Phase 2) should be evaluated further as part of the phased implementation programme.

26.7 Infrastructure

Moving forward, regular interfacing and liaison with the assigned project teams representing Eskom, the Masodi WWTW construction, and the Joint Water Forum should continue to assess developments and status of external infrastructure projects, directly affecting the Platreef Project.

Continued investigations into availability of alternative water sources for the project is recommended for redundancy and security of supply. Continued monitoring of the progress toward the building of the infrastructure to support the water supply is recommended.

26.8 Environmental, Social and Community

Ivanhoe has a programme of work in place to comply with the necessary environmental, social and community requirements. Key work should continue to include:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRD Act), the National Environmental Management Act (NEM Act) as well as the EP and IFC Performance Standards.
- Stakeholder Engagement Process (SEP) in accordance with the NEM Act and the IFC Principles.
- Specialist investigations in support of the ESIA.
- An updated Integrated Water Use License Application (IWULA) in compliance with the National Water Act (NWA).
- An updated Integrated Waste Management License in compliance with the National Environmental Management Waste Act (NEMWA).

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27.2 Glossary of Symbols and Units

Table 27.1 Table of Symbols and Units

Meaning	Symbol / Unit
Minute (plane angle)	'
Second (plane angle)	"
Percent	%
Less than	<
Greater than	>
Degrees Celsius	°C
Micrometre (micron)	µm
Annum (year)	a
Billion	b
Billion pounds	blb
Billion tonnes	bt
Centimetre	cm
Square centimetre	cm^2
Cubic centimetre	cm^3
Day	d
Days per week	d/wk
Dry metric tonne	dmt
Gram	g
Grams per tonne	g/t
Hour (not hr)	h
Hectare (10,000 m^2)	ha
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per tonne	kg/t
Thousand hours	kh

Meaning	Symbol / Unit
Kilometre	km
Kilometre per hour	km/h
Kilopascal Gauge	kPa(g)
Square kilometre	km^2
Thousand Troy ounces	koz
Kilopascal	kPa
Thousand tonnes	kt
Thousand tonnes per day	kt/d
Thousand tonnes per hour	kt/h
Litre	L
Pound	lb
Metre	m
Million	M
Square metre	m^2
Cubic metre	m^3
Metres per second	m/s
Million years	Ma
Metres above (mean) sea level	masl
Milligram	mg
Million pounds	Mlb
Millimetre	mm
Millimetres per annum	mm/a
Millimetres per hour	mm/h
Million ounces	Moz
Megapascal	MPa
Million tonnes	Mt
Millivolts	mV
Troy Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Metric tonne (1,000 kg)	t
Tonnes per annum	t/a
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per cubic metre	t/m^3
Watts per m^2	W/m^2
Week (seven days)	wk
Wet metric tonnes	wmt

27.3 Glossary of Abbreviations and Terms

Table 27.2 Table of Abbreviations and Terms

Description	Abbreviation / Term
Atomic Absorption Spectroscopy	AAS
Annual Information Form	AIF
Ammonium nitrate fuel oil	ANFO
Acoustic televiewer	ATV
Absolute value relative difference	AVRD
Bulk air cooler	BAC
Broad-based black economic empowerment	B-BEE
K2014089596 (South Africa) (RF) Proprietary Limited	BEE Co
Bushveld Igneous Complex	BIC
Bikkuri Model	BIK
Basal Melagabbronorite	BMGN
Base metal sulfide	BMS
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Carboxymethyl cellulose	CMC
Co-Operative Governance, Human Settlements and Traditional Affairs	CoGHSTA
Cemented paste fill	CPF
Clinopyroxenite	CPX
Coarse reject duplicates	CRD
Cemented rock fill	CRF
Certificed reference materials	CRM
Comments and Response Report	CRR
Corporate Social Responsibility	CSR
Critical Zone	CZ
Diamond drillhole	DDH
Drift-and-Fill	DF
Department of Mineral Resources and Energy	DMRE
Democratic Republic of Congo	DRC
Department of Water and Sanitisation	DWS
Environmental Authorisation	EA
Electrode-array-focussed resistivity	EAL
Enterprise Development	ED
Exploratory Data Analysis	EDA
Environmental Management Programme	EMP
Electronic multi-shot	EMS
Expression of Interest	EOI

Description	Abbreviation / Term
Engineering, Procurement, and Construction Management	EPCM
Environmental and Social Impact Assessment	ESIA
Fresh air passes	FAP
Footwall Assimilated Zone	FAZ
Fédération Internationale des Ingénieurs-Conseils	FIDIC
Forward Looking Statement	FLS
Front end loader	FEL
Feasibility study	FS
Footwall	FW
Fall-waveform-sonic	FWFS
General and administration	G&A
Gigabit (10^9 bits)	Gbit
Geographic information system	GIS
Great North Fault zone	GNF
Global Positioning System	GPS
Hemicellulose	HC
Heritage Impact Assessment	HIA
Historically disadvantaged South Africans	HDSA
Inductively coupled plasma-mass spectrometry	ICP-MS
Inductively coupled plasma-optical emission spectroscopy	ICP-OES
Induced polarisation	IP
Interim Protection of Informal Land Rights Act, 1996	IPILRA
Internal Rate of Return	IRR
Joint Water Forum	JWF
K2014043815 (South Africa) (RF) Proprietary Limited	EntrepreneurCO
K2014043822 (South Africa) (RF) Proprietary Limited	Community TrustCo
K2014043829 (South Africa) (RF) Proprietary Limited	Employee TrustCo
Low angled features	LAF
Local Area Network	LAN
Local Economic Development	LED
Limpopo Department of Economic Development, Environment and Tourism	LEDET
Load-haul-dump	LHD
Life-of-mine	LOM
Lower critical zone	LCZ
Lower zone	LZ
Merensky Cyclic Unit	MCU
Management Discussion and Analysis	MD&A

Description	Abbreviation / Term
Mokopane Interested and Affected Communities Committee	MIACC
Mokopane Interested and Affected Communities Development Forum	MIACDF
Mogalakwena Local Municipality	MLM
Mineral and Petroleum Resources Development Act 28 of 2002	MPRDA
Mineral and Petroleum Titles Registration Office	MPTRO
Mining right	MR
Mining Right Application	MRA
Maptek's stope optimiser	MSO
Material take-off quantity (m^3)	MTO
Million (metric) tonnes per annum	Mtpa
Magnetic susceptibility	Msus
Main transmission station	MTS
Mine Technical Services	MTS
Megavolt amperes	MVA
Main Zone	MZ
Marginal Zone Norites	MZN
Non-acid forming	NAF
Norite Cycles	NC
National Environmental Management Act 107 of 1998	NEMA
National Heritage Resources Act 25 of 1999	NHRA
Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects	NI 43-101
Notified maximum demand	NMD
Nearest neighbour	NN
Net Present Value	NPV
Net Present Value at 8% Discount Rate	NPV8%
Net Smelter Return	NSR
Overhead line	OHL
Ordinary kriging estimation method	OK
Optical televiewer	OTV
Potentially acid forming	PAF
Pay as you earn	PAYE
Preliminary Economic Assessment	PEA
Prefeasibility study	PFS
Platinum group element	PGE
Platinum group mineral	PGM
Pyroxenite-Norited-Zone	PNZ
Public Participation Processes	PPP

Description	Abbreviation / Term
Prospecting Right	PR
Point resistance	PR
Particle size distribution	PSD
Quality assurance and quality control	QA/QC
Qualified Person	QP
Return air raise	RAR
Reverse circulation	RC
Rustenburg Layered Suite	RLS
Chinese Renminbi (also called Yuan)	RMB
Rock Mass Rating	RMR
Run of mine	ROM
Rock quality designation	RQD
Semi-autogenous grinding	SAG
South African Heritage Resources Agency	SAHRA
South African National Roads Agency Limited	SANRAL
South African Revenue Services	SARS
System for Electronic Document Analysis and Retrieval (SEDAR) is a filing system developed for the Canadian Securities Administrators	SEDAR
Social Impact Assessment	SIA
Stay-in-business	SIB
small, medium and micro enterprise	SMME
Stope orientation zones	SOZ
Self potential	SP
SAG Performance Index	SPI
SAG Power Index	SPI
Spatial Planning and Land Use Management Act, 2013	SPLUMA
North seeking Gyro	SRG
Standard Reference Material	SRM
Species of Special Concern	SSC
Surface Use and Cooperation Agreement	SUCA
Short-wave infrared	SWIR
Tennis Ball Marker	TBM
Total cost package	TCP
Turfspruit Cyclic Unit	TCU
Time Domain Reflectometers	TDR
Total Dissolved Solids	TDS
Thabazimbi-Murchison Lineament	TML
Technical Report	TR

Description	Abbreviation / Term
Tailings storage facility	TSF
Unconfined Compressive Strength	UCS
Upper Critical Zone	UCZ
Unemployment Fund	UIF
Unconfined Tensile Strength	UTS
Upper Zone	UZ
Vertical crater retreat	VCR
Ventilation on demand	VOD
Vertical seismic profile	VSP
Work breakdown structure	WBS
Waste Management Licence	WML
Waste Water Treatment Works	WWTW
South African Rand	ZAR