OreWin
Independent Mining Consultants

Ivanhoe Mines Ltd.

Platreef Integrated Development Plan 2025

March 2025

Job No. 23002

IMPORTANT NOTICE

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Title Page

Project Name:	Platreef Project
Title:	Platreef Integrated Development Plan 2025
Location:	Limpopo Province, Republic of South Africa
Effective Date of Technical Report:	15 February 2025
Mineral Resource Amenable to Selective Underground Mining Methods:	15 February 2025
Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	15 February 2025
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	15 February 2025
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Validation of resource models as current using current economic inputs:	15 February 2025
Effective Date of Mineral Resources:	15 February 2025
Effective Date of Mineral Reserves:	15 February 2025

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Signature Page

Project Name:	Platreef Project
Title:	Platreef Integrated Development Plan 2025
Location:	Limpopo Province, Republic of South Africa

Effective Date of Technical Report: 15 February 2025

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1 SUMMARY

1.1 Introduction

The Platreef Integrated Development Plan 2025 (Platreef IDP25) 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 2022 Feasibility Study (Platreef 2022 FS) and the Platreef Integrated Development Plan 2020 (Platreef IDP20) on the Platreef Project that have formed the basis of the Platreef IDP25, which summarises the current Ivanhoe development strategy for the Platreef Project.

The Platreef IDP25 provides updates to the Project Mineral Reserves, and the studies at Feasibility (FS), and a Preliminary Economic Assessment (PEA). The following are the key features of the Platreef IDP25:

- Platreef 2025 FS (Plant expansion to 4.07 Mtpa).
- Platreef 2025 PEA (Plant expansion to 10.67 Mtpa).

The Platreef IDP25 is a Phased Development Plan based on continued development and production from Phase 1 at 770 ktpa production which is then followed by Phase 2, which adds a single 3.30 Mtpa concentrator, increasing the production rate to 4.07 Mtpa. Phase1 and Phase 2 are in the Platreef 2025 FS.

The Platreef 2025 PEA describes a change in production rate for the Project that will require separate capital costs and infrastructure. The Platreef 2025 PEA includes Phase 3 which includes two more plant expansions of 3.3 Mtpa each to bring the total processing capacity to 10.67 Mtpa.

The summary results of the Platreef IDP25 are in Table 1.1.

Table 1.1 Platreef IDP25 Summary Results

Item	Units	4.1 Mtpa FS	10.7 Mtpa PEA[6]
		Total/Average Life-of-Mine	
Total Ore Milled	million tonnes	130	235
Platinum	g/t	1.88	1.83
Palladium	g/t	1.93	1.92
Gold	g/t	0.29	0.29
Rhodium	g/t	0.13	0.13
3PE+Au[1]	**g/t**	**4.22**	**4.17**
Copper	%	0.16	0.16
Nickel	%	0.33	0.34
Average Production			
3PE+Au[1]	kozpa	432	927
Copper	ktpa	5	12
Nickel	ktpa	9	20
Life-of-mine	years	35	29
Capital Cost			
Initial capital (4.1 Mtpa)[5]	$ million	1,243	1,243
Expansion capital (to 10.7 Mtpa)	$ million	N/A	803
Sustaining capital	$ million	1,590	3,505
Peak funding	$ million	917	949
Operating Cost			
Total cash cost after credits[2]	$ / 3PE+Au oz	599	511
All-in cash cost after credits[3]	$ / 3PE+Au oz	704	641
Site Operating Costs	$ / tonnes milled	60	56
Financial Metrics[4]			
Operating margin	%	40	45
After-Tax $NPV_{8\%}$	$ million	1,393	3,195
After-Tax IRR	%	20	25
Project payback period	years	7.8	8.7

1. 3PE+Au = platinum, palladium, rhodium and gold.
2. Includes the impact of the gold, platinum and palladium streaming agreements.
3. All-in cash costs include sustaining capital costs.
4. Long-term metal price assumptions for economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.
5. 4.1 Mtpa FS Initial Capital includes $70 million for the completion of Phase 1.

6. The PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves—and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

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 main assets:

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

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 Organization for Metals and Energy Security; and JGC Holdings Corporation, 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 24°05'S and 28°59'Ein 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

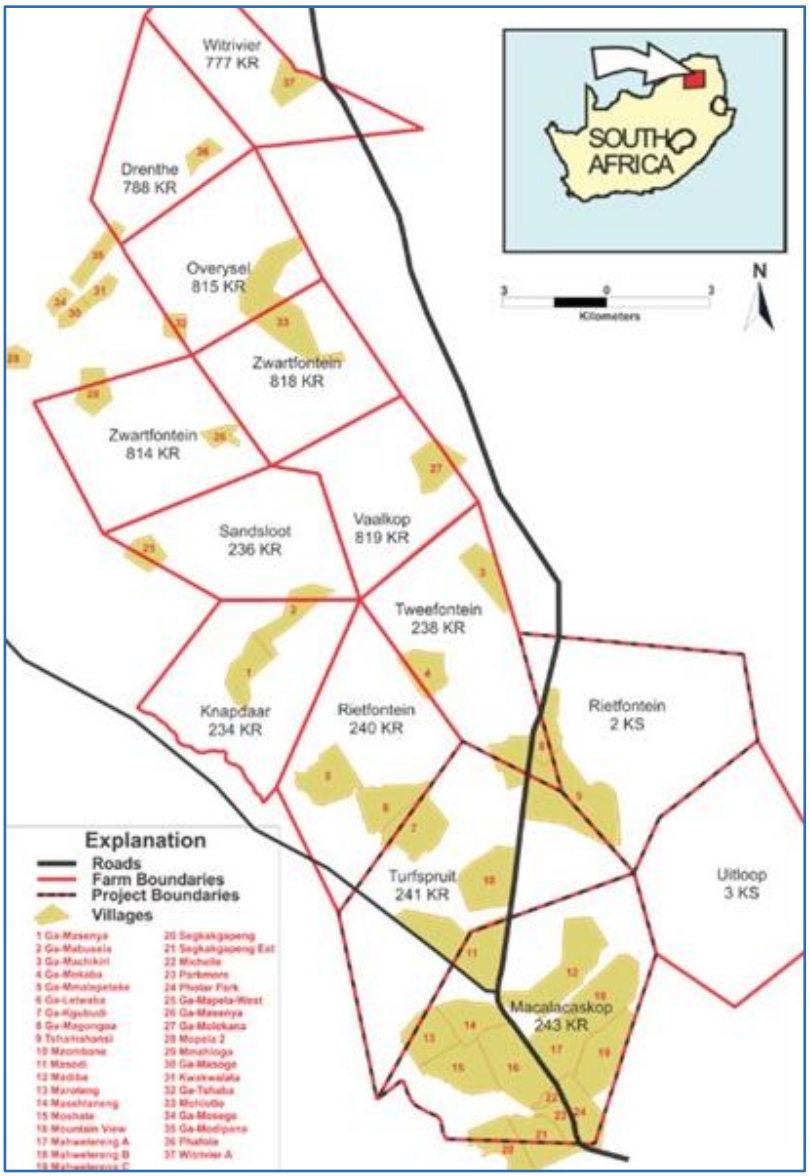
Ivanplats Proprietary Limited was the holder of a Prospecting Right, which was notarially executed on 2 February 2006 and registered in the South African Mineral and Petroleum Titles Registration Office (MPTRO) on 9 February 2006 under registration number 55/2006 PR, entitling Ivanplats to prospect for 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.

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 restrictions on large public gatherings, aimed at preventing the spread of COVID-19.

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, Rustenburg 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 IDP25.

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 and feasibility 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 (NC1and 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.

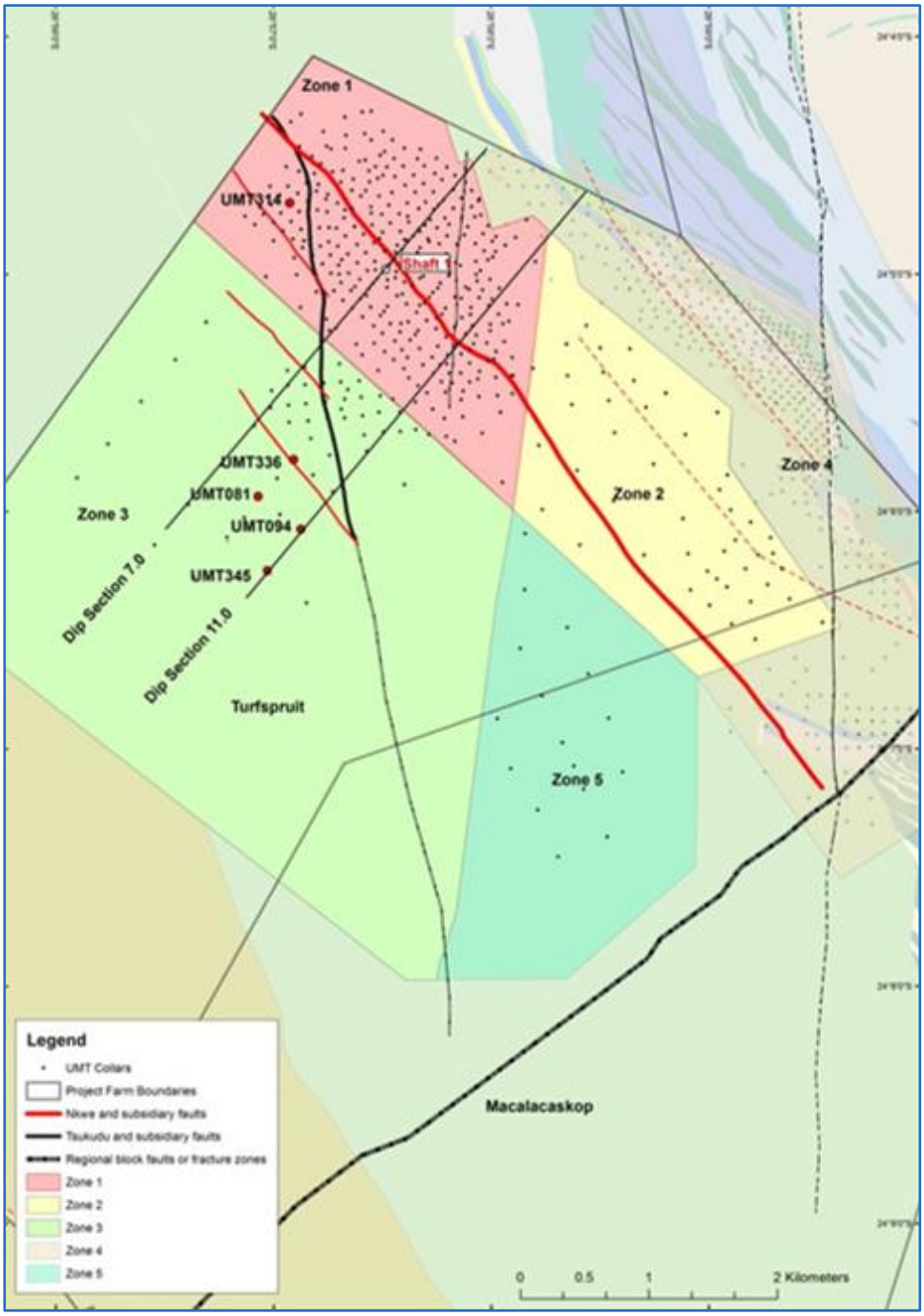
Within the TCU, high grade PGE–Ni–Cu mineralisation is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, bound by chromitite stringers and containing coarse-grained sulfides; this is known as T2, with the mineralised portion referred to as T2MZ. The 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), as shown in 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 and TMT015 was drilled between them. 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 geotechnical 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 was 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 Laboratory (Ultra Trace) in Perth, Genalysis Laboratories in Perth and Johannesburg (Genalysis), SGS Metallurgical Services (SGS) in South Africa, Acme in Vancouver, and ALS Chemex (ALS) in Vancouver. Bureau Veritas Minerals Pty Ltd (Bureau Veritas) assumed control of Ultra Trace during June 2007 and was 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

During the exploration work programmes, Wood Plc (Wood) reviewed the sample chain of custody, quality assurance and quality control (QA/QC) procedures, and qualifications of analytical laboratories. In addition, the assay database, core logging, and geological interpretations were reviewed.

The Qualified Person has inspected a number of drillhole cores and reviewed the logging, sampling and assaying processes.

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 methods 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 was estimated using revised geological interpretations and incorporated 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 was 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 Mineral Resource is Mr Jeremy Witley, Pr. Sci. Nat., of the MSA Group (Pty) Ltd. Mineral Resources have been estimated using drill core 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 (%)**	**PtEq (g/t)**
3 g/t	204	2.11	2.11	0.34	0.14	4.70	0.18	0.35	7.01
2 g/t	346	1.68	1.70	0.28	0.11	3.77	0.16	0.32	5.82
1 g/t	716	1.11	1.16	0.19	0.08	2.55	0.13	0.26	4.17
Indicated Mineral Resources Contained Metal									
Cut-off 3PE+Au	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlb)**	**Ni (Mlb)**	**PtEq (Moz)**
3 g/t	–	13.9	13.9	2.2	0.9	30.9	800	1,597	46.0
2 g/t	–	18.7	18.9	3.1	1.2	41.9	1,226	2,438	64.7
1 g/t	–	25.6	26.8	4.5	1.8	58.8	2,076	4,108	95.9
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 (%)**	**PtEq (g/t)**
3 g/t	225	1.91	1.93	0.32	0.13	4.29	0.17	0.35	6.55
2 g/t	506	1.42	1.46	0.26	0.10	3.24	0.16	0.31	5.22
1 g/t	1,431	0.88	0.94	0.17	0.07	2.05	0.13	0.25	3.62
Inferred Mineral Resources Contained Metal									
Cut-off 3PE+Au	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlb)**	**Ni (Mlb)**	**PtEq (Moz)**
3 g/t	–	13.8	14.0	2.3	1.0	31.0	865	1,736	47.4
2 g/t	–	23.2	23.8	4.3	1.6	52.8	1,775	3,440	84.9
1 g/t	–	40.4	43.0	7.8	3.1	94.3	4,129	7,759	166.5

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: 8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni, and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. 3PE+Au = platinum, palladium, rhodium and gold.

8. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by the assumed price and recovery for each metal.
9. Totals may vary 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.

PtEq grade and content is affected by changes to metal prices and recoveries.

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. MSA 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 Complex (BC) 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 IDP25. 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 2025 FS cost estimates have been done to a feasibility study level of accuracy. For further detail on cost estimates, refer to Section 21.

The stope shapes used in the Platreef 2022 FS stayed largely the same with minimal adjustments in Platreef 2025 FS. The adjustments made were mostly based on technical inputs (not economical) that included bracket pillars around major faults zones and changing the initial stopes on 750 m Level to longhole stopes as opposed to drift and fill. The calculation inputs to the updated NSR values have changed with commodity prices changing and updated recovery models. The NSR runs using 2022 FS inputs versus Platreef 2025 FS inputs showed good correlation with variances smaller than 3% (lower NSR value for Platreef 2025 FS) up to 2024 and 4.4% for the LOM. The main contributors for the slightly lower NSR values were lower commodity prices and lower concentrator recovery inputs for lower grade ore.

A cut-off grade analysis was completed by mining method, and the resulting cut-off grade ranged from $47–$58/t. Ivanplats' operational philosophy required additional allowances for capital and profit margin to be included in the NSR cut-off evaluation. Based on BDT20 the 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.

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, 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. This was done in the studies of 2015 and 2017, although it has not been updated in this study the Platreef operation team is currently reevaluating all stope designs. Current analysis determined the Platreef 2025 FS designs are valid. Until 2029 a cut-off grade for low grade development of 1.75 g/t 3PE+Au has been used. After 2029 the cut-off grade is $130/t NSR.

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.

The Platreef 2025 FS cost estimates have been done to a feasibility study level of accuracy.

Table 1.3 shows the Probable Mineral Reserve for Platreef.

Table 1.3 Platreef Mineral Reserve as at 15 February 2025

Classification	Ore (Mt)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Proven	-	-	-	-	-	-	-	-
Probable	129.7	1.88	1.93	0.29	0.13	4.22	0.16	0.33
Total	**129.7**	**1.88**	**1.93**	**0.29**	**0.13**	**4.22**	**0.16**	**0.33**

Classification	Ore (Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (kt)	Ni (kt)
Proven	-	-	-	-	-	-	-	-
Probable	129.7	7.82	8.05	1.21	0.54	17.62	209	426
Total	**129.7**	**7.82**	**8.05**	**1.21**	**0.54**	**17.62**	**209**	**426**

1. Mineral Reserves have an effective date of 15 February 2025. The Qualified Person for the estimate is Mr 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 Pt, $815/oz Pd, $1,300/oz Au, $1,500/oz Rh, $8.90/lb Ni and $3.00/lb Cu.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Long-term metal-price assumptions used for the feasibility study economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Totals may vary due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

1.12 Geotechnical Investigations

There have been no additions to the geotechnical database since the FS 2016 and FS 2022 reports. Consequently, the geotechnical rock mass data and laboratory testwork (stress regime and strength tests) remain unchanged. The investigation was based on all available geotechnical and structural data, including data from geotechnically logged boreholes. Geotechnical design parameters were derived to manage potential risks, governing stope and mine access design, including backfill and support requirements. The updated mine design has been reviewed in line with the geotechnical design parameters.

1.12.1 Geological and Geotechnical Characteristics

The Tshukudu fault remains a major geotechnical hazard due to poor-quality rock. Development through the fault must be carefully planned to minimize delays and costs, as it traverses the entire lease area and is necessary for accessing ore to the west. Specialized support, such as spiling rings, resin injection, and arches, is recommended, with a slow development rate through the fault. The fault's alteration varies, suggesting that some areas may allow for development with fewer delays and less intensive support, provided there is no water ingress. Geotechnical drilling will be necessary to delineate and characterise the Tshukudu fault during implementation.

Other geological features may cause instability, leading to ground falls, overbreak, production delays, equipment damage, and increased support costs. If excavations must intersect these structures, implementing support strategies, good mining practices, hazard identification, and monitoring is essential to manage risks.

Rock mass conditions range from very poor to extremely good, but most of the rock mass is generally fair to good. Lower rock mass quality should be expected, and excavations in these areas may require more stringent support or experience increased overbreak. The MZ, FAZ, and T1 units are well-represented compared to the NC2 unit, which had limited core available for geotechnical analysis, leading to low confidence in its characterization. Additional information is required for assessing development infrastructure in deeper mine areas. Future boreholes will need to intersect and extend beyond the LZ, as current boreholes were terminated at this level.

The stress testing conducted across the project area revealed that the pre-mining stress state at all tested locations is moderate to low relative to the strength of the rock. The maximum horizontal to vertical stress ratio is 1.1. Although the major geological structures identified on the Platreef property have not been associated with any large-magnitude tectonic earthquakes, it should be noted that the planned mining activities will cause significant stress changes along these major geological structures.

1.12.2 Long-Term Access Development and Critical Excavations

Numerical analysis indicates that ramps, main drives, and crosscut developments are sited away from stoping-related stress changes. Specialized support is anticipated for the Tshukudu fault, with reduced development rates expected in fault zones. No critical excavations intersect the Tshukudu and Nkwe fault zones. Additional support is recommended when excavations intersect major structures.

1.12.3 Vertical Shafts and Ore Passes

The main vertical shafts are sited within the shaft pillar area, as these should not be adversely affected by stress changes over the life of mine. Geotechnical investigations continue for the development of shafts. It is important to avoid geological structures, which may cause challenges during raiseboring.

The estimated longevity for the analysed silos and ore passes are lower than their required throughput during their operational life. All the designed silos/ore passes will need to be rehabilitated or replaced. Stand-by silos/ore passes need to be planned earlier as the designed ones are not anticipated to last for the expected life due to high throughput tonnages expected and this will affect production. Protective, 300 mm abrasion resistant liners and cable bolting (through any intersected major structures) are recommended to be installed to improve the longevity.

1.12.4 Stope Access Development

Low stress damage is anticipated in stope drifts during early mining phases, with S1 support adequate for most stope drifts. High stress concentrations are expected in stope access drives and drifts when sill pillars are formed after 2030, necessitating a change in support category. The centre-out mining sequence helps manage stress levels, with S1 support adequate for most access and stope drives.

1.12.5 Longhole Stoping

Cemented Paste Fill (CPF) will be utilised in Longhole Open stopes to maintain stability. The majority of Longhole stopes will be mined using top accesses, improving overbreak management. Tight filling is required in primary stopes to maximize secondary stope recovery. High stress concentrations during sill pillar extraction will necessitate careful sequence adaptation and support installation.

1.12.6 Drift and Fill

Drift and fill will be carried out where the orebody is thinner and CPF will also be used to maximise extraction. A primary, secondary, and tertiary sequence is recommended for geotechnical control. Numerical analysis indicates low failure depth in primary drifts, increasing with secondary drift extraction. Tight filling is essential for stability, with additional cable anchors required if deterioration occurs.

1.12.7 Monitoring

Monitoring is essential in geotechnical mine design to evaluate predicted behaviour against actual performance and optimize designs. Key activities include regular geotechnical mapping, elastic modelling, and lidar scanning for excavation monitoring. Backfill management and monitoring will be important to ensure suitable performance. A seismic monitoring system is advised from 2030 to understand rock mass behaviour during sill pillar formation. This was designed to manage seismic risks, ensuring sensitivity and accuracy in detecting rock mass damage and strainbursts. These strategies are crucial for maintaining safety and optimizing mine operations.

1.13 Mining

The Platreef 2025 FS includes two main phases. Phase 1 is defined as establishing Shaft 1 remaining infrastructure, development and other infrastructure required to sustain 770 ktpa. Phase 2 commences just after equipping Shaft 3 when production begins to ramp up to 4.07 Mtpa and Plant 2 is commissioned.

Key points of the Platreef 2025 FS:

- Shaft 1 changeover and mechanised access to shaft bottom has been established.

- Development has started and continuous on 750 m, 850 m and 950 m Levels. The initial focus is to establish infrastructure required on every level to commission Shaft 3 capable of hoisting 4 Mtpa combined from the three levels by February 2026. After Shaft 3 commissioning the focus is on accessing the stoping areas with higher ground handling capacity than in 2022 FS, thereby expediting the ramp up.
- First longhole stoping establishment – August 2025.
- Shaft 3 commissioned and stoping tonnage ramp up commence - February 2026.
- Shaft 4 equipped for ventilation – Q3'25.
- Plant 1 ore processing – Q4'25.
- Plant 2 ore processing – Q4'27.
- Shaft 5 holed for ventilation – Q1'27.
- Shaft 2 commissioned – Q3'28.

1.13.1 Mining Method and Mine Design

For the Platreef 2025 FS, changes were made to the mine design to meet the requirements for the Phased Development Plan including development to establish Shaft 3 as a hoisting shaft and not a ventilation shaft as per 2022 FS. Also, design changes were made to streamline rock handling from lower levels to Shaft 3 via a crushing and conveyor system and dedicated tramming truck haulages on each level to separate logistics from ground handling. The major changes from 2022 FS to 2025 FS are shown below.

Table 1.4 Comparison of 2022 FS with 2025 FS

Item	2022 FS	2025 FS
Ore Mining Rate Maximum Hoisting and ROM	5.17 Mtpa 6.0 Mtpa	4.07 Mtpa 4.7 Mtpa (hoisting) & 5.25 Mtpa (ROM)
Shaft 3	Ventilation return shaft	Equipped to hoist 4 Mtpa
Shaft 2	Major shaft – 6 Mtpa	Commissioned as logistics shaft in Q3'28.
Ore Handling	Lowest tip 950 m Level	Lowest tip 1,050 m Level (conveyor system from 1,100 m Level to 950 m Level
Crushers	2 x crushing stations	5 x crushing stations
Development Sizes	5 x 5 m	6 x 5.5 m
Development Jumbos	**Function** Face drilling Bolter utilised for bolts **Rates** Dev.: 225 m/mth/jumbo Drift and fill: 25,000 t/mth	**Function** Barring Support: Bolts & mesh (bolters removed) Face drilling **Rates** Dev.: 60-180m/mth ramp up Drift and fill: 20,000 t/mth

Item	2022 FS	2025 FS
Loading and Hauling	18 t LHD and 54 t Truck Trucks used on every level	14 t LHD and 42 t Truck (<2029) 18 t LHD and 54 t Truck (>2029) (Shaft 1 size limitation – 65 t planned for Phase 3) Ore passes on inter levels to main haulage levels. Limited trucks on inter levels for development only. Trucks used to haul tonnes from ore passes to main level tips with minimal hauling in ramps.
Hauling	All tonnes hauled back to main shaft tips from all levels	Trucks hauling limited to main levels and development on inter-levels. Use ore passes for ore tonnes on each level that reports on main levels
LH Stopes	Bottom-up mining No easer for slot drilling All down-holes	Levels top-down Between levels bottom-up Easier for slot blasting incl. Initial stopes using up-holes but LOM stopes using down-holes
Target Initial Stopes	High grade drift and fill stopes far away from shaft complex	Medium grade LH stopes close to shaft
UG Equipment Quantities	Central workshops on main levels	De-centralised approach with drill rig being maintained in satellite workshops close to working areas.

The Platreef Project is designed based on highly-mechanised longhole stoping and drift and fill mining methods.

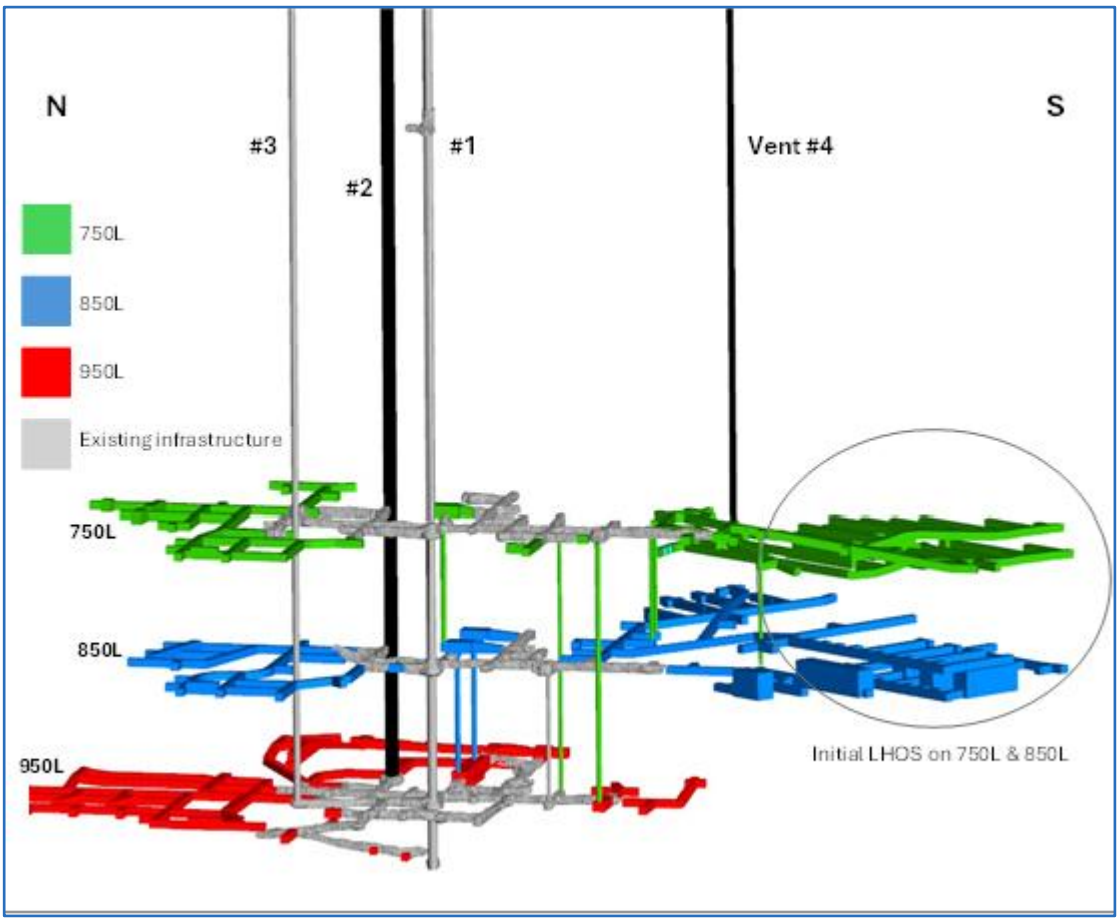
The Platreef 2025 FS evaluates a phased development of Platreef, with an initial 770 ktpa underground mine and a 770 ktpa capacity concentrator, targeting high grade mining areas close to Shaft 1. First concentrate production for this option is targeted in 2025, with the hoisting Shaft 3 commissioned in Q1'26 and personnel and material Shaft 2 commissioned in Q3'28. The second concentrator with a 3.3 Mtpa capacity enabling a combined process capacity of 4.07 Mtpa will be commissioned in Q4'27.

1.13.1.1 Mine Access

During Phase 1 of the Platreef 2025 FS, the main access to the mine will be via a 986.3 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. The initial focus will be to establish Shaft 3 underground infrastructure that includes tips, ore passes, crusher and conveyance system from different levels to Shaft 3. The secondary priority will be to access the initial stopes on 750 m and 850 m Levels. Shaft 3 will enable the hoisting capacity to increase from 720 ktpa to 4.7 Mtpa. Shaft 1 will also be a ventilation return shaft with Shaft 1 serving as an intake shaft. The secondary focus will be to access the first stopes on 750 m and 850 m Levels respectively.

Figure 1.4 shows the Shaft 1 to 4 locations and main access levels in an elevated view (looking east).

Figure 1.4 Mine Access Isometric Looking 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 and drift and fill access ramps will access the production areas. 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

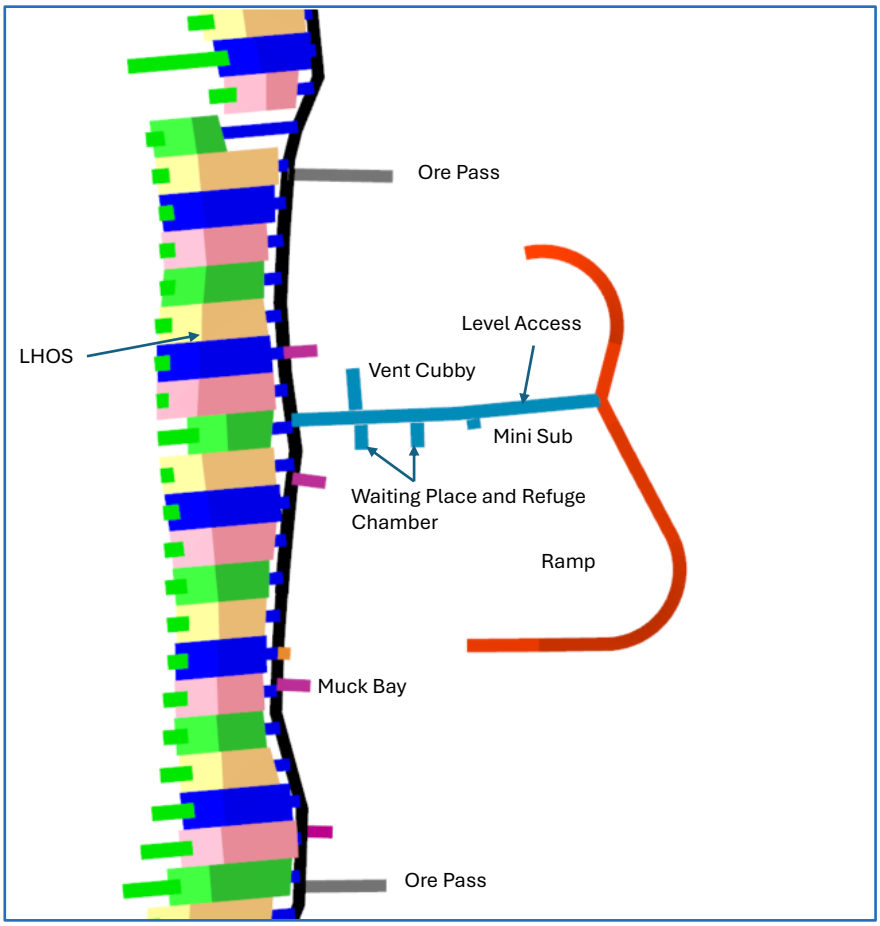


Figure 1.6, Figure 1.7 and Figure 1.8 show the plan view of the initial development on three main levels of 750 m, 850 m, and 950 m. A view of the initial development is shown in Figure 1.9.

Figure 1.6 Platreef 2025 FS – Initial (Pre-Production) Development on 750 m Level

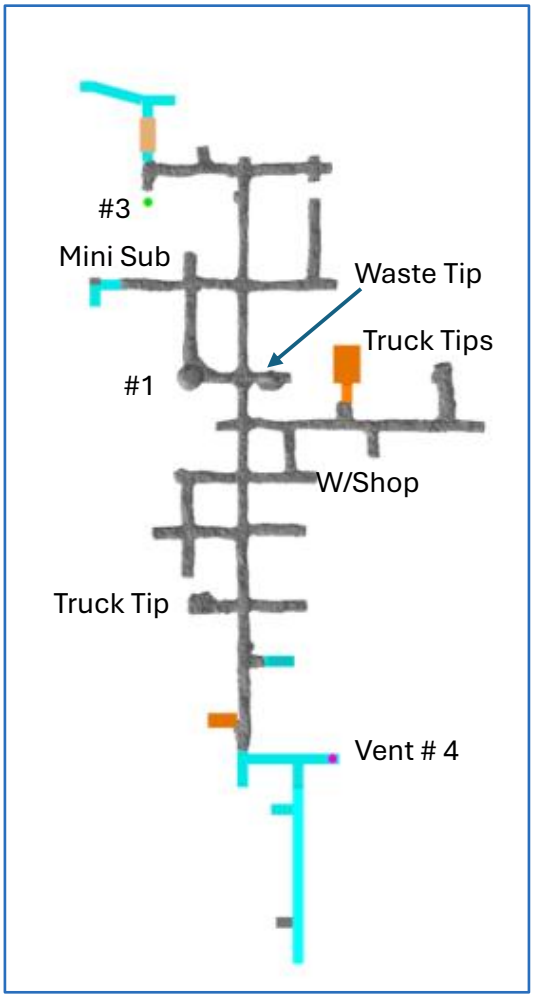


Figure 1.7 Platreef 2025 FS – Initial (Pre-Production) Development on 850 m Level

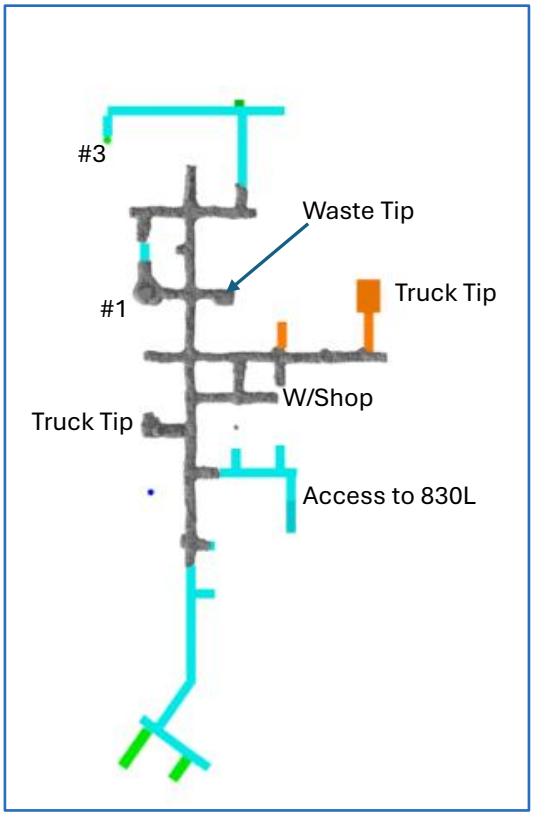


Figure 1.8 Platreef 2025 FS – Initial (Pre-Production) Development on 950 m Level

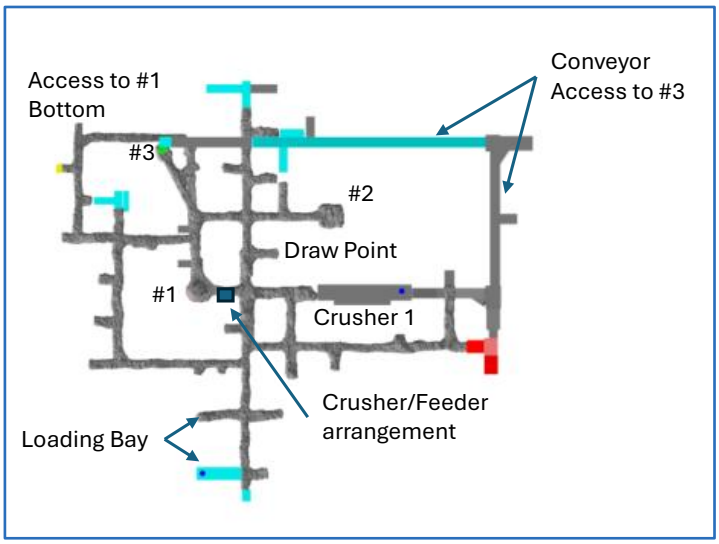
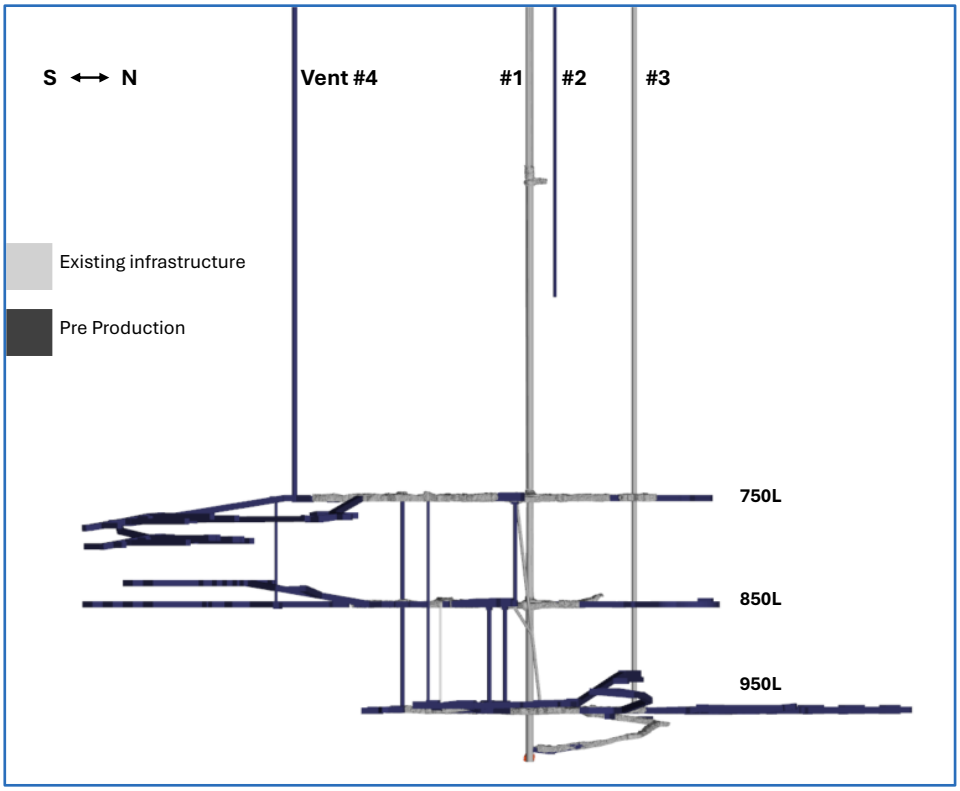
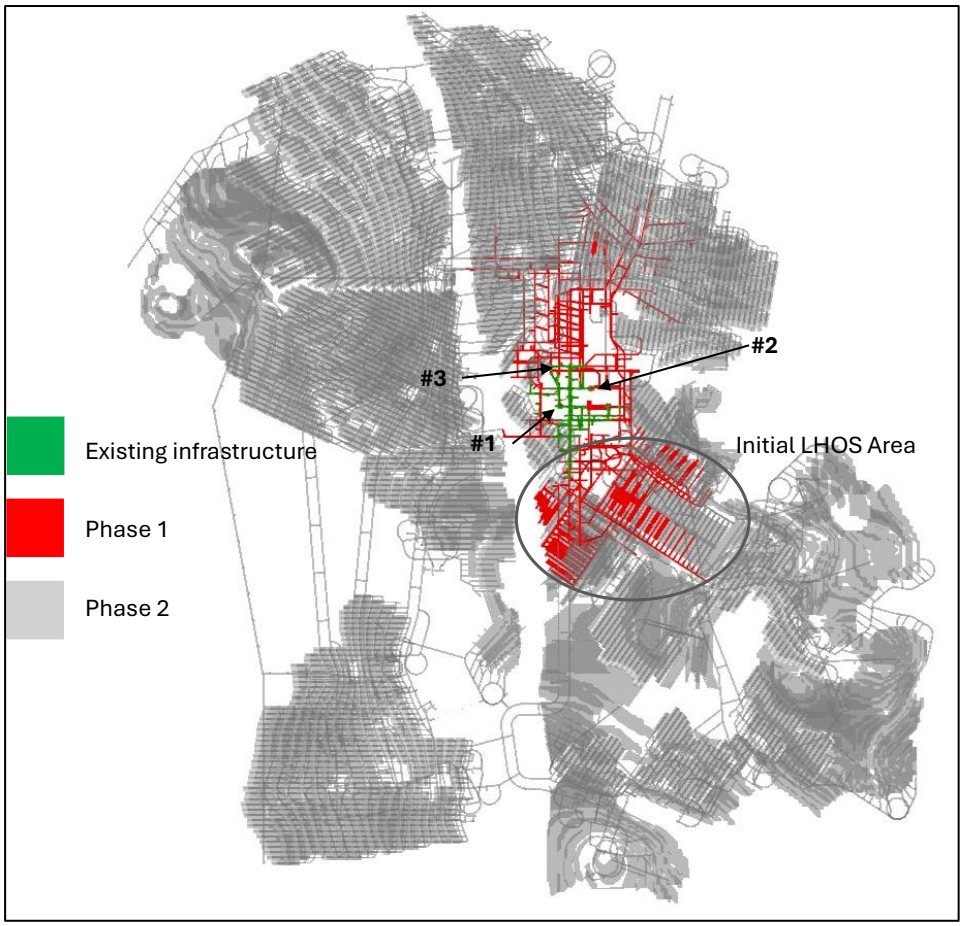


Figure 1.9 Platreef 2025 FS – Initial (Pre-Production) Development



In Phase 2, primary access to the mine will be by a 996 m deep, 10 m diameter personnel and material (logistics) shaft (Shaft 2) and Shaft 1 and Shaft 3 will remain as secondary and tertiary accesses to the mine. During mine production in Phase 1, Shaft 1 will serve as a ventilation intake too. In Phase 2, Shafts 1, 2 and 5 will also serve as ventilation intakes while Shafts 3, 4, 6 and 7 will be ventilation exhaust raises. Shafts 3–7 will be raisebored shafts and reamed to 5.1mØ. See Figure 1.10 for Phase 1 and Phase 2 infrastructure development.

Figure 1.10 Platreef 2025 FS – Phase 1 and Phase 2



The shafts complex will connect to 750 m, 850 m and 950 m Levels while four main access levels will be established as primary haulage levels. These are the 750 m, 850 m, 950 m and 1,050 m haulage levels. Figure 1.11 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 as shown in Figure 1.12. 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 1.11 Mine Access Layout

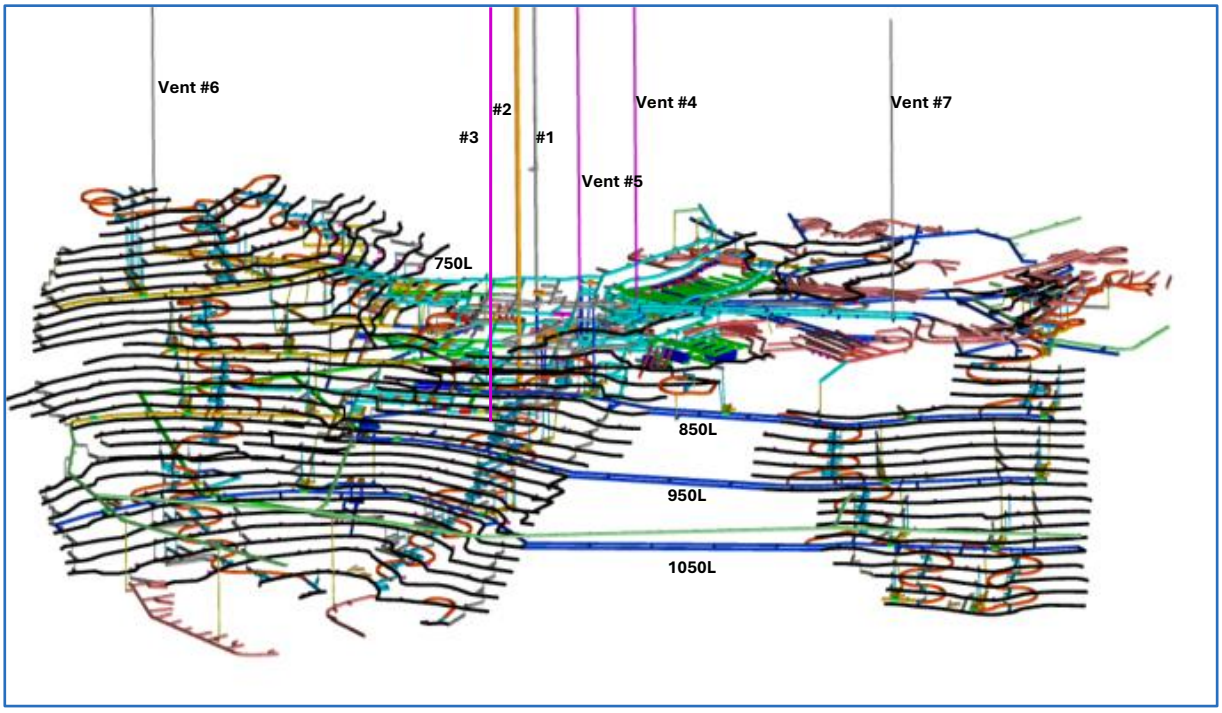
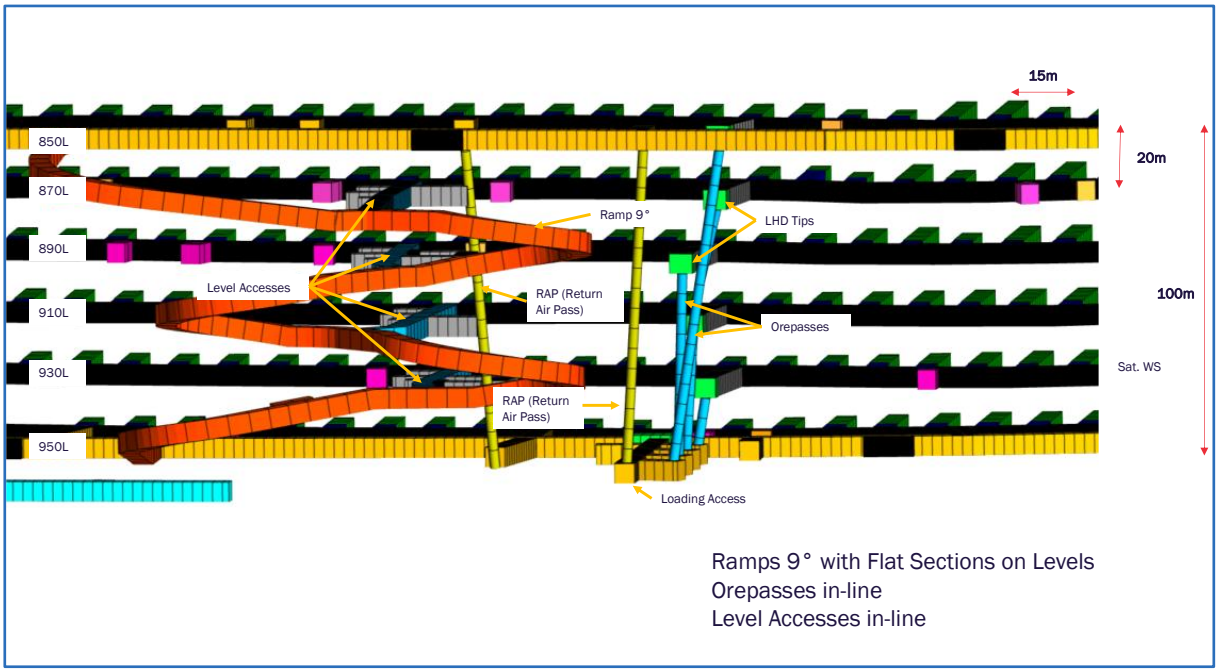


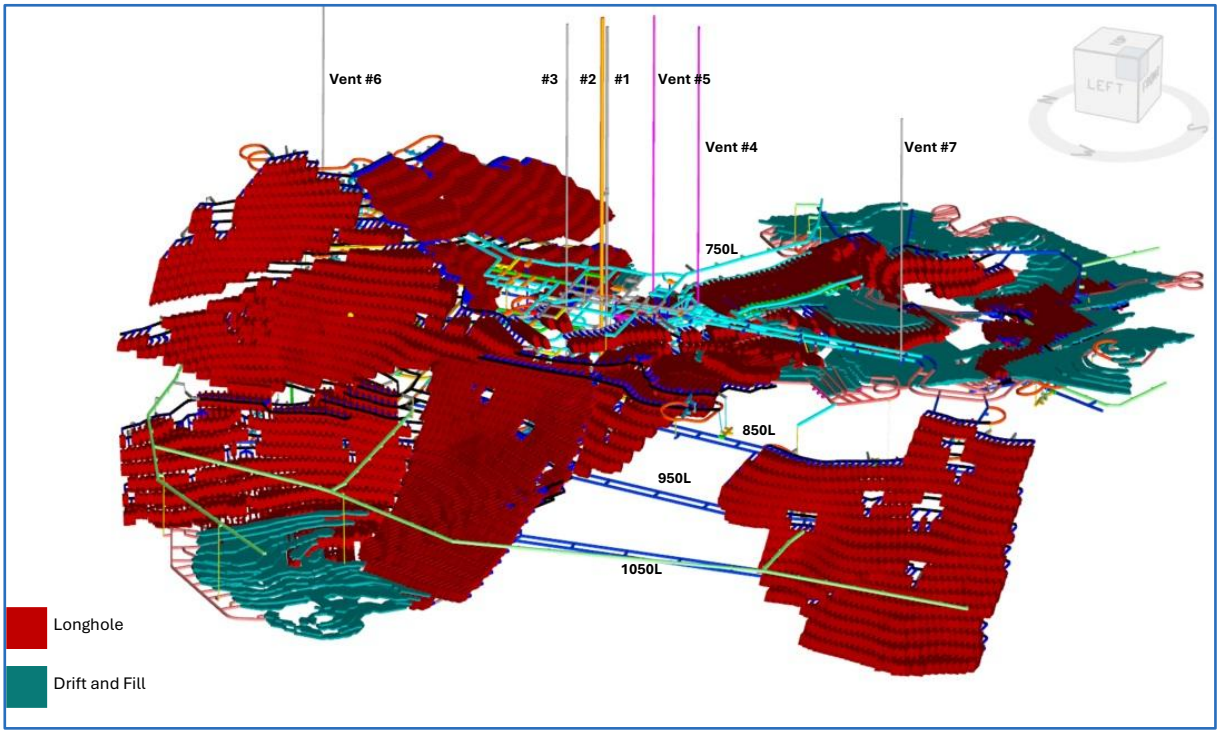
Figure 1.12 Mine Access Section



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 proceed in an overhand fashion. Figure 1.13 is an isometric view of the mining areas by method (looking north-east). Mining methods are explained in detail in Section 16.2.

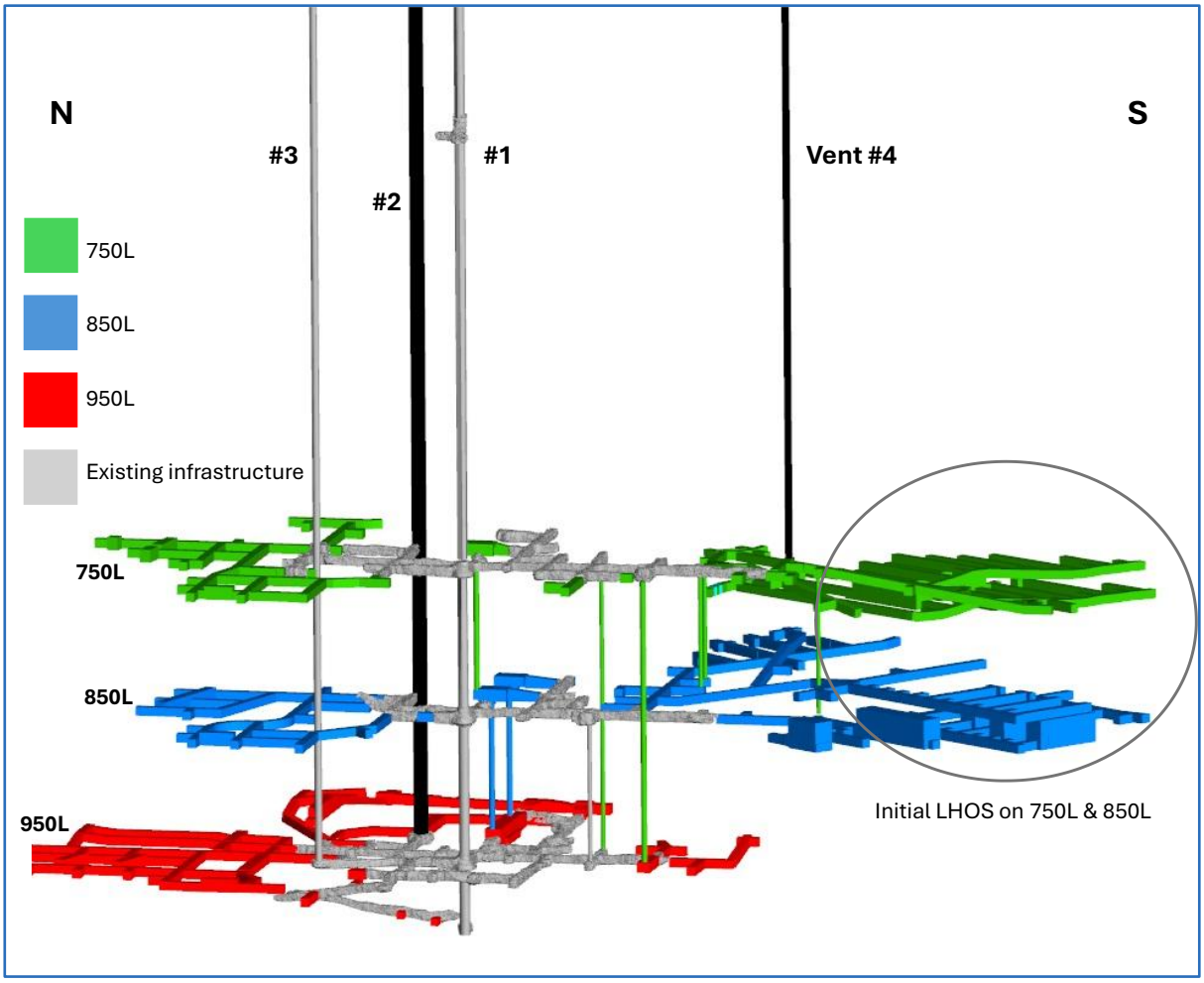
Figure 1.13 Elevated View of Mining Areas by Method



1.13.2 Development Plan

The completed and planned short-term development at Platreef Mine is shown in Figure 1.14.

Figure 1.14 Completed and Planned Short Term Development at Platreef Mine



The remaining mine development has been broken down into four main stages.

- Stage 1 – Lateral infrastructure development to support Shaft 3
  - This stage includes the holing from different elevations into Shaft 3, tips on all three levels, ore passes, ventilation passes, crusher excavation, conveyor excavation and loading level. Main restrictions are ground handling limitations and ventilation.
- Stage 2 – Lateral development to initial stopes and services infrastructure
  - The second stage is the development on 750 m and 850 m Levels to the initial stopes shown in Figure 1.14 and access to Shaft 4.
  - Development of initial workshops, emulsions bays, paste fill holes, dams and other services infrastructure before Shaft 3 is commissioned. The main constraints are ground handling, ventilation and logistics.
- Stage 3 – Lateral development after Shaft 3 commissioning

- The third stage is the ramp up and acceleration of level development off Shaft 1 and Shaft 3 until Shaft 2 is completed and functional. Additional development and longhole stoping crews are added to ramp up to 4 Mtpa. Shaft 1 is utilised for logistics only. This stage is limited by ground handling up to 2028 when 4 Mtpa is reached and the major constraint is logistics via Shaft 1.

- Stage 4 – Lateral development after Shaft 2 commissioned.
  - The fourth stage occurs after Q3'28 when Shaft 2 is commissioned for personnel and material. The remaining constraint now is ground handling with the combined hoisting capacity of Shaft 1 and 3 being 4.7 Mtpa.

1.13.3 Underground Infrastructure

Underground infrastructure entails dewatering, rock handling, 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 face dewatering, tertiary dewatering, secondary dewatering, and primary dewatering systems. 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 dams, transfer dams and settling dams. During mine production, water will be collected in sumps located throughout the mine. The gravity transfer dams will be equipped on the upper levels and 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 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 will be established.

Mine water inflow is estimated at 35 L/s during maximum production in the Platreef 2025 FS. The pumping system is designed for 150 L/s to account for spikes from initial groundwater inflows and paste backfill flushing 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

The ore and waste handling system will be constructed in distinct phases:

- Current Shaft 1: Relies on load haul dumps (LHDs) tipping directly into the shaft.
- Shaft 3 System Initial: Will introduce a total of four truck tips (2 x on 750 m Level and 2 x 850 m Level) with a crushing station on 950 m Level and a conveyor system feeding into a bin and loading system at Shaft 3.
- Shaft 3 System Future: The system described above will remain in operation, and additional conveyors and crushing stations will be added on 910 m Level, 950 m Level and 1,050 m Level.

1.13.5 Other Infrastructure Underground

Fleet maintenance will be done underground, and workshops will be constructed on each main level near 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 to 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 Level 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

Fresh, refrigerated air will be supplied via Shafts 1, 2 and 5 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. The ventilation system is based on a "once-through" system, eliminating the re-use of ventilation between drifts, mining levels and mining zones. Heat, dust and blasting fumes will be extracted directly to the main return system utilising a forced-exhaust secondary ventilation system. This ventilation strategy has major benefits on accumulated heat loads, airborne pollutants (such as diesel particulate matter (DPM)), re-entry times, mining flexibility and workplace conditions in general.

Refrigerated air will be supplied from surface bulk air coolers (BAC) situated in the sublevel drifts at Shaft 1 and Shaft 2. The refrigeration and cooling strategy is based on bulk air cooling and will not make use of underground chilled water. A detailed refrigeration study was conducted by others and is not included in this scope of work. The outcome of this study was used in the refrigeration design.

1.14 Metallurgical Testwork Overview

Various metallurgical testwork campaigns have been conducted since October 2001 on 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.

The Platreef metallurgical testwork campaigns can be summarised as follows:

- Prior to 2006: Testwork was predominantly conducted on lower grade platinum group element (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 HG PGE material.
- 2015 PFS: A series of metallurgical testwork campaigns were carried out on the Platreef mineralised material over the period 2010 to 2014. This, named 'Phase 1 to Phase 6' testing included comminution characterisation, bench-scale flotation testwork and laboratory scale dewatering testwork. The findings from Phase 1 to Phase 6 testing were presented in the Platreef 2015 PFS.

- 2017 FS: The 2015 to 2016 FS metallurgical testwork conducted as part of the 2017 FS was performed on approximately 1,286 kg of HQ drill core (comminution) and 1,140 kg of quarter PQ drill core (flotation) samples originating from the Platreef deposit. Samples were selected to represent the major ore types and ore blends expected to be processed over the life-of-mine (LOM). This, named 'Phase 7' testwork aimed to further characterise the flotation response, support flow sheet development, and to evaluate the degree of variability that could be expected across the deposit. The programme included comminution variability testing, mineralogical characterisation, open circuit flotation development and optimisation testwork, open circuit flotation variability testwork, bench-scale locked cycle flotation testwork and tailings dewatering testwork. The findings were presented in the Platreef 2017 DFS.
- 2022 PDP FS: Further testwork was conducted in 2020 and 2021 as part of the PDP FS. This, named 'Phase 8 and Phase 9' testwork aimed to evaluate the potential inclusion of a high-pressure grinding roll (HPGR) circuit, to further characterise the flotation response of Platreef composite samples, evaluate the potential for Jameson cell technology in the cleaner flotation circuit, determine the effect on flotation response when using site water and produce bulk concentrate samples for settling, filtration and Kell hydrometallurgical refining testwork. The PDP testwork was conducted on approximately 2.9 t of un-crushed drill core sample interval remainders from the 2017 FS variability testwork drilling campaign and two bulk samples totalling 47 t, comprised of crushed rock from surface stockpiles. The findings were presented in the Platreef 2022 PDP FS.
- 2025 IDP (Current): Further flotation and tailings dewatering testwork was conducted in 2022 and 2024. The programme included bench-scale flotation testwork on a 60 kg drill core composite sample remainder from the 2022 PDP FS test campaign which had been kept in cold storage. The aim was to further evaluate the potential to incorporate Jameson cell technology in the cleaner circuit. Additionally, bench-scale tailings settling and filtration testwork was conducted on the flotation tailings to further characterise the tailings dewatering circuit design and performance parameters.

Testwork has been conducted on samples representing the geometallurgical units T1 (feldspathic pyroxenite), T2U (poikilitic and/or pegmatoidal orthopyroxenite), T2L (typically olivine bearing and may vary from olivine-bearing orthopyroxenite to poikilitic harzburgite) as well as the footwall contaminated zone (CZ). The CZ zone has appropriately been reclassified as the footwall assimilated zone (FAZ). The majority of the testwork was conducted on drill core samples, however the programme also included testing of bulk samples from surface stockpiles for the 2022 PDP FS mini pilot plant (MPP) and HPGR test campaigns. The selection of samples, done in conjunction with the mining and geological teams, submitted for the metallurgical testwork are deemed to be sufficient.

Comminution testwork during both the 2015 PFS and 2017 FS indicated that the samples can be characterised as being hard to very hard with a Bond Ball Work Index (BBWi) in the range of 19.0–24.2 kWh/t. The Platreef material is thus not suitable for semi-autogenous grinding milling. The comminution variability testwork showed only minor variation in hardness. The abrasion index (Ai) tests indicated that the samples could be classified as having a medium abrasion tendency.

Multiple single-pass HPGR crushing tests by Thyssenkrupp using a semi-pilot scale HPGR (SMALLWAL) were conducted on a bulk shaft intercept composite sample sourced from surface stockpiles during the 2022 PDP FS testwork programme. The HPGR testwork confirmed the Platreef sample was amenable to HPGR achieving a specified throughput of 310–338 t.s/m^{3}h at a specific energy of 0.94–1.47 kWh/t.

Based on the testwork, a three-stage crushing circuit and ball mill circuit has been recommended as the basis of design. The crushing circuit is comprised of primary Jaw crushing (within mining scope), secondary cone crushing and a tertiary HPGR circuit. The 85th percentile comminution test data were used as the design basis for the crusher and grinding circuit. The average ore hardness and abrasion index data have been used to derive the operating cost estimate.

The flotation testwork programme, over various phases, has included bench-scale, flow-sheet development testwork, in combination with variability testwork (open circuit and locked cycle) using industry-standard testing techniques. Additionally preliminary Mini Pilot Plant (MPP) testwork was conducted to produce bulk concentrate for settling, filtration and Kell hydrometallurgical refining testwork.

Flotation testwork has confirmed an optimal flotation circuit feed size of 80% passing 75 µm and a flotation feed solids concentration of 30% w/w. Various flow sheet configurations and reagent regimes have been tested as follows:

- The 2015 PFS was based on an MF1 flow sheet utilising an oxalic acid and thiourea reagent regime and conventional cleaning technology.
- Further optimisation in the 2017 DFS confirmed the MF1 flow sheet using conventional cleaning technology, however the reagent regime was changed to a more conventional combination of thionocarbamate copper collector and sodium isopropyl xanthate (SIPX) as the primary collectors which is referred to as the copper collector reagent suite.
- Preliminary MPP runs during the 2022 PDP FS indicated that sodium isobutyl xanthate (SIBX) reagent is a potential alternative to the Cu collector reagent suite. This was also noted during the flow sheet development testing but was not adopted due to froth stability concerns. The mini pilot plant commissioning however, achieved adequate froth stability, but performance was not fully optimised and did not fully replicate the copper collector reagent suite performance.

The copper collector reagent regime has been more comprehensively tested and optimised and has thus been selected as the basis of design. The alternative SIBX reagent suite may be considered for further trials during full scale operation as a potential optimisation opportunity.

Bench-scale flotation testwork conducted at three independent laboratories confirmed that the type of grinding media used was critical, highlighting that the flotation feed Eh is negative when the sample is milled using carbon steel grinding media. A positive Eh prior to flotation was achieved by using high chrome grinding media which consistently returned superior flotation results compared to tests using carbon steel media.

Open circuit bench-scale flotation variability tests showed minor variation between the performance for the individual domain point samples (T1, T2L, T2U and CZ) and various ore blend composites. 3PE+Au recovery was however found to be dependent on the 3PE+Au head grade and target concentrate grade. For Platreef, as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). It was however also shown that under an excessively high mass pull regime, concentrate grades are significantly lower with limited opportunity for any gain in recovery. A 3PE+Au concentrate grade of 85 g/t could be achieved for head grades in the range 3.2–4.6 g/t where both the open circuit and locked cycle bench-scale variability testwork indicated that targeting higher concentrate grades in excess of 100 g/t (3PE+Au) resulted in reduced metal recoveries. Concentrate grades in excess of 100 g/t could only be achieved for head grades in the range 4.6–8.8 g/t.

Bench-scale Jameson cell testwork (open circuit and locked cycle) at XPS in 2022, has shown that a single Jameson cell is able to replicate or better the performance of two stages of conventional cleaning. Jameson flotation technology has thus been recommended for the final HG cleaner, MG cleaner and LG re-cleaner duties.

The impact of site water has also been evaluated where comparative testwork on two composite samples using filtered site water from the Masodi grey water system achieved a similar 3PE+Au upgrade profile as the baseline tests using laboratory tap water. There was, however, notable variance in the ranges for repeat baseline tests.

Optimised locked cycle flotation testing on a drill core sample blend composites with a 3PE+Au feed grade ranging from 2.4–4.9 g/t and averaging 3.8 g/t, achieved recoveries (3PE+Au) ranging from 83.1–88.7% at a final concentrate grade of 60–95 g/t. An average 3PE+Au recovery of 85.2% was achieved and the concentrate grade averaged 83 g/t 3PE+Au.

Mini pilot plant (MPP) testwork, was conducted at Mintek in June 2021 and November 2021 with the primary aim of producing bulk concentrate samples for Kell hydrometallurgical refining and concentrate de-watering testwork. Additionally, the intention was to derive process design information to supplement the design data derived from the bench-scale flotation testwork. The latter objectives were only partially achieved as the circuit was not adequately commissioned, stabilized and optimised due to several operational challenges. The results are deemed to reflect a non-optimised, preliminary commissioning performance achieving an average concentrate grade of 57 g/t at 87% recovery (2PE+Au) for the November 2021 run on a bulk sample with a head grade of 3.5 g/t 2PE+Au.

Based on the testwork findings, the flotation circuit flow sheet includes a rougher and cleaner circuit to treat the mill product (P_{80} of 75µm) for recovery of 3PE+Au, Ni and Cu minerals to the final concentrate. The cleaner flow sheet reflects a split configuration treating the fast, medium and slow-floating fractions separately and incorporates Jameson Cell dilution cleaning for the final cleaning duties. A scavenger circuit will treat the scavenger rougher concentrate and cleaner tailings to ensure optimal recovery.

Settling and filtration testwork on tailings and concentrates confirmed that the tailings and concentrate samples can be dewatered using conventional dewatering technologies.

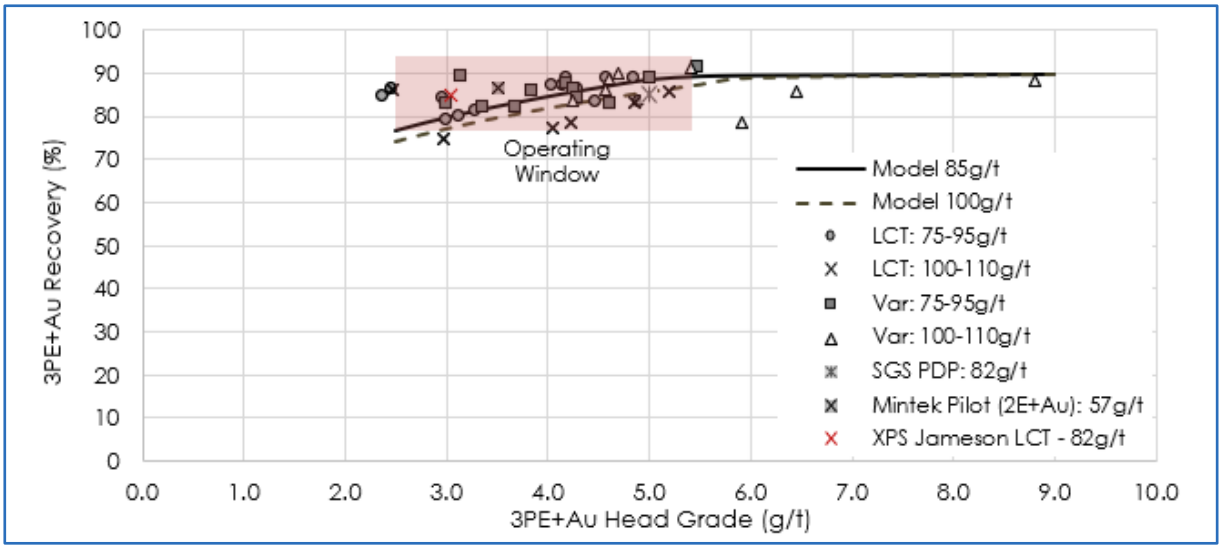
Settling testwork on tailings and concentrates recommended an optimal thickening flux of 0.30 t/m²h for the concentrate duty and 0.40–0.50 t/m²h for the tailings duty. A thickener underflow solids concentration of 60–65% w/w was achieved for both applications.

Concentrate pressure filtration testwork achieved a product moisture of <14% w/w with filtration fluxes averaging approximately 600 kg/m²h.

Tailings vacuum disc and vacuum belt filtration testwork achieved optimal filtration rates of 400–978 kg/m²/hr and a cake moisture content ranging from 16–23% w/w. Tailings pressure filtration tests produced filter cakes with a moisture content ranging from approximately 9–14% w/w and an optimal average filter flux of approximately 270–330 kg/m²h. Flow moisture point (FMP) and transportable moisture limit (TML) tests have also been conducted on uncemented dewatered tailings with the estimated values reported to be 14.8% w/w and 13.3% w/w respectively.

The 2017 DFS optimised locked cycle testwork using the copper collector reagent suite was used to derive relationships between 3PE +Au head grade, mass pull and final concentrate grade. These were used to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as reported for the 2017 DFS and summarised in Figure 1.15.

Figure 1.15 3PE+Au Recovery as a Function of Concentrate Grade



DRA, 2024

Since completion of the 2017 DFS, results for the additional open circuit and locked cycle flotation bench-scale testwork at SGS, Mintek and XPS over the period 2020–2022 were found to fall within the ranges evidenced in the 2017 DFS variability testwork. This shows consistency between the various test campaigns and performance modelling using the recovery algorithms previously developed for the 2017 DFS.

Updated metallurgical performance projections have been derived for the concentrator plant based on the IDP25 mine plan, using the recovery algorithms previously developed for the 2017 DFS. The metallurgical performance projections, when applied to the mine schedule, have indicated that when targeting a 3PE+Au concentrate grade of 85 g/t, the 3PE+Au recovery is expected to average 85.8% over the LOM, while the copper and nickel recoveries are expected to average 87.3% and 70.9%, respectively. The concentrate sulfur content is expected to range from 8–22% and average 12–15%.

1.15 Recovery Methods

Following on from the previous 2017 DFS and 2022 PDP FS, further progression of the Platreef mining and processing strategy for the current Platreef 2025 IDP has resulted in a production plan that incorporates a phased ramp up to ultimately target a combined processing capacity of 10.67 Mtpa.

The Phase 1 Project is currently being executed and includes the construction of Shaft 1 and a 0.77 Mtpa concentrator. The 0.77 Mtpa Phase 1 concentrator incorporates a conventional three-stage crushing and ball milling circuit followed by flotation (MF1). The design is as per the 2022 PDP FS but with the inclusion of Jameson Cell dilution cleaning for the final flotation cleaning duties, aligned to testwork findings.

Phase 1 will be followed by the addition of 3.3 Mtpa of processing capacity in Phase 2 to target a combined concentrator production rate of 4.07 Mtpa. The Phase 2 Project, currently at FS level, includes the construction of Shaft 3 and Shaft 2 (personnel and material only) in combination with a concentrator plant capacity expansion.

The Phase 2 concentrator design includes all processing requirements from receipt of the ROM material from stockpiles and or ROM silo through to final concentrate load out and production of filtered tailings. The handling of filtered tailings and DTSF operation is not within the concentrator scope. Phase 2 will include the addition of:

- A new 4.07 Mtpa crushing circuit incorporating secondary-cone crushing and a tertiary HPGR. The new Phase 2 crushing circuit will feed both the Phase 1 and Phase 2 milling circuit modules, and the Phase 1 crushing circuit will become redundant.
- A new 3.3 Mtpa milling and flotation module comprised of two stages of ball milling in series and flotation (MMF).
- A new 4.07 Mtpa tailings dewatering and pumping circuit in combination with a tailing's filtration plant located at the Rietfontein Dry Tailings Storage Facility (DTSF). The new Phase 2 tailings filtration circuit will process the combined tailings, and the Phase 1 tailings filtration circuit will become redundant.

The process design has been developed based on the testwork findings and assessments, various desktop level trade-off studies and design information from the historical study phases in combination with relevant DRA design information. The flow sheet is well known in industry and has historically been proven as a suitable processing route for various platinum ores.

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

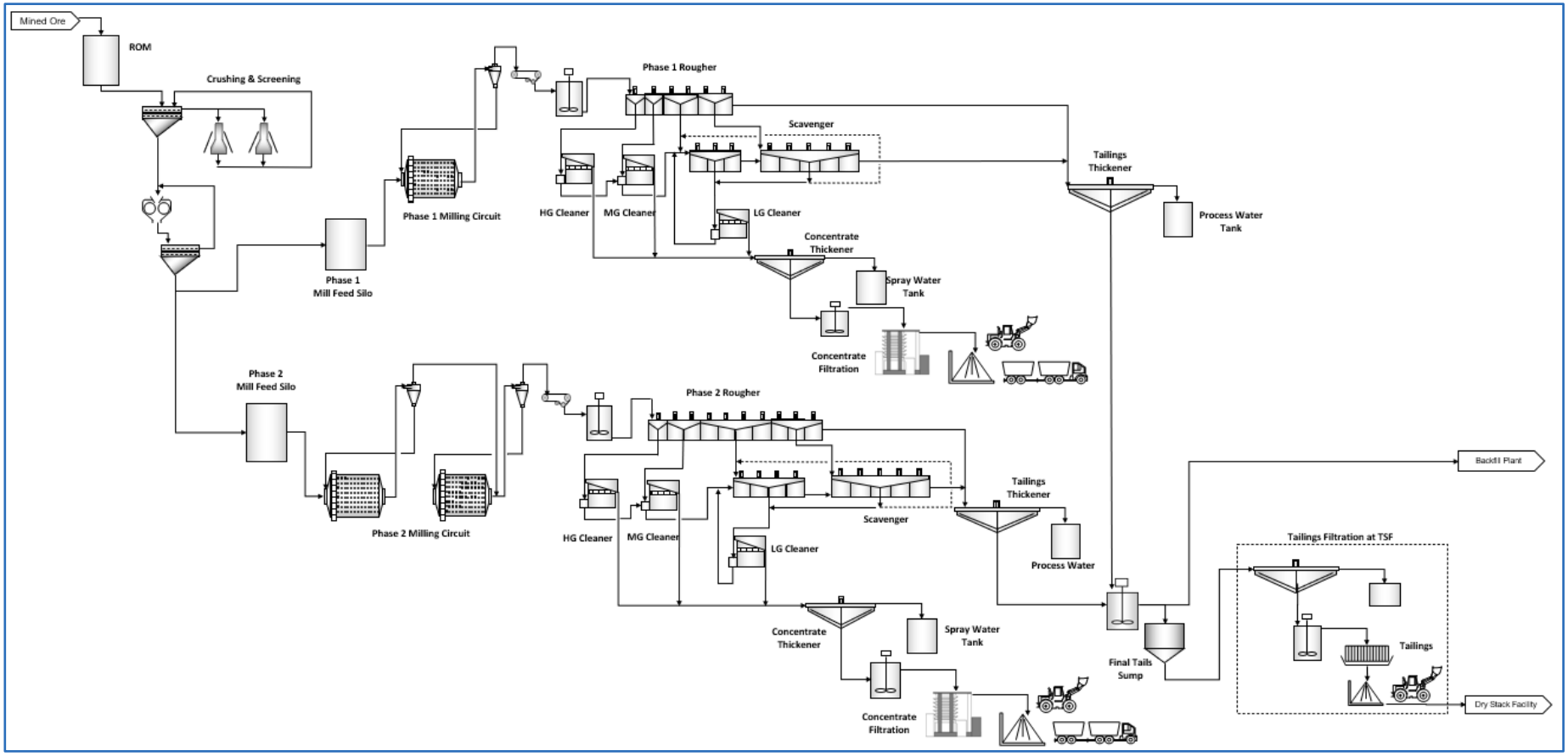
The engineering design has taken cognisance of the environmental and social impacts regarding noise, dust, light, vibration and visual pollution. Dust suppression is present on all ROM transfer points. Silos are specified with concrete roofs and the Phase 2 conveyors are sheeted to minimise noise, dust, and light pollution.

The final concentrator layouts were selected, 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 Phase 2 concentrator plant has been designed in accordance with the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities.

The future Project Phases 3 and 4, currently at PEA level, will aim to further expand production to 10.67. This will include bringing Shaft 2 (Rock hoisting) online in combination with expansion of the crushing circuit and the addition of two 3.3 Mtpa concentrator milling and flotation modules which will be identical to the Phase 2 module. The concentrator tailings dewatering and pumping circuit capacity will also be expanded and two additional tailings filtration modules (located at the DTSF) will be installed, one each in Phases 3 and 4 respectively.

Figure 1.16 4.07 Mtpa Phase 2 Simplified Concentrator Flow Sheet



DRA, 2025

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 1.17 shows the Platreef Project location within local region.

Figure 1.17 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 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.

On 17 January 2022, Ivanhoe concluded an agreement with the Mogalakwena Local Municipality to receive local treated effluent water from the Masodi WWTW to supply most of the bulk water needed. A minimum of 3,000 m3/d will be supplied for the first phase of production at Platreef. A further agreement was signed between Ivanplats and the Mogalakwena Local Municipality on 15 June 2023 to increase the supply from the Masodi WWTP up to a maximum of 10,000 m3/d of treated water, depending on the roll out of the supporting municipal infrastructure. The associated pumps at Masodi WWTP and piping infrastructure to the Platreef site have been installed as part of the Phase 1 construction.

For the operational phase of the mine, an updated dynamic daily time step water balance model was applied to develop an average water balance for Phase 2 (January 2028 to LOM). The simulation shows that on average 9,809 m3/d water is required to operate the mine. The sources of water are an average of 5,602 m3/d bulk water supply from the Masodi WWTW to supply the site filtered water requirements, 222 m3/d from the four site boreholes to supply the site operational potable water requirements and the remaining 3,985 m3/d is supplied from moisture in the ore, from rainfall/run-off collected on site in the stormwater ponds and groundwater ingress to the workings.

The bulk water requirements will increase to approximately 6,800 m3/d as the Phase 2 concentrator ramps up. The variation in the make-up volume required is due to the seasonality and annual variation of the rainfall. The reduction in the water requirement occurs during the wet season and in wet years when the rain and run-off volumes are available on site to meet the filtered water requirements.

Provision has been made to further treat the water received from the Masodi WWTW via an on-site filtration plant for use in the process plant.

1.16.2 Project Power Supply

The bulk power supply is sourced from Eskom, the South African national power utility and this is installed as Phase 1 of the Project and was commissioned in August 2024. The existing NMD is 100 MVA. The estimated NMD will increase to 110 MVA in the year 2032, and an application to increase the NMD will be required in future.

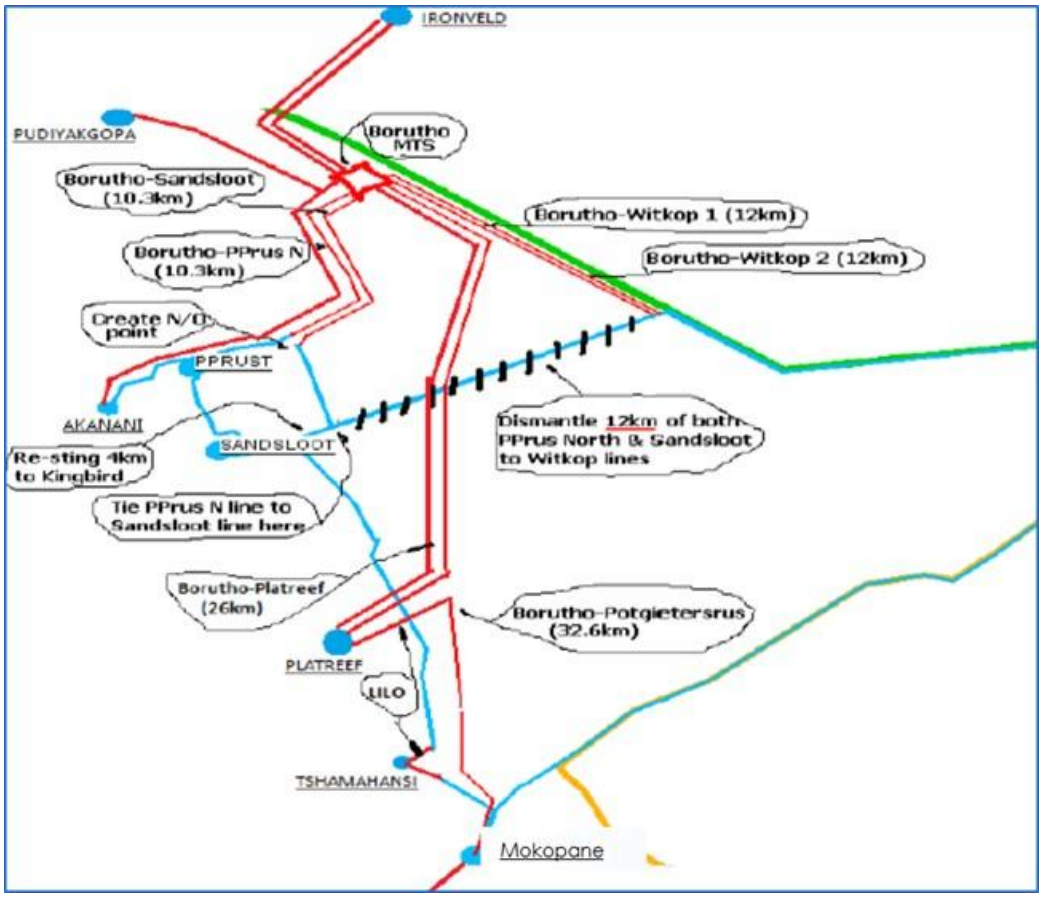
The installed capacity in the Eskom 132/33 kV outdoor substation is four40 MVA 132/33 kV transformers. The outdoor Eskom 132/33 kV substation is complete, but only the 33 kV circuit breakers required for Phase 1 are installed in the 33 kV consumer substation, with space provision for the future circuit breakers. The additional 33 kV circuit breakers and cable installations are allowed for as part of this study. The design capacity is 120 MVA, with the fourth 40 MVA transformer for redundancy.

The 132/33 kV substation is supplied from Eskom Borutho main transmission station via two 132 kV overhead lines (OHLs). Each OHL is rated for 175 MVA for full redundancy.

In addition to the Eskom bulk power agreement, Ivanplats has entered into a 10 year agreement with an independent power producer (IPP) for a minimum commitment of 48,300 MWh per annum of solar power. The solar power will be wheeled through the Eskom network. The reduction in the cost for the solar power is allowed for in the electrical operating cost. The scheduled start date for the wheeling is June 2026.

The construction of an initial on-site 9,589 MWh per annum solar power plant is complete and was commissioned in February 2025. The cost savings due to the on-site solar power plant is included in the electrical operating cost.

Figure 1.18 Proposed Power Transmission



DRA, 2017

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 new dry stacked tailings storage facility (DTSF) 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 9 Mt of tailings and the tailings generated by the 770 ktpa concentrator plant will report and stored here, until the Rietfontein facility is commissioned.

The proposed tailings storage facility (DTSF) site for the second phase is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed DTSF site is considered a feasible site, considering all applicable engineering and environmental standards for tailings storage facilities. The DTSF will have an operating life of 36 years, during this time approximately 90.4 Mt of tailings will be stored within the DTSF. The remainder of the tailings will be used as backfill in the underground mine. The DTSF 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 67% on average.

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. Dry stack 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 Reifontein DTSF will be constructed in a phased approach to limit upfront capital expenditure. The DTSF will be constructed in two lifts as follows:

- The first lift will accommodate tailings for the first 12 years of deposition at the site and will be comprised of three portions, namely Portion 1, Portion 2 and Portion 3.
- The second lift will accommodate tailings from Year 13 to the end of life-of-mine and will be constructed in a single phase, referred to as Portion 4.

Tailings will be delivered to the filter plant situated at the stacking facility utilising the same pumping systems from the processing plant. Dewatered tailings will be delivered to the DTSF using conveyors.

Aside from the borrow material used in the starter dam and drainage infrastructure, which includes a Pollution Control Dam, the facility will be developed and operated using dewatered tailings as a dry stack facility. A liner system with leak detection will be in place upon start-up.

It is envisioned that progressive rehabilitation will be executed during operations of the DTSF. The operational side slopes will be shaped to the final rehabilitated slope of 1V:4H and a capping layer will be constructed as part of progressive rehabilitation during operations, followed by placement of a growth medium and the establishment of vegetation. 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 770 ktpa plant. This dewatered and filtered tailings stream will be hauled and placed on the initial phase 1 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. WSP 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 combine 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 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.

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 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.

1.18.1 Permitting Status

Ivanplats conducted an amendment of some of the existing authorisations and licences to facilitate the aspects of Phase 2 of the Platreef Project in 2022. This process required a full ESIA process before commencing with the proposed changes. Subsequently, following the amendment process, in 2023, an integrated EA was granted by the Department of Mineral Resources and Energy (DMRE) and an amended WUL and secondary WUL by the Department of Water and Sanitation (DWS).

- The following changes informed the amendment process as completed:
  - Changes to the infrastructure plan for Phase 1, such as:
  - The inclusion of the Masodi Waste Water Treatment Works (WWTW) pipeline.
  - A change of the existing authorised waste rock footprint to an on-site dry tailing storage facility (DTSF).
  - Repositioning of the waste rock footprint.
  - Establishment of an ore stockpile (run-of-mine).
  - Construction and operation of a 700 ktpa crushing and screening plant.
  - Expansion of stormwater management systems, adding two additional dams to service the DTSF and concentrator plant areas.
  - Expansion of the existing diesel based electricity generating capacity to more than 20 MW.
  - The change of the authorised upstream conventional tailings storage facility (TSF) (Rietfontein KS) to a DTSF (In progress).
  - The backfilling with residue material into the mine's underground voids.
- Changes to the Phase 2 infrastructure such as the inclusion of an additional cut-off drain for the existing concentrator plant, the south-western clean stormwater management drain expanded further down to the Mogalakwena River and the upgrading of internal haul roads.
- These amendments required a full ESIA process prior to the commencement of these activities.
- Additional specialist studies and public participation have been undertaken to inform the applications for these amendments, which were authorised in 2023.
- In an effort to reduce the Mine's carbon footprint and to ensure continued electricity supply for the Platreef operations, Ivanplats investigated alternative energy resources for the operations. Various options were investigated, and the development of a 19 MW solar photovoltaic (PV) facility was selected as the most feasible option. A basic ESIA authorisation process was conducted to attain the environmental license to develop the facility, with the associated electrical infrastructure, in 2022, with an authorisation granted by the DMRE in 2023.
- The solar facility development will be undertaken in a phased approach, commencing with a 5 MW supply as Phase 1 and forecast operational in Q1'25.

1.18.2 Social and Public Participation

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

1.18.3 Community

The Platreef Project will contribute to the local economy through both direct and indirect employment opportunities that 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 re-occurrence 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 integrated EA, EMP and WUL conditions will be undertaken in accordance with the authorisations and submitted to the relevant authorities.

1.19 Platreef Integrated Development Plan 2025

1.19.1 Platreef 2025 FS Overview

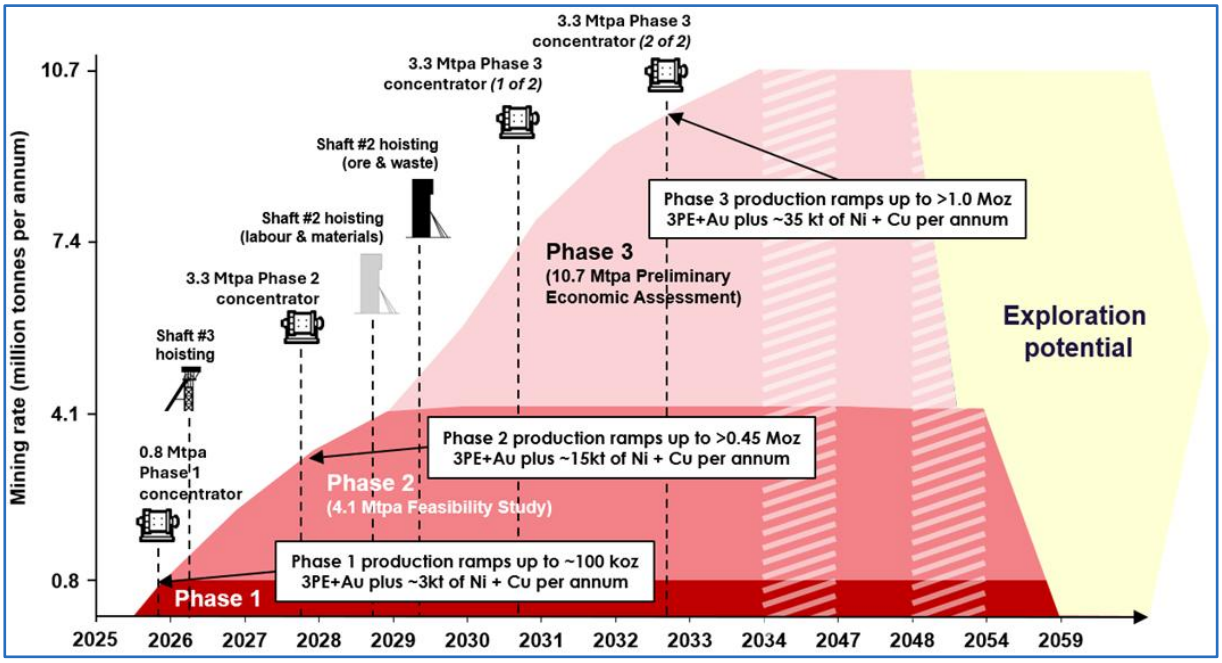
The Platreef 2025 FS includes two main phases. Phase 1 is defined as establishing Shaft 1 remaining infrastructure, development and other infrastructure required to sustain 770 ktpa. Phase 2 commences just after equipping Shaft 3 when mine production begins to ramp up to 4.07 Mtpa and Plant 2 is commissioned. The study outlines Phase 1 production from Q4'25, followed by the Phase 2 expansion 2 years later in Q4'27.

Key steps involved in preparing the Platreef 2025 FS are as follows:

- Phase 1 concentrator 770 ktpa was completed in June 2024 and Shaft 1 commissioned for hoisting labour and material in 2022. Therefore, the first feed of ore is expected to commence in Q4'25.
- Phase 2 new 3.3 Mtpa concentrator module built alongside the Phase 1 module, increasing the processing capacity to 4.1 Mtpa.
- Shaft 3 will provide an additional 4 Mtpa of hoisting capacity (ore and waste rock).
- Shaft 2 will initially be used to hoist labour and materials during support Phase 1s and 2.

The Platreef 2025 FS mining method selection focused on mechanised mining with high productivities. Mine design work aimed to maximise production grades and reduce operating costs. The 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. The Platreef IDP 2025 development and production timeline schematic is shown in Figure 1.19.

Figure 1.19 Platreef IDP 2025 Development and Production Timeline



The Platreef 2025 FS has an estimated life of mine (LOM) total cash cost for the 4.1 Mtpa FS of $599 per ounce (oz) of platinum, palladium, rhodium, and gold (3PE+Au), net of nickel and copper by-product credits. The Platreef 2025 PEA has lower LOM total cash costs fall to $511 per oz.

The key features of the Platreef 2025 FS include:

- Development of a large, mechanised, underground mine is planned at an initial 770 ktpa and expansion to 4.07 Mtpa.
- Planned average annual production rate of 432 koz of platinum, palladium, rhodium and gold (3PE+Au).
- Estimated initial capital requirement of approximately $1,243M.
- After-Tax NPV of $1,393M, at an 8% discount rate.
- After-Tax IRR of 19.77%.
- The Platreef 2025 FS maintains options available to accelerate expansions, to the 10.7 Mtpa scenario, as the market dictates.

The key dates for the Platreef 2025 FS are summarised in Table 1.5.

Table 1.5 Platreef 2025 FS Key Dates

Activity Name	Start	Finish
Shaft 3 Equipping	Q4'24	Q1'26
Shaft 2 Sinking to –60 m Level	Q4'25	Q1'26
First Concentrator Start Up		Q3'25
Shaft 2 Sinking to –995 m Level	Q1'26	Q1'28
Shaft 2 Equipping Complete		Q3'28
Plant 2 Start Up		Q4'27
Mine Production Steady State (4.07 Mtpa)	Q3'29	

1.19.2 Platreef 2025 FS Production Summary

Sublevel blasthole (longhole) stoping with cemented paste fill, supplemented with development waste rock in secondary stopes, 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.

The mine production forecast is shown in Figure 1.20 and the key average annual production results over the planned life of the mine are shown in Table 1.6. Concentrator feed for the life-of-mine are depicted in Figure 1.21.

Figure 1.20 Platreef 2025 FS Mining Production

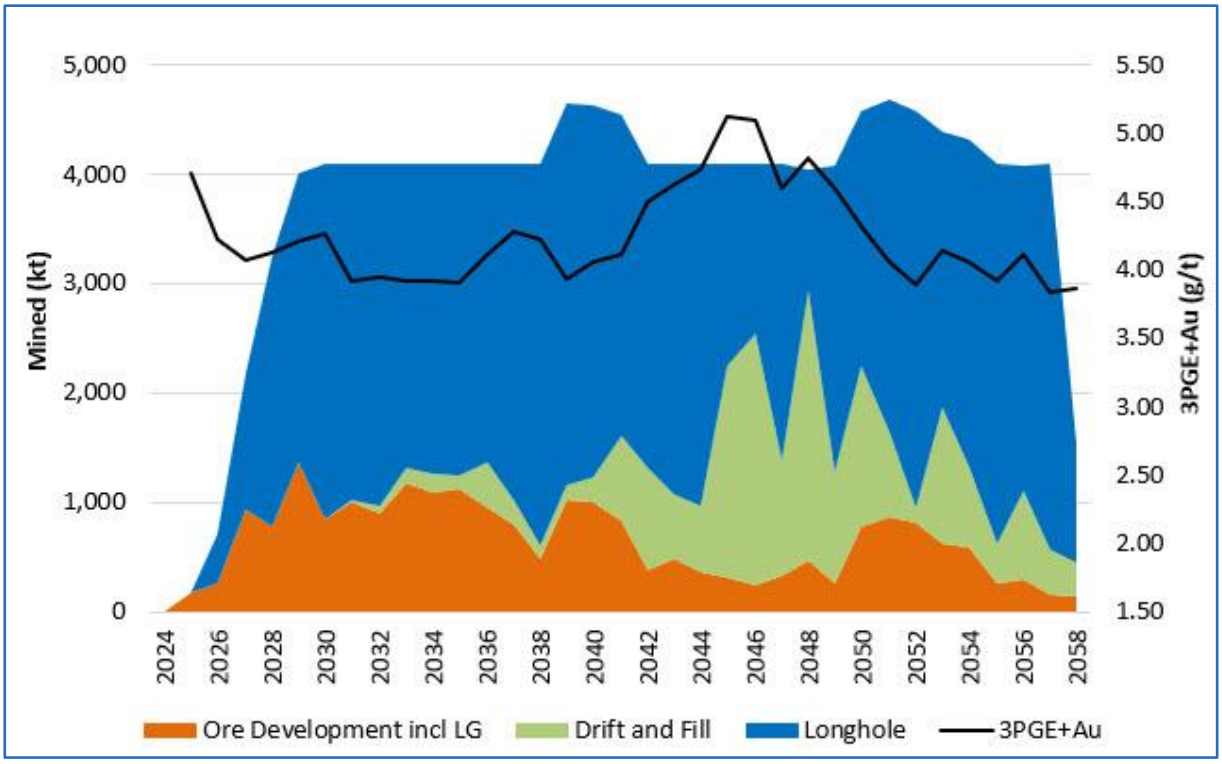
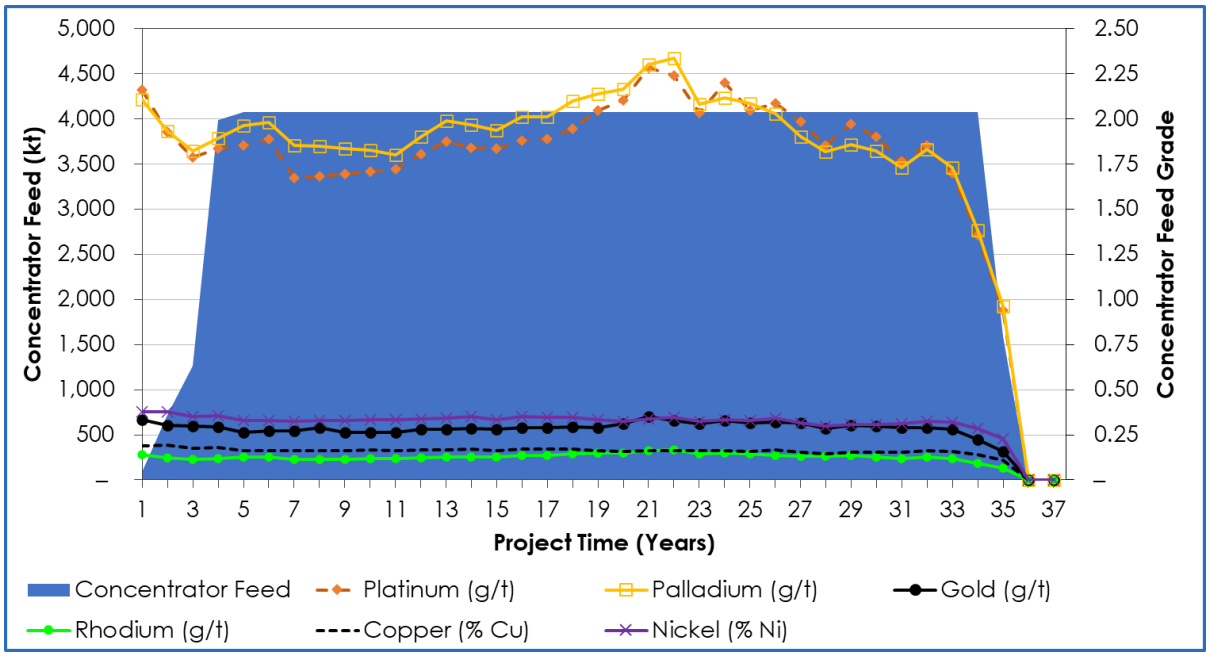


Table 1.6 Platreef 2025 FS Average Annual Production Forecast

Item	Unit	LOM Average
Mined and Processed	Mtpa	3.71
Platinum	g/t	1.88
Palladium	g/t	1.93
Gold	g/t	0.29
Rhodium	g/t	0.13
3PE+Au	g/t	4.22
Copper	%	0.16
Nickel	%	0.33
Concentrator Recoveries		
Platinum	%	86.9
Palladium	%	86.3
Gold	%	78.3
Rhodium	%	79.8
Copper	%	87.3
Nickel	%	70.8
Concentrate Produced	ktpa	158
Platinum	g/t	38.3
Palladium	g/t	39.1
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE+Au	g/t	85.1
Copper	%	3.3
Nickel	%	5.5
Contained Metal		
Platinum	kozpa	194
Palladium	kozpa	198
Gold	kozpa	27
Rhodium	kozpa	12
3PE+Au	kozpa	432
Copper	Mlbpa	12
Nickel	Mlbpa	19

1. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.
2. Production over 35 years.

Figure 1.21 Platreef 2025 FS Concentrator Production



OreWin, 2025

1.19.3 Platreef 2025 FS Capital and Operating Cost Summary

The total initial (pre-production) expansion and sustaining capital costs required, including contingency, are shown in Table 1.7.

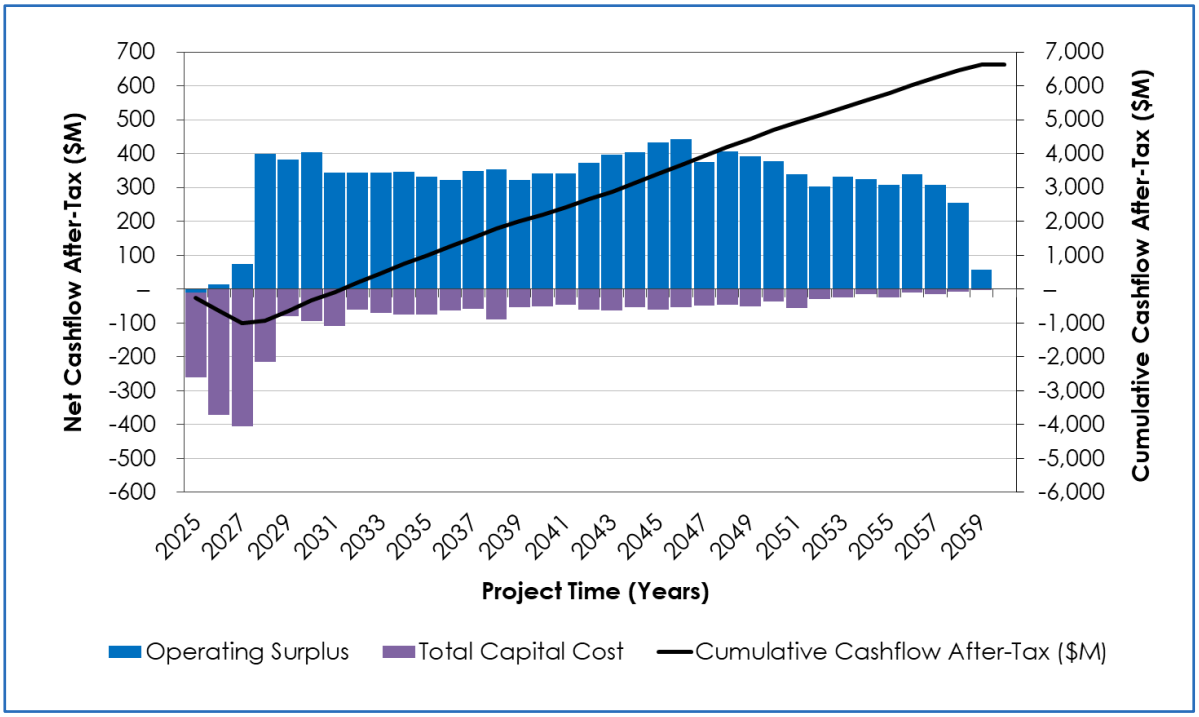
Table 1.7 Platreef 2025 FS Capital Expenditure Summaries

Description	Initial & Expansion Capital ($M)	Sustaining Capital ($M)	Total Capital LOM ($M)
Mining			
Mining	581	1,364	1,944
Capitalised Operating Costs	75	0	75
Subtotal	**655**	**1,364**	**2,019**
Concentrator and Infrastructure	**373**	**104**	**477**
Other Costs			
Owner's Cost	95	34	129
Closure Cost	0	10	10
Capitalised Operating Costs	20	0	20
Subtotal	**116**	**44**	**159**
Capex Before Contingency	**1,144**	**1,511**	**2,655**
Contingency	99	78	177
Capex After Contingency	**1,243**	**1,589**	**2,832**

1. Initial capital reflects the capital costs from 1 January 2025, to achieve initial production from the 770 ktpa Phase 1 concentrator, followed by expansion capital to achieve nameplate Phase 2 production of the 4.1 Mtpa FS.
2. Remaining capital cost for Phase 1 is $70M.
3. Totals may vary due to rounding.

The Net Cash Flow After-Tax and the Cumulative Cash Flow After-Tax is shown in Figure 1.22.

Figure 1.22 Platreef 2025 FS Cumulative Cash Flow After-Tax



OreWin, 2025

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.

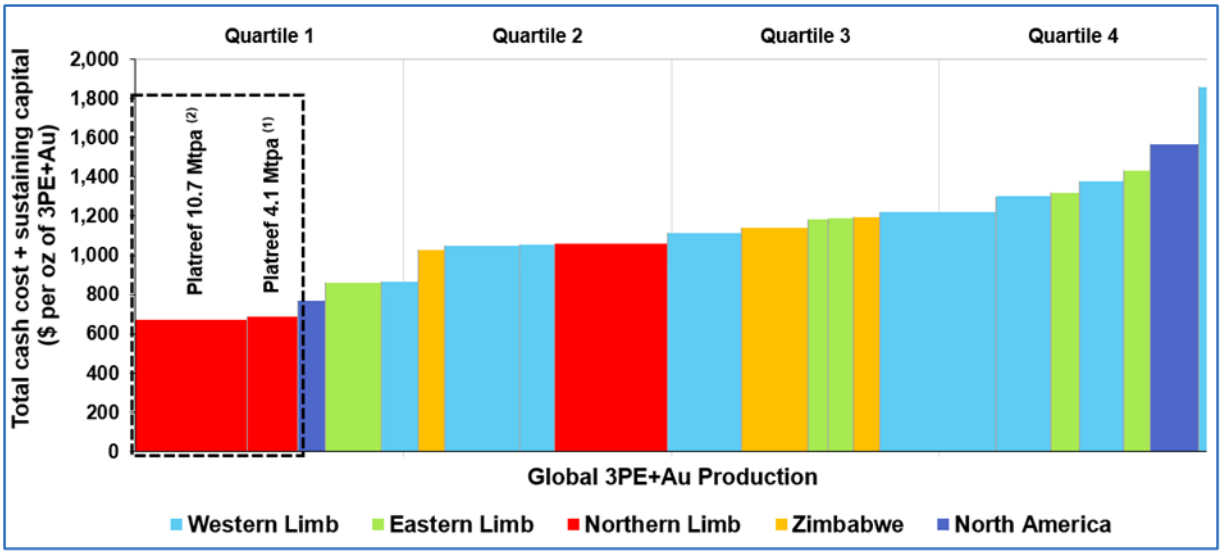
Higher nickel and copper grades contribute to lower cash costs for operations on the Northern Limb as illustrated by Figure 1.23. Among global primary platinum-group-metals producers, Platreef's estimated net total cash cost of $704/oz 3PE+Au, net of copper and nickel by-product credits and including stay-in-business (SIB) capital costs, ranks at the bottom of the cash-cost curve.

Table 1.8 Platreef 2025 FS Cash Costs After Credits

Item	Recovered 3PE+Au ($/oz)		
	Stage 1 2025 - 2027 Average	Stage 2 2028 - 2059 Average	LOM 2025-2059 Average
Mine Site Cash Costs	1,010	508	516
Transport	8	8	8
Treatment & Refining	393	366	367
Au, Pd & Pt Streaming Agreements	156	118	119
Royalties	8	78	77
Total Cash Costs Before Credits	**1,575**	**1,078**	**1,086**
Nickel Credits	424	373	374
Copper Credits	151	113	114
Total Cash Costs After Credits	**1,000**	**592**	**599**
Sustaining Capital Costs	–	108	105
Total Cash Costs After Credits & Sustaining Capital	**1,000**	**700**	**704**

Totals may vary due to rounding.

Figure 1.23 Global Primary Producers' Net Total Cash Cost + Sustaining Capital (2024E) US$/3PE+Au oz



SFA (Oxford), Ivanplats. Notes: Cost and production data for the Platreef project is based on the Platreef's 2025 4.1 Mtpa FS and 10.7 Mtpa PEA parameters, applying SFA South African industry average smelting and refining costs. SFA's estimated peer group cost and production data for 2024 is based on H1 2024 figures, extrapolated out to produce an estimate for the full calendar year and 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 2024 average basket prices and exchange rates of 18.78:1 ZAR:USD, US$980/oz platinum, US$1,009/oz palladium, US$4,753/oz rhodium, US$2,300/oz gold, US$17,150/t nickel and US$8,727/t copper. (1) Platreef 4.1 Mtpa between years 4 to 35. (2) Platreef 10.7 Mtpa between years 4 to 29.

Table 1.9 Platreef 2025 FS Operating Costs and Revenues

Item	LOM Total ($M)	Milled ($/t)		
		Stage 1 2025 – 2027 Average	Stage 2 2028 – 2059 Average	LOM 2025-2059 Average
Gross Sales Revenue	**27,590**	**216.43**	**212.59**	**212.65**
Less: Realisation Costs				
Transport	117	0.89	0.90	0.90
Treatment and Refining	5,544	44.38	42.70	42.73
Au, Pd & Pt Streaming Agreements	1,793	17.56	13.76	13.82
Royalties	1,170	0.86	9.15	9.02
Total Realisation Costs	**8,624**	**63.70**	**66.52**	**66.47**
Net Sales Revenue	**18,965**	**152.74**	**146.07**	**146.18**
Site Operating Costs				
Mining	4,922	62.84	37.53	37.94
Processing and Tailings	1,789	18.43	13.71	13.79
Infrastructure	363	6.28	2.74	2.80
General and Administration	722	26.40	5.23	5.57
Total	**7,796**	**113.95**	**59.22**	**60.09**
Operating Margin	**11,170**	**38.78**	**86.86**	**86.09**
Operating Margin (%)	**40%**	**18%**	**41%**	**40%**

Totals may vary due to rounding.

1.19.4 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2025. It is presented in 2025 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 2025 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 2025 FS are shown in Table 1.10.

Table 1.10 Platreef 2025 FS Commodity Price Assumptions

Parameter	Unit	2025	2026	2027	Long Term
Platinum	$/oz	1,130	1,200	1,240	1,200
Palladium	$/oz	1,090	1,110	1,090	1,130
Gold	$/oz	2,550	2,480	2,400	2,170
Rhodium	$/oz	4,670	5,000	5,000	5,000
Copper	$/lb	4.60	4.70	4.80	4.25
Nickel	$/lb	8.10	8.30	8.10	8.50

1.19.5 Treatment Charges and Refining Charges

In the Platreef 2025 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 Table 1.11.

Table 1.11 Platreef 2025 FS Refining Charges (% Gross Sales)

Parameter	LOM
Copper	28.8%
Nickel	30.0%
Platinum	16.5%
Palladium	16.5%
Gold	16.5%
Rhodium	17.3%

1.19.6 Concentrate Transport Costs

In the Platreef 2025 FS the average distance of the smelters is 277 km. The transport cost applied to the financial model is $0.07 per wet tonne concentrate per km.

1.19.7 Royalties and Taxes

The majority of taxes and fees payable to the government under Republic of South Africa legislation are the Corporate Income Tax (27%) 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 2025 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	%	27.00
Opening Tax Losses	ZAR million	129
Opening Depreciation	ZAR million	21,483
Working Capital		
Receivables	weeks	15.00
Payables	weeks	4.00

1.19.8 Exchange Rates

Costs estimated in ZAR have been converted to US dollars at an exchange rate of 18.5 ZAR/USD. A comparison between the exchange rates used in the Platreef 2025 FS and the 2022 FS is shown in Table 1.13.

Table 1.13 Exchange Rates

Exchange Rates	Forex Amount	
	2025 FS	**2022 FS**
ZAR	18.5	16.00
EUR	0.71	0.71
AUD	1.18	1.18
CNY	5.70	5.70
GBP	0.63	0.63
JPY	100.00	100.00
NOK	7.00	7.00
SEK	7.00	7.00

1.19.9 Project Results

The results of the financial analysis show an After-Tax NPV8% of $1,393M. The Platreef 2025 FS exhibits an After-Tax IRR of 19.77% and a payback period of 7.8 years. The estimates of cash flows have been prepared on a real basis as at 1 January 2025 and a midyear 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 2025 FS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	8,338	6,628
	5.0%	3,001	2,450
	8.0%	1,689	1,393
	10.0%	1,150	950
	12.0%	769	632
	15.0%	386	306
	20.0%	24	-11
Internal Rate of Return	–	20.48%	19.77%
Project Payback Period (Years)	–	7.80	7.80

1.19.10 Summary of Platreef 2025 FS

The key features of the Platreef 2025 FS include:

- Development of a large, mechanised, underground mine is planned at an initial processing rate of 770 ktpa with expansion to 4.07 Mtpa.
- Planned average annual production rate of 432 koz of platinum, palladium, rhodium and gold (3PE+Au).
- Estimated initial capital requirement of approximately $1,243M.
- After-Tax NPV of $1,393M, at an 8% discount rate.
- After-Tax IRR of 19.77%.
- The Platreef 2025 FS maintains options available to accelerate expansions, to the 10.7 Mtpa scenario, as the market dictates.

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

Table 1.15 Platreef 2025 FS Key Production and Financial Results

Item	Unit	Total
Mined and Processed	Mt	130
Platinum	g/t	1.88
Palladium	g/t	1.93
Gold	g/t	0.29
Rhodium	g/t	0.13
3PE+Au	g/t	4.22
Copper	%	0.16
Nickel	%	0.33
Concentrate Produced	kt	5,523
Platinum	g/t	38.28
Palladium	g/t	39.11
Gold	g/t	5.33
Rhodium	g/t	2.43
3PE+Au	g/t	85.15
Copper	%	3.31
Nickel	%	5.47
Contained Metal		
Platinum	koz	6,797
Palladium	koz	6,944
Gold	koz	947
Rhodium	koz	432
3PE+Au	koz	15,120
Copper	Mlb	403
Nickel	Mlb	666
Key Financial Results		
Life-of-Mine	Years	35
Initial and Expansion Capital	$M	1,243
Sustaining Capital	$M	1,589
Total Cash Costs After Credits	$/oz Rec. 3PE+Au	599
Total Cash Costs After Credits, Expansion and Sustaining Capex	$/oz Rec. 3PE+Au	704
Site Operating Costs	$/t Milled	60.09
After-Tax NPV8%	$M	1,393
After-Tax IRR	%	19.77
Project Payback Period	Years	7.8

1. Initial capital includes a remaining $70 million for the completion of Phase 1.
2. Totals may vary due to rounding.
3. 3PE+Au = platinum, palladium, rhodium and gold.
4. Economic analysis metal price assumptions: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.

1.19.11 Summary of Platreef 2025 PEA

The 10.7 Mtpa Platreef 2025 PEA considers the Phase 3 expansion once the major Shaft #2 is available for hoisting in 2029. This expansion further doubles the annualized platinum, palladium, rhodium, and gold production to over 1 million ounces of 3PE+Au, plus approximately 22,000 tonnes of nickel and 13,000 tonnes of copper. The 10.7 Mtpa PEA generates an NPV8% of $3.2 billion, and an IRR of 25%, at consensus long-term metal prices.

The Platreef 2025 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

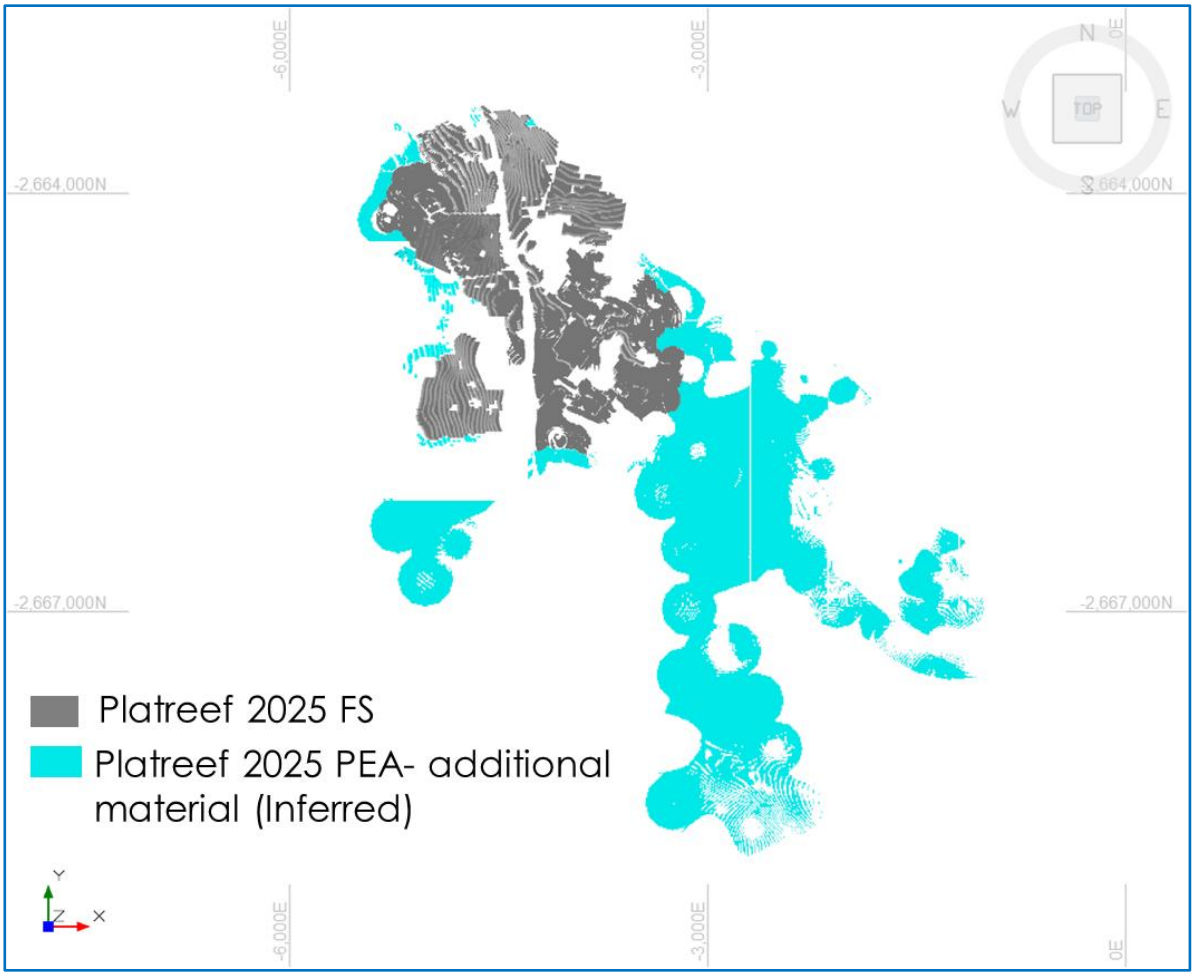
The 10.7 Mtpa Platreef 2025 PEA uses Shaft #2 and Shaft #3 for hoisting ore and waste with a combined total capacity of over 12 Mtpa. The incremental expansion capital cost for the 10.7 Mtpa PEA is estimated at $803 million, leveraging the significant surface and underground infrastructure already constructed during Phase 2. The 10.7 Mtpa PEA delivers an NPV8% of $3.2 billion and an IRR of 25%, based on long-term consensus prices over a mine life of 29 years.

Key steps involved in preparing the Platreef 2025 PEA are as follows:

- Phase 1 concentrator 770 ktpa was completed in June 2024 and Shaft 1 commissioned for hoisting labour and material in 2022. Therefore, the first feed of ore is expected to commence in Q4'25.
- Phase 2 new 3.3 Mtpa concentrator module built alongside the Phase 1 module, increasing the processing capacity to 4.1 Mtpa.
- Shaft 3 will provide an additional 4 Mtpa of hoisting capacity (ore and waste rock).
- Shaft 2 will support Phase 1 and 2 by commence first with hoisting labour and materials.
- Phase 3 consists of a further two 3.3 Mtpa concentrator modules, built sequentially and located alongside the Phase 1 and 2 modules, which increase the total Phase 3 concentrator capacity to 10.7 Mtpa.

Figure 1.24Figure 24.1 shows the plan of the areas included in the Platreef 2025 FS mine design and the additional material included in the Platreef 2025 PEA mine design.

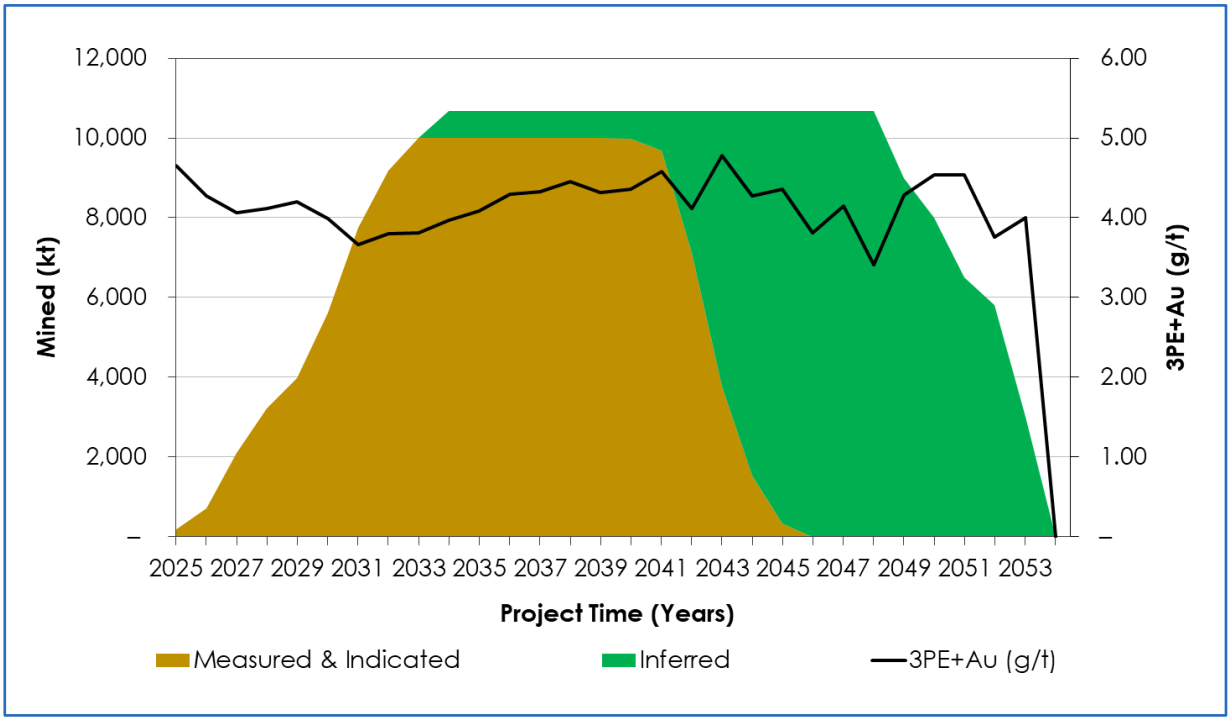
Figure 1.24 Platreef 2025 PEA Design



OreWin, 2025

The production of the Platreef 2025 PEA by Resource category is shown in Figure 1.25

Figure 1.25 Platreef 2025 PEA by Resource Category



OreWin, 2025

1.19.11.1 Platreef 2025 PEA Financial Results Summary

The key features of the Platreef 2025 PEA include:

- The incremental expansion capital cost for the 10.7 Mtpa Platreef 2025 PEA is estimated at $803 million.
- After-Tax Net Present Value (NPV) of $3,195M, at an 8% discount rate.
- After-Tax Internal Rate of Return (IRR) of 25.03%.

The results of the financial analysis show an After-Tax NPV8% of $3,195M. The Platreef 2025 PEA exhibits an After-Tax IRR of 25.03% and a payback period of 8.7 years. The estimates of cash flows have been prepared on a real basis as at 1 January 2025 and a mid year discounting is taken to calculate Net Present Value (NPV), these can be found in Section 24. A summary of the financial results is shown in Table 1.16.

Table 1.16 Platreef 2025 PEA Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	16,682	12,733
	5.00%	6,865	5,263
	8.00%	4,171	3,195
	10.00%	3,015	2,301
	12.00%	2,182	1,652
	15.00%	1,329	983
	20.00%	518	339
IRR	–	26.82%	25.03%
Project Payback (Years)	–	8.7	8.7

1.20 Conclusions

1.20.1 Platreef Integrated Development Plan 2025

The Platreef IDP25 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 2025 FS provides the basis for detailed planning of the Project execution and to update the long-term development plans for Platreef.

1.20.2 Geology and Mineral Resource Estimates

Mr Jeremy Witley 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, 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 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.
- 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 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).

1.20.3 Mining

The mine plan and expenditure schedule presented herein is reasonable. The plan is based on Platreef 2025 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.

1.20.4 Metallurgy

The metallurgical testwork programme, over the various phases, has yielded sufficient information to develop a definitive metallurgical flow sheet, and the testwork concluded is considered adequate. Detailed mineralogical analysis of selected ores, and the tailings 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 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 are deemed to be sufficient.

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

A mini pilot plant campaign was conducted, primarily, to produce bulk concentrate samples for downstream hydrometallurgical refining testwork and concentrate dewatering testwork. 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 testwork and concentrate dewatering testwork but did not provide comprehensive data to fully confirm the metallurgical performance as achieved in bench-scale locked cycle testwork.

The results for open circuit and locked cycle flotation testwork at SGS, Mintek and XPS over the period 2020–2022 (Post 2017 FS) were found to fall within the ranges evidenced in the 2017 FS variability testwork. This shows consistency between the various test campaigns.

The locked cycle test results as derived during the 2017 FS are considered adequate for deriving metallurgical performance projections.

The proposed Phase 2 comminution circuit which includes a multi-stage crusher circuit followed by ball milling is the preferred option for the concentrator based on the comminution characteristics and throughput requirements while also considering the future capacity expansion in Phases 3 and 4. The inclusion of an HPGR circuit for the tertiary crushing duty in Phase 2 was previously identified as an opportunity during the 2022 PDP FS which now has been incorporated in the process design.

The proposed flotation circuit is based on interpretation of the results obtained from the bench-scale open circuit and locked cycle 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 tailings handling and dewatering circuit is based on interpretation of the results obtained from the bench-scale thickening and filtration testwork. In Phase 2, a new tailings receiving, thickening and pressure filtration circuit, sized for the combined 4.07 Mtpa throughput, will be installed at the Rietfontein DTSF to treat the combined pumped concentrator tailings slurry. For Phase 2 a filtered tailings moisture content of 12–14% (w/w) will be targeted thus plate and frame pressure filters have been recommended for the Phase 2 tailings filtration duty.

The proposed phased concentrator capacity ramp up and modular design approach is also considered to be appropriate for this level of study.

1.20.5 Infrastructure

A number of mining projects are in the development phase on the Bushveld 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 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 with an agreed provision up to 10,000 m3/d of treated effluent from the Masodi WWTW to be supplied. Alternative bulk water supply sources have not been further investigated.

The installation of the Eskom Bulk supply substation and overhead lines was completed in August 2024. The present Notified Maximum Demand is 100 MVA and should be increased to 110 MVA in year 2032. A Power Supply agreement has been signed between Ivanplats and Eskom.

The construction of an on-site 9,589 MWh per annum solar power plant is complete with commissioning performed during February 2025.

1.21 Recommendations

1.21.1 Platreef Integrated Development Plan 2025

The Platreef 2025 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 2025 FS describes a phased development plan with an initial production rate of 770 ktpa and then an expansion to 4.07 Mtpa.

1.21.2 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.

1.21.3 Geology and Mineral Resource Estimates

Ivanhoe plans to focus on the development of the Platreef underground mine, and no additional exploration drilling is expected within the next few years. FW mineralisation can be further evaluated in future from underground drilling, 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.

1.21.4 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 should be completed as prioritised in the current schedule to better define the contacts for proper infrastructure placement.

1.21.5 Metallurgical

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

No additional testwork is currently planned. The Phase 1 Project is currently being executed and includes the construction of an initial 0.77 Mtpa concentrator, which will be followed by a phased capacity expansion in Phase 2 to increase throughput to 4.07 Mtpa. The planned future Phases 3 and 4 will aim to further increase concentrator throughput to 10.67 Mtpa. Further optimisation of the flow sheet during the Phase 1 commissioning, ramp up and optimisation phase should evaluate the potential for concentrate grade optimisation and improved 3PE+Au recovery by recycling the scavenger cleaner tailings to the rougher circuit. Additionally, the potential to conduct further trials of an SIBX reagent suite should be considered.

Open circuit and locked cycle flotation testwork has consistently shown that as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). However, it was also shown that if the circuits target an excessively high mass pull, the concentrate grades can be depressed without any notable gain in recovery. The final concentrate recovery and grade will need to be continuously optimised during full scale operations by targeting, controlling and optimising concentrate mass pull.

1.21.6 Infrastructure

The concluded agreements with the Mogalakwena Local Municipality appear to provide sufficient bulk water supply for the Platreef Project via treated effluent from the Masodi WWTW. Further investigations into alternative bulk water sources (notably the ORWRDP) may be required, however, should concerns with the treated effluent supply be encountered.

Continuing interfacing with Eskom to ensure alignment on the electrical supply requirements and load ramp up. Acceleration of the Project 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 Organization for Metals and Energy Security; and JGC Holdings Corporation, 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 Platreef IDP25, 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 IDP25 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 IDP25 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 IDP25:

- OreWin Pty Ltd (OreWin): Overall report preparation, underground development and production costs, and financial model, underground mine plan and Mineral Reserve.
- The MSA Group (MSA): 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.
- WSP Group 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 IDP25 uses metric units of measure. The currency used is Q4'24 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, 1.3, 1.11, 1.13, 1.17, 1.19, 1.21 and 1.22; Section 2; Section 3; Section 4; Section 6; Section 15; Section 16.2 to 16.7; Section 19; Section 21 as shown in Section 21.1; Section 22; Section 23; Section 24; Sections 25.1, and 25.3; Sections 26.1, and 26.4; and Section 27.
- Jeremy Witley, Pr.Sci.Nat. (SACNASP), FGSSA, Head of Mineral Resources, The MSA Group (Pty) Ltd, was responsible for: Sections 1.4 to 1.10, 1.21 and 1.22; Section 2; Section 3; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Section 25.2; Section 26.2 and 26.3; 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 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 1.1, 1.2, 1.3, 1.11, 1.13, 1.17, 1.19, 1.21 and 1.22; Section 2; Section 3; Section 4; Section 6; Section 15; Section 16.2 to 16.7; Section 19; Section 21 as shown in Section 21.1; Section 22; Section 23; Section 24; Sections 25.1, and 25.3; Sections 26.1, and 26.4; and Section 27.
- Louise Lintvelt, B.Sc. (Chemical Engineering), Pr Eng ECSA (202001104), employed by DRA Global Ltd as Process Study Consultant, was responsible for: Sections 1.14, 1.15, 1.21, 1.22; Section 2; Section 3; Section 13; Section 17; Section 21 as shown in Section 21.1; Section 25.4; Section 26.5; and Section 27.
- Ryan Males, MSc Eng (Civil), PR Eng ECSA (20050006), employed by DRA Global Ltd as Vice President - Civil Engineering, was responsible for: Sections 1.16, 1.21 and 1.22; Section 2; Section 3; Section 5; Section 18.1 to 18.4, 18.6 to 18.15; Section 21 as shown in Section 21.1; Section 25.5; Section 26.6; and Section 27.
- Riaan Thysse, B.Eng., Pr Eng ECSA (20120335), employed by WSP Group Africa (Pty) Ltd as Business Unit Lead, was responsible for Sections 1.18, 1.21 and 1.22; Section 2; Section 3; Section 18.5; Section 20; Section 21 as shown in Section 21.1; Section 25.6; Section 26.7; and Section 27.

2.3 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Mr 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, 3 April 2019, 16 March 2022, 21 June 2022, 25 - 26 April 2023, 20 February 2025. 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. Mr 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 Jeremy Witley visited the Project from 22 January to 26 January 2024. The purpose of the visit was to inspect drill core, review the sampling and logging, and gain familiarity with the Project through various workshops and discussions on the geological setting of the Mineral Resource. No exploration was being carried out at the time of the site visit.
- Mr William Joughin visited the property 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. Subsequent visits were carried out on 27 September 2016 to assess the rock conditions and support in Shaft 1, and on 13 June 2023 to inspect underground conditions in the shaft pillar area. William Joughin 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 Curtis Smith visited the property on 3 April 2019, 16 March 2022, 21 June 2022, 25 - 26 April 2023, 20 February 2025. 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. Mr 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.
- Mrs Louise Lintvelt visited the property on 9 June 2015 and 23–24 January 2018. The site visits included briefings from Ivanhoe geology and metallurgical personnel, inspection of drill core in combination with input and planning related to drilling and sampling requirements for metallurgical testwork. Mrs Lintvelt 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 Ryan Males visited the property on 3 October 2023 to evaluate progress on the Platreef Phase 1 Project. This entailed site inspections of the main Civils and Bulk Earthworks construction packages for the process plant, buildings and mining surface infrastructure including the various stormwater management measures for which he is the Pr Eng signatory of the associated WULA documentation and drawings. He also visited the site on 5 February 2025 to review progress of current Mining infrastructure and buildings in construction, the water treatment facility and Raw Water Dam and #2 Shaft Headgear.

- Mr Riaan Thysse has not visited the site. WSP's design and project delivery teams visited the property on the 16th of February 2024 for a geotechnical investigation at Rietfontein site and for the Plant Site TSF construction quality assurance during 2024 to date.

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:	15 February 2025
Mineral Resource Amenable to Selective Underground Mining Methods:	15 February 2025
Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	15 February 2025
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	15 February 2025
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Validation of resource models as current using current economic inputs:	15 February 2025
Effective Date of Mineral Resources:	15 February 2025
Effective Date of Mineral Reserves:	15 February 2025

2.5 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of the Platreef IDP25 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 IDP25 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 Mr Bernard Peters, Mr Curtis Smith and Mr Jeremy Witley have fully relied upon, and disclaim responsibility for, information derived from the report:

- Ivanplats March 2025: Platreef 4 Mtpa Phase 2 Feasibility Study, Title and Legal.

This information is used in Sections 1.2, 1.3.1 and Section 4 of the Platreef IDP25 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 March 2025: Platreef 4 Mtpa Phase 2 Feasibility Study, Title and Legal.
- 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 Mr Bernard Peters, Mr Curtis Smith and Mr Jeremy Witley 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:

- Ivanplats March 2025: Platreef 4 Mtpa Phase 2 Feasibility Study, Marketing.

Mr Bernard Peters and Mr Curtis Smith, the QP for the marketing assumptions, has relied on Ivanhoe and disclaims responsibility for the marketing assumptions in Section 19 and Section 22. Mr Jeremy Witley has also fully relied upon and disclaims responsibility for this information in support of the Mineral Resource estimate in Section 14.

3.4 Environmental

Mr Riaan 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 Jeremy Witley has also fully relied upon and disclaims responsibility for this information in support of the Mineral Resource estimate in Section 14.

3.5 Infrastructure

Mr Ryan Males the QP for the Infrastructure, has relied on Ivanhoe and DRA Projects and 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 No. 07/A61G/GCJAIBF/6975 Reference No. WU7725 (dated 7th September 2021)
- Amended Water Use Licence of No. 07/A61G/GCJAIBF/6975 Reference No. WU22334 (dated 25th August 2023)
- Memorandum of Agreement between Mogalakwena Local Municipality and Ivanplats Propriety Limited in respect of Sewerage Infrastructure Repairs and Upgrades in Mogalakwena Local Municipality (dated 15 June 2023)

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, dry stack tailings storage facility (DSTSF) and supporting infrastructure, located approximately 280 km northeast of Johannesburg and 8 km from the town of Mokopane in Limpopo Province of South Africa.

4.1.1 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$290M. In June 2014 and September 2014, Ivanplats concluded its B-BBEE transaction in two 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

4.1.2 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 dry stack 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.

4.1.3 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.

4.1.4 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.

An Electricity Supply Agreement has been signed with Eskom SOC Limited ("Eskom") for the supply of bulk power with a Notified Maximum Demand of 100 MVA. The Eskom sourced bulk power supply was installed as Phase 1 of the Project and commissioned in August 2024.

4.1.5 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' 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 Integrated Development Plan 2025 (Platreef IDP25) and should be read in its entirety and must accompany every copy made of the Platreef IDP25. The Platreef IDP25 has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Platreef IDP25 has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by OreWin Pty Ltd, The MSA Group (Pty.) Ltd, SRK Consulting (South Africa) (Pty) Ltd, DRA Projects (Pty) Ltd and WSP Global Inc as the Report Contributors.

The Platreef IDP25 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 IDP25 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 IDP25 with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR).

The results of the Platreef IDP25 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 IDP25. 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 IDP25 are to the accuracy stated in the Platreef IDP25 only and rely on assumptions stated in the Platreef IDP25. The results of further work may indicate that the conclusions, estimates and assumptions in the Platreef IDP25 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 IDP25. 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 IDP25 will be prone to fluctuations with time and changing industry circumstances.

The Platreef IDP25 should be construed in the light of the methods, procedures, and techniques used to prepare the Platreef IDP25. Sections or parts of the Platreef IDP25 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%

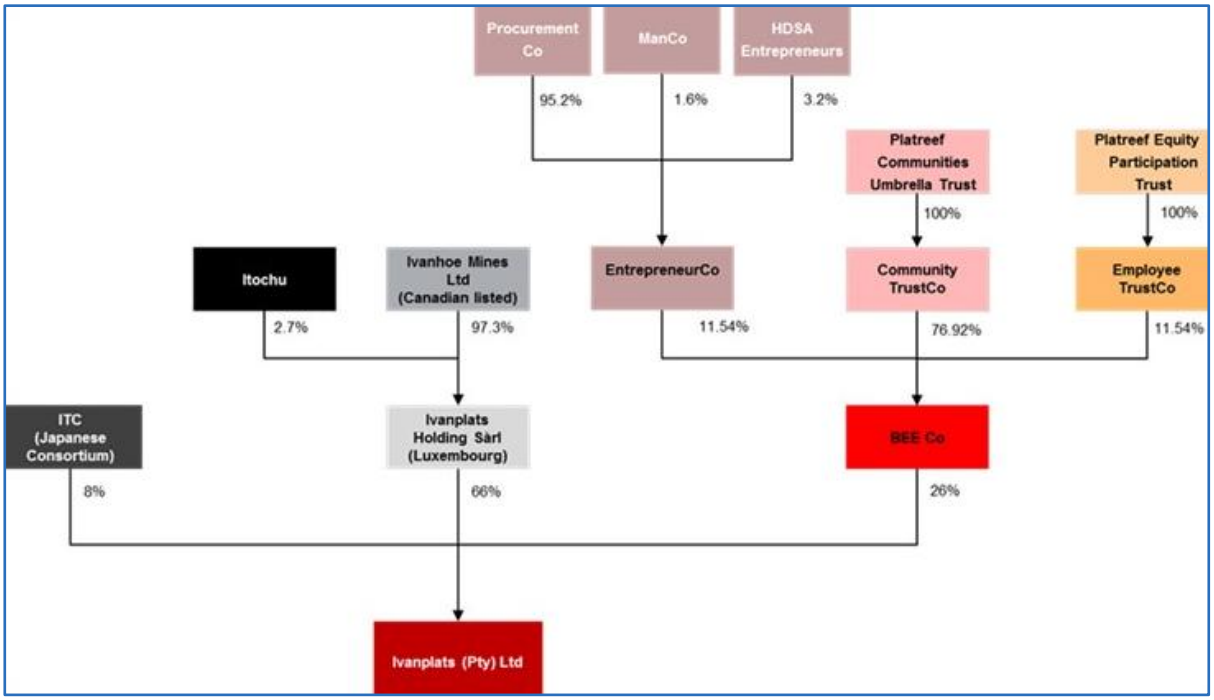
Community Trust Co is 100% owned by the Bonega Communities Trust, a trust established for the benefit of the inhabitants of the 20 villages comprising the Mokopane Community, a recognised traditional community under the jurisdiction of the Mokopane Traditional Council.

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, 2025

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

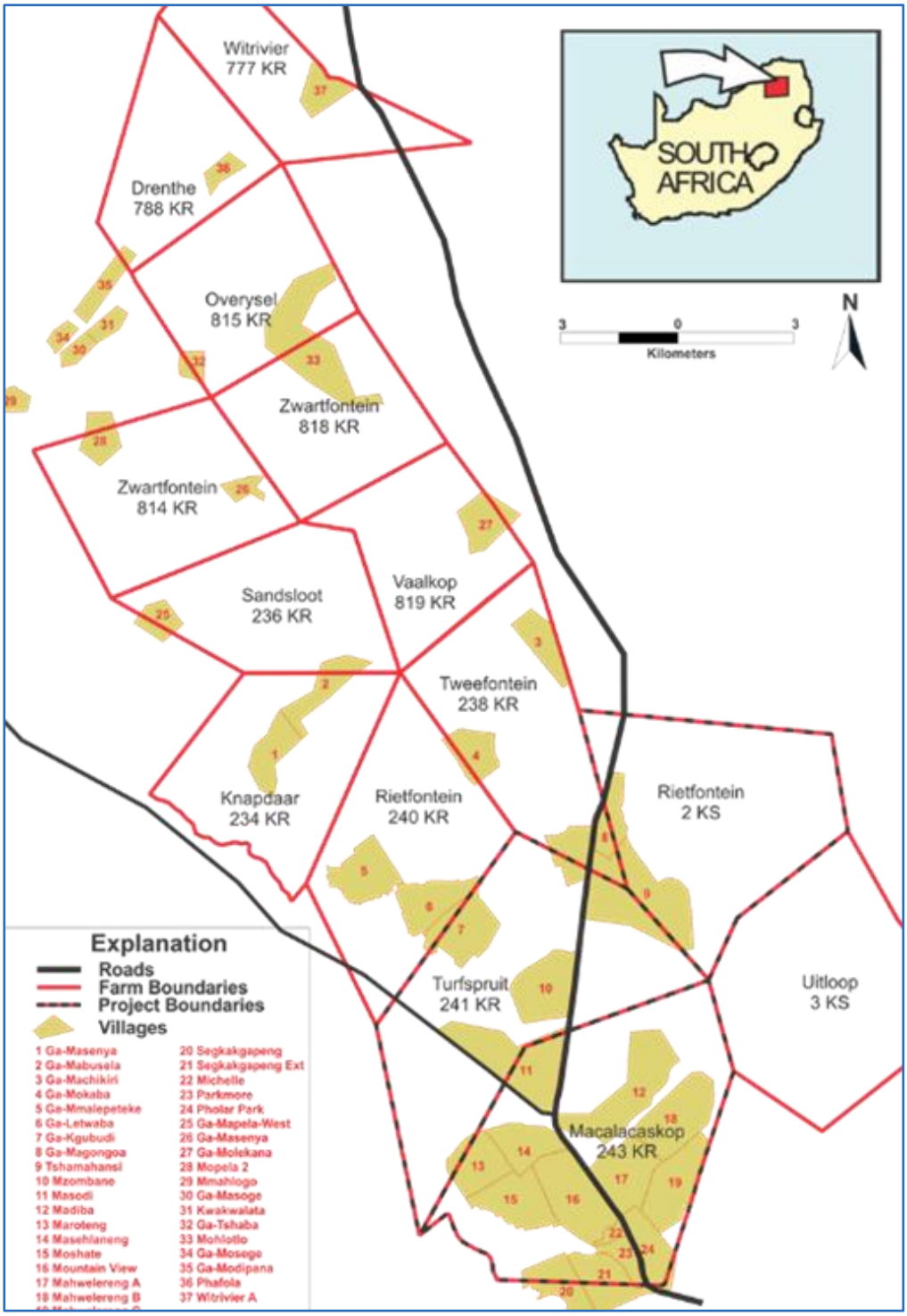
The Platreef Project is located at about 24°05'S and 28°59'E. The Project is located in the Limpopo Province of the Republic of South Africa on two farms: Turfspruit (3,561 ha), and Macalacaskop (4,281 ha). Ivanplats intends to extend the Platreef Project to Rietfontein (2,878 ha) in order to accommodate a DSTSF 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;
- 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 2,878 ha. 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 10 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 cooperation 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.

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 Servitude Areas

Ivanplats, 2025

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' 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 (DSTSF) 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.

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 an 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 agreement regulates the management of Ivanplats, 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:

4.3.4.1 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.

4.3.4.2 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.

4.3.4.3 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.

4.3.4.4 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.

4.3.4.5 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 waste water treatment works ("WWTW") 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:

- 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 R215 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; and
  - 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 kilolitre (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 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

An Electricity Supply Agreement has been signed with Eskom. The Notified Maximum Demand is 100 MVA.

In addition to the Eskom supply, Ivanplats has entered into a 10 year agreement with an independent power producer (IPP) for a minimum commitment of 48,300 MWh per annum of solar power. The solar power will be wheeled through the Eskom network. The scheduled start date for the wheeling is June 2026.

4.5 Royalties and Encumbrances

4.5.1 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:

4.5.1.1 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%

4.5.1.2 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).

4.5.2 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 in 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.

4.5.2.1 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 tailings disposal, may be located within the Rietfontein 2 KS area.
- The Right MR was granted in favour of Ivanplats on30 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 and Mineral Reserves. 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 Mr Bernard Peters, the information discussed in this Section supports the Platreef IDP25 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 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 large 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 bulk power supply is sourced from Eskom, the South African national power utility and this is installed as Phase 1 of the Project and commissioned in August 2024. An Electricity Supply Agreement has been signed with Eskom for a Notified Maximum Demand of 100 MVA.

The 132/33 kV substation is supplied from Eskom Borutho main transmission station via two 132 kV overhead lines (OHLs). Each OHL is rated for 175 MVA for full redundancy. The installed capacity in the Eskom 132/33 kV outdoor substation is four of 40 MVA 132/33 kV transformers. The design capacity is 120 MVA, with the fourth 40 MVA transformer for redundancy.

In addition to the Eskom bulk power agreement, Ivanplats has entered into a 10 year agreement with an independent power producer (IPP) for a minimum commitment of 48,300 MWh per annum of solar power. The solar power will be wheeled through the Eskom network. The reduction in the cost for the solar power is allowed for in the electrical operating expenditure cost. The scheduled start date for the wheeling is June 2026.

The construction of an initial on-site 9,589 MWh per annum solar power plant is complete and was commissioned in February 2025. The cost savings due to the on-site solar power plant is included in the electrical Opex cost. The rotational load shedding implemented by Eskom due to a lack of generating capacity is very much reduced due to additional generators being commissioned and improved maintenance of the aging coal thermal power stations. Eskom may still be required to implement load curtailment on large clients. Under load curtailment Eskom will inform the mine in advance should it be required to reduce load.

The initial 5 MVA constructions supply to the mine from the Mahwelereng 33 kV substation will be decommissioned and all the loads supplied from the newly commissioned 132/33 kV substation. These changes are scheduled for completion in early 2025.

The Platreef Project power requirement for a 4.07 Mtpa underground mine has been updated to predict an average Notified Maximum Demand (NMD) of approximately 110 MVA. Ivanplats will be required to submit an application to Eskom for the required increase in the Notified Maximum Demand which will be required in 2032.

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 groundwater for construction purposes and for dewatering of the underground workings.

Ivanplats has also concluded agreements with the Mogalakwena Local Municipality (MLM) to receive local treated effluent water from the Masodi Waste Water Treatment Works (WWTW) as the bulk water supply needed for operations. Ivanplats contributed financial assistance towards the construction and commissioning of the Masodi WWTW which was offset in the agreed rate per m3 for the treated effluent supply.

A minimum of 3,000 m³/d will be supplied for the first phase of production at Platreef increasing up to a maximum of 10,000 m³/d of treated effluent water, depending on the roll out of the supporting municipal infrastructure.

The average water balance for Phase 2 (January 2028 to LOM) determined that, on average, 9,809 m³/d water is required to operate the mine. The sources of water are an average of 5,602 m³/d bulk water supply from the Masodi WWTW to supply the site filtered water requirements, 222 m³/d from the four site boreholes to supply the site operational potable water requirements and the remaining 3,985 m³/d is supplied from moisture in the ore, from rainfall/runoff collected on site in the stormwater ponds and groundwater ingress to the workings.

The bulk water requirements will increase to approximately 6,800 m³/d as the Phase 2 concentrator ramps up. Provision has been made to further treat the water received from the Masodi WWTW via an on-site filtration plant for use in the process plant.

As the anticipated bulk water supply requirements could be addressed via the agreed allocation from the Masodi WWTW, Ivanplats is not currently considering any alternative bulk water supply options (e.g. participating in the proposed ORWRDP pipeline from Flag Boshielo Dam as previously envisaged.

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 upgraded N11, a paved, dual carriage road intersecting with the N1 at Mokopane and the N4 near Middelburg.

The N11 is the main access road passing the site, and as this is a national road, it falls under the jurisdiction of South African National Roads Agency Limited (SANRAL). An intersection with the N11 onto the mine property has been constructed in line with SANRAL requirements, which allows for heavy vehicles to turn north and south back onto the N11. Access to the planned DTSF is from a new intersection planned on the future N11 bypass road. The access road to the DTSF and new tailings filtration plant will be an asphalt-surfaced road to accommodate day-to-day maintenance traffic.

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 5.1 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 5.1 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 Rustenburg 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 IDP25

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 2022 Feasibility Study (www.sedar.com, Peters et al., 2022).

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 Complex (BC), which is the largest of the known layered igneous intrusions, covering an area >65,000 km². The BC hosts up to 75% of the world's platinum resources (Naldrett et al, 2009).

The BC 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 BC, 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 geological discussion are shown in Figure 7.4.

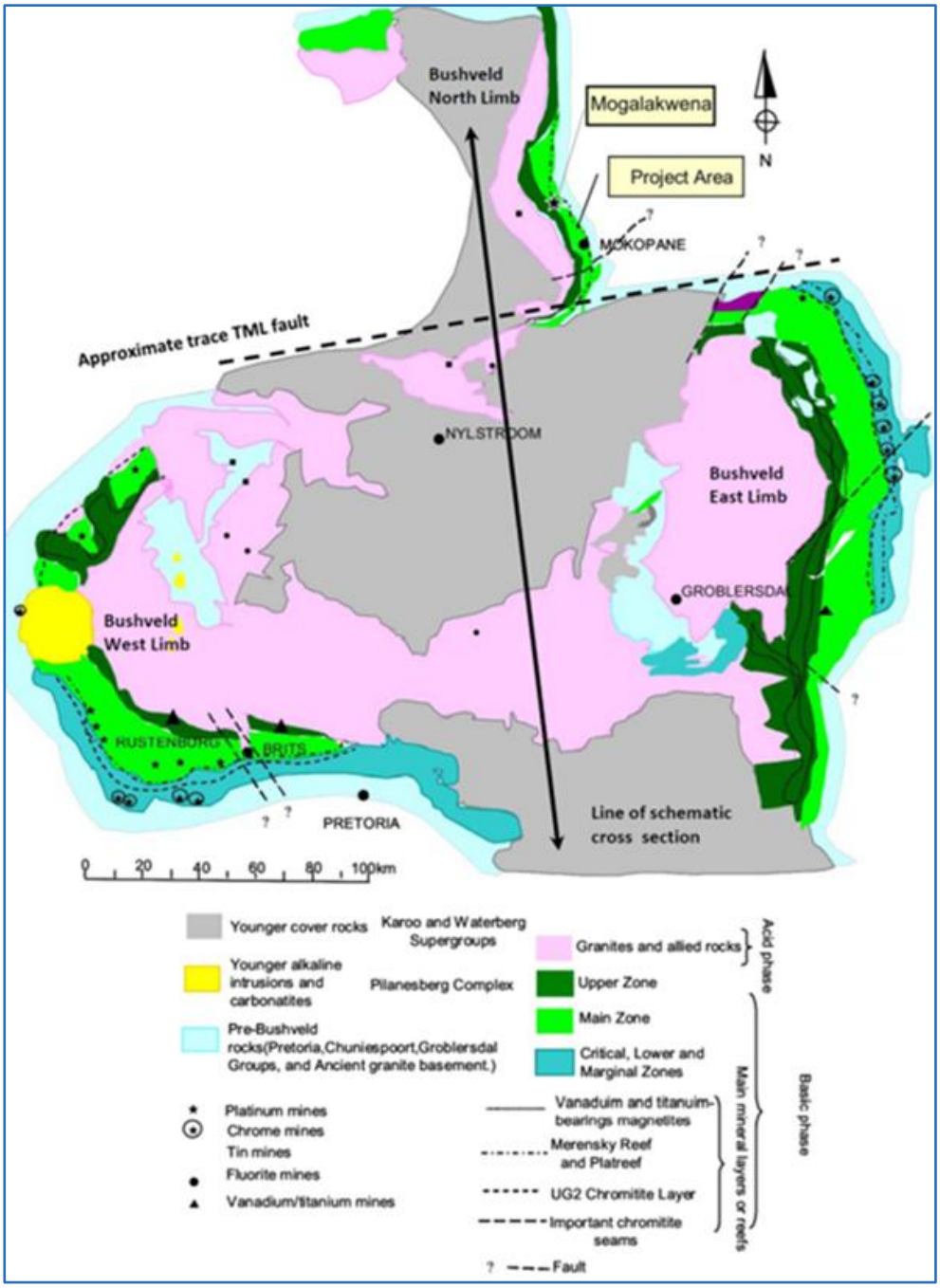
Figure 7.1 provides a location and regional geology map for the BC. Figure 7.2 provides a diagrammatic cross-section through the BC.

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

7.2 Project Geology

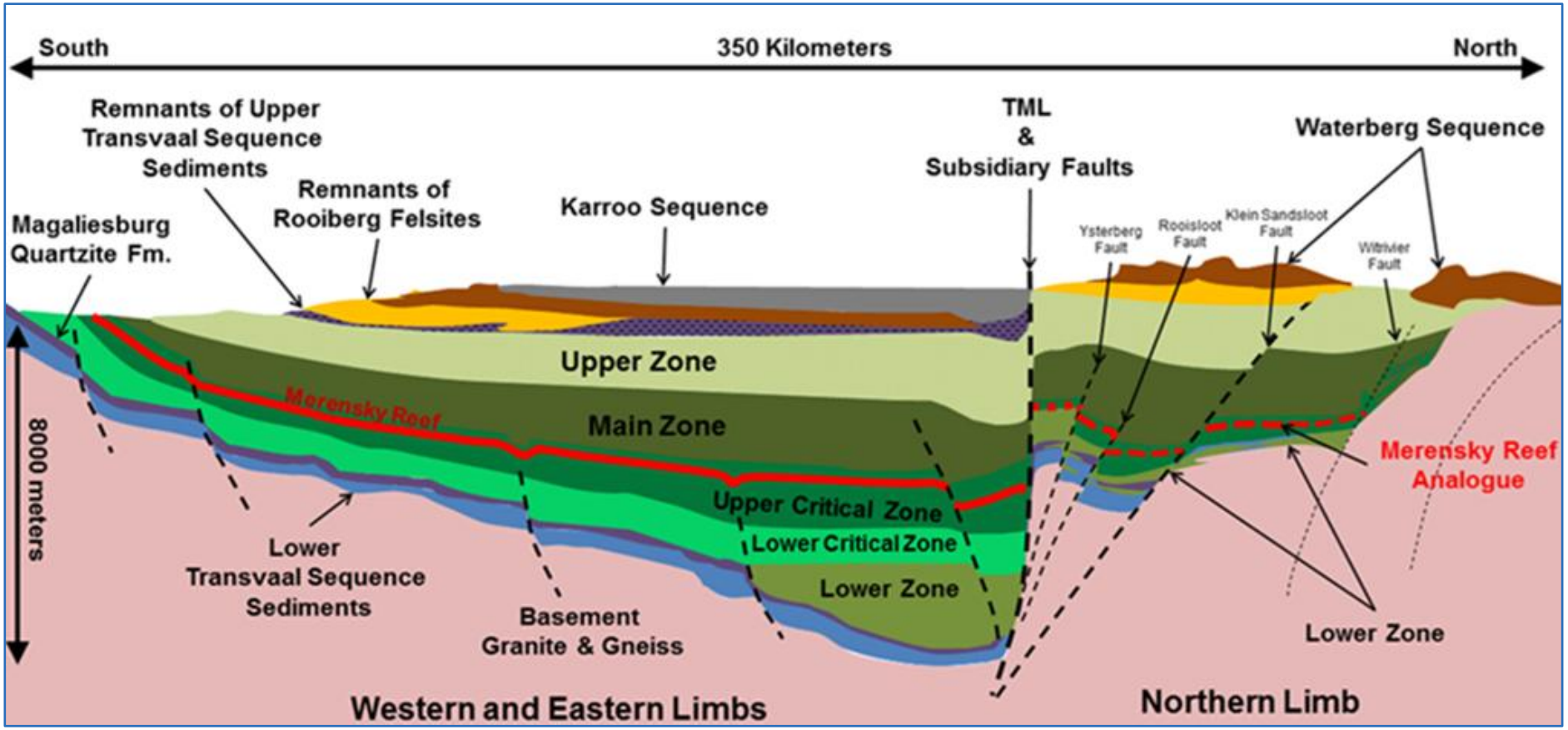
Figure 7.3 shows the Project geology projected to surface. The locations of Zones 1 to 5 referred to in the following geologic discussion are shown in Figure 7.4.

Figure 7.1 Regional Geological Plan of the Bushveld Complex



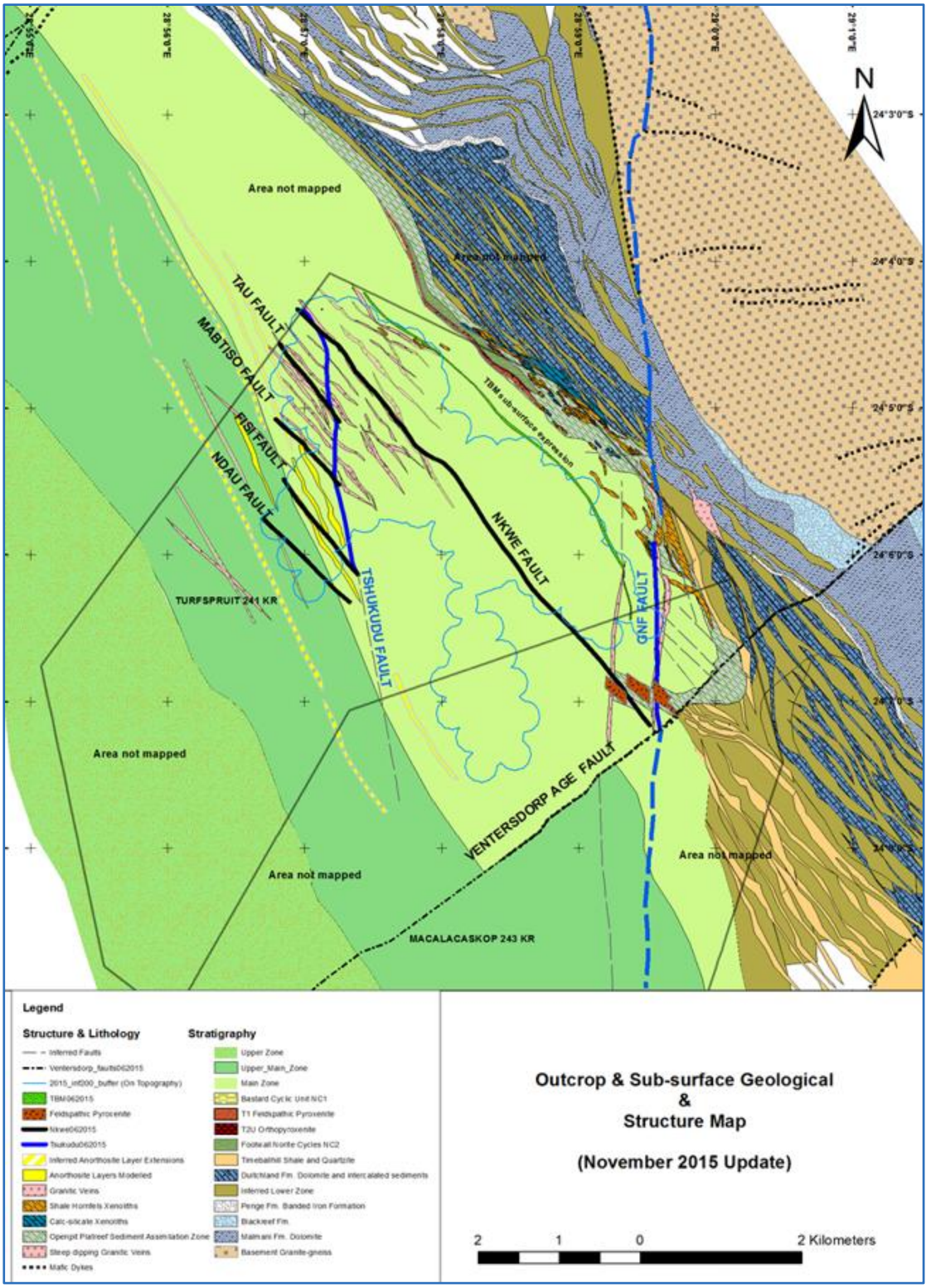
Modified after Viljoen and Schürmann, 1998; section line represents location of section in Figure 7.2

Figure 7.2 Schematic Cross-Section through the Bushveld Complex



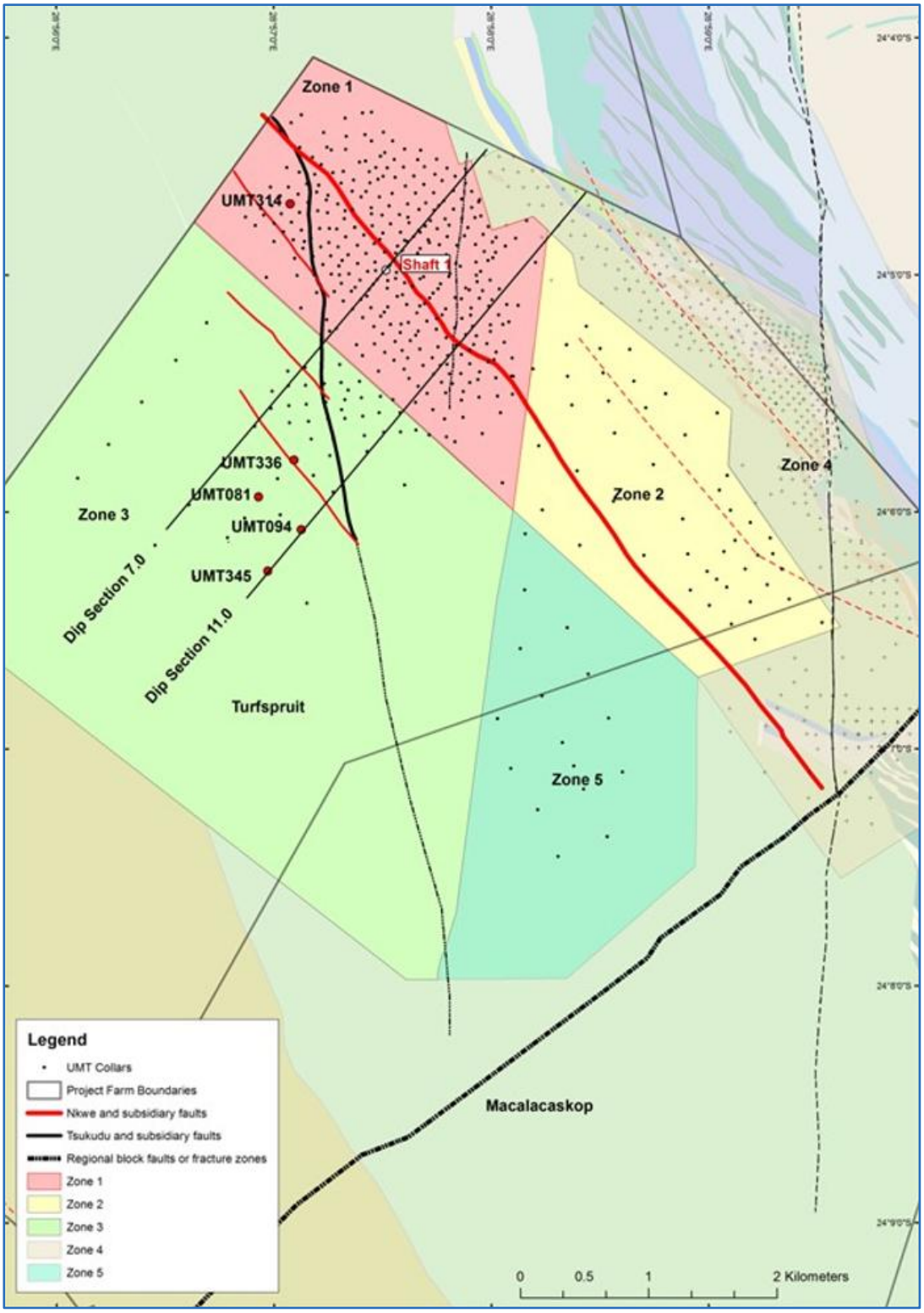
Ivanhoe, 2012; modified after Kruger, 2005. Figure is schematic and not to scale. Section line illustrated is shown in Figure 7.3.

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. Metamorphic and metasomatic processes associated with the magma-sediment interaction zones are the subject of various studies 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 UCZ. The down-dip Zone 3 (Figure 7.4) on the farm Turfspruit 241 KR 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 BC.

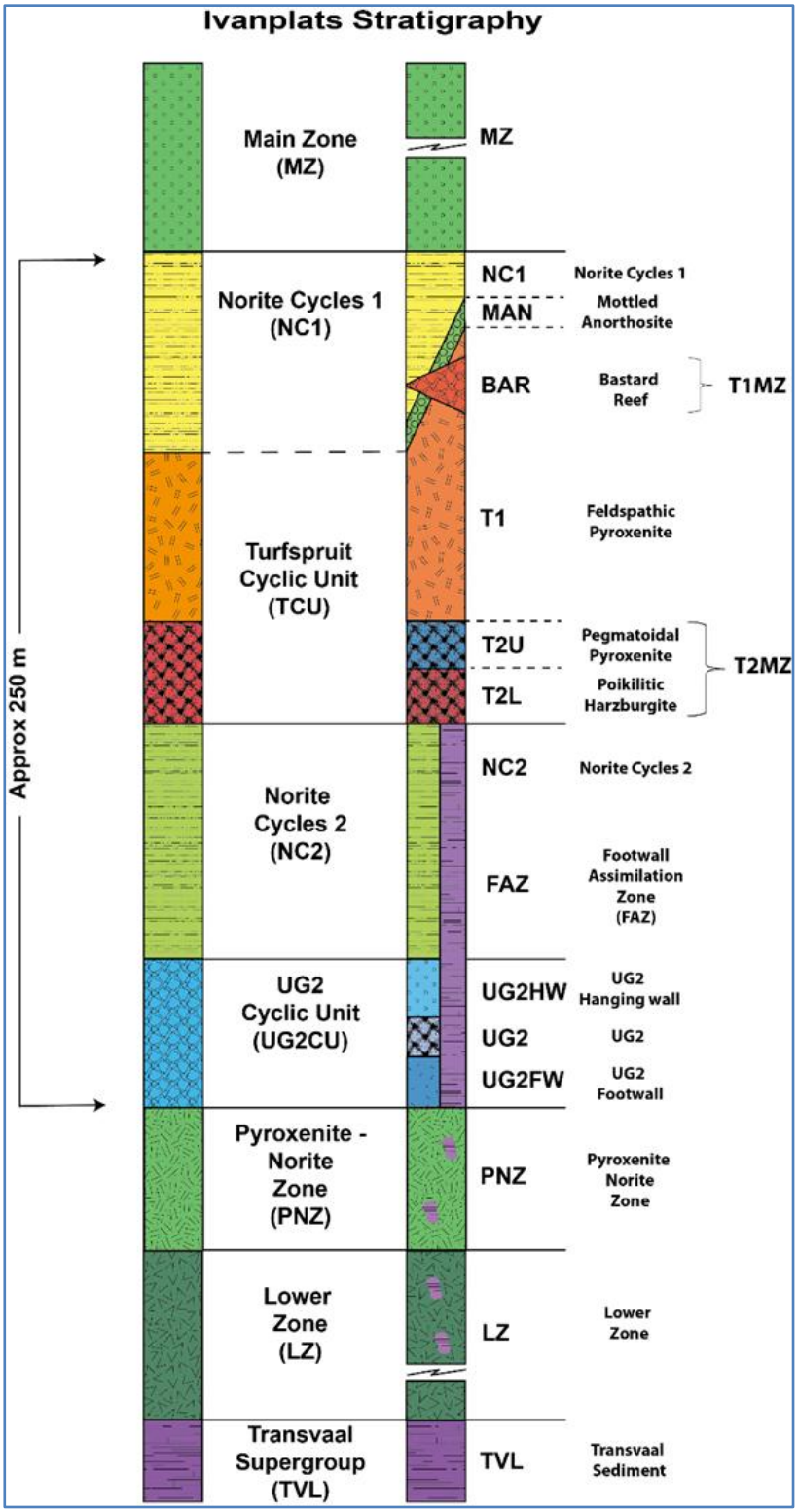
Each cyclic unit consists of a regular sequence of norite to pyroxenite and olivine cumulates with sub-horizontal rhythmic layering (cycles) developed with varying degrees of cyclicity. Layering can be spectacularly developed with regular cyclic layers (going upward) of chromite-orthopyroxenite-norite-anorthosite (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).

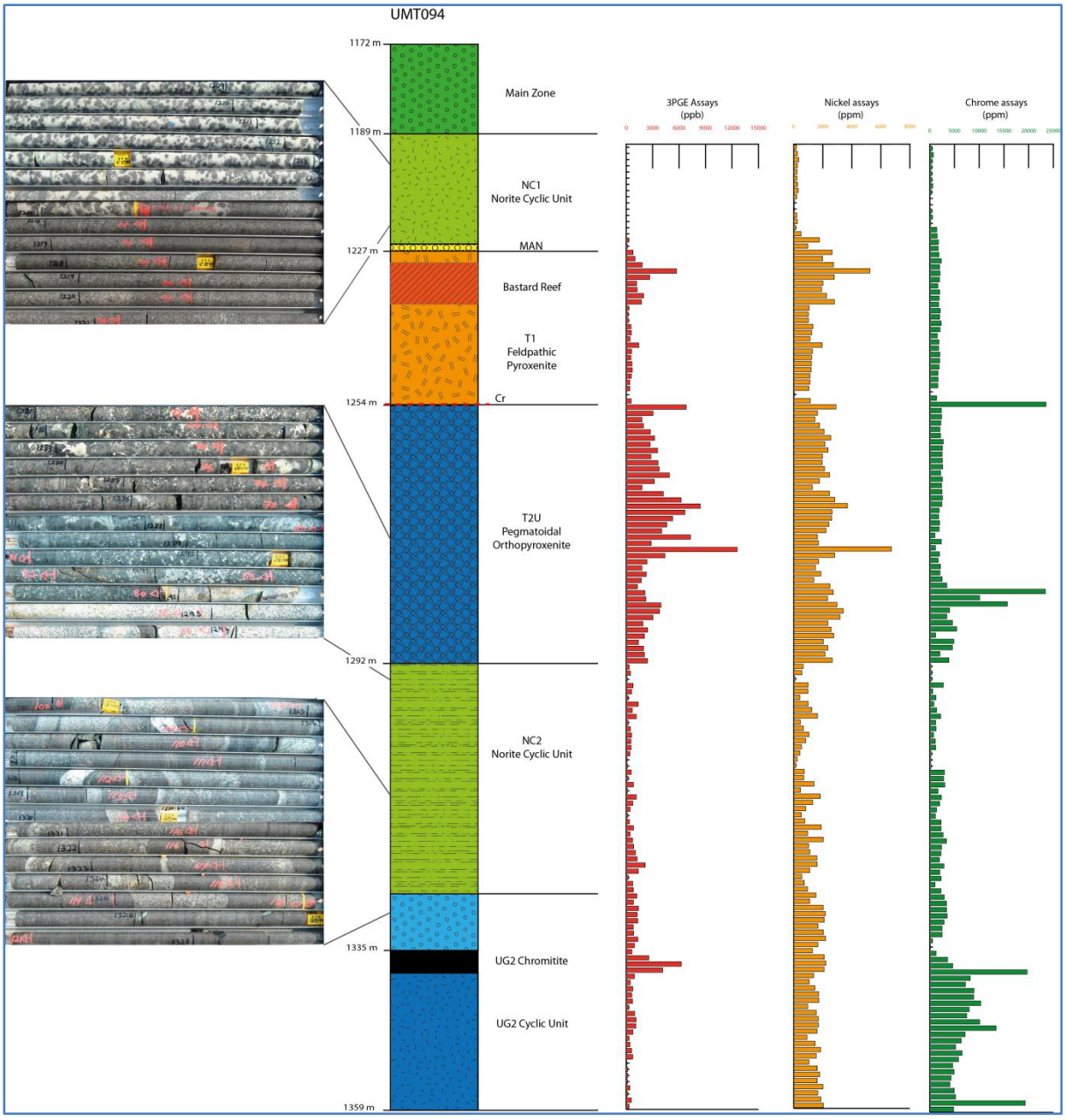
The four major magmatic cyclic units are shown in Figure 7.6. The major cyclic units consist of a series of alternating leucocratic to mafic to ultramafic lithologies within the interval from the base of the UG2 Cyclic Unit to the contact of the Main Zone gabbronorite. The cyclicity is most recognisable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

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; tick-marks 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 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.

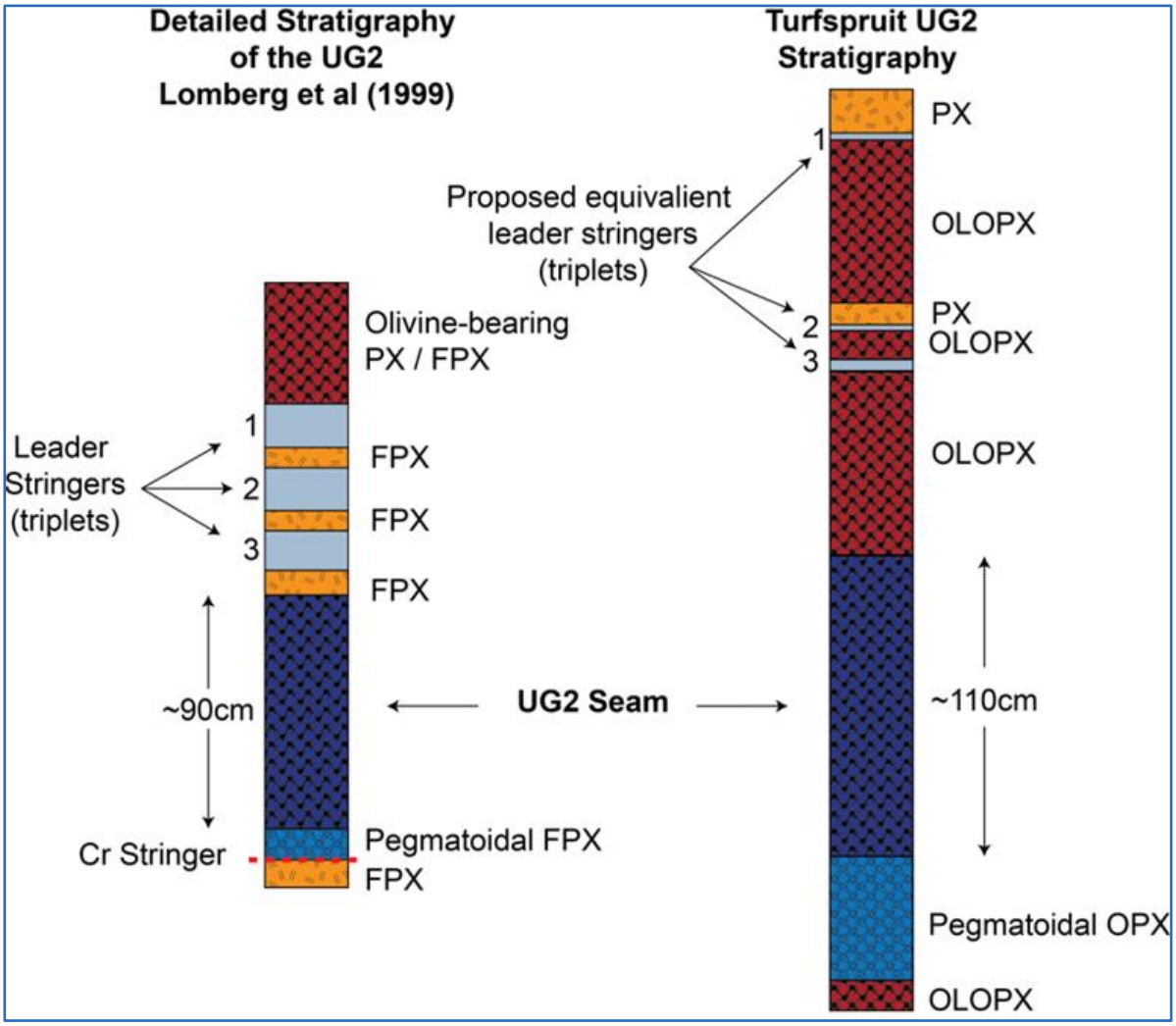
7.2.5 Pyroxenite-Norite-Zone (PNZ)

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)

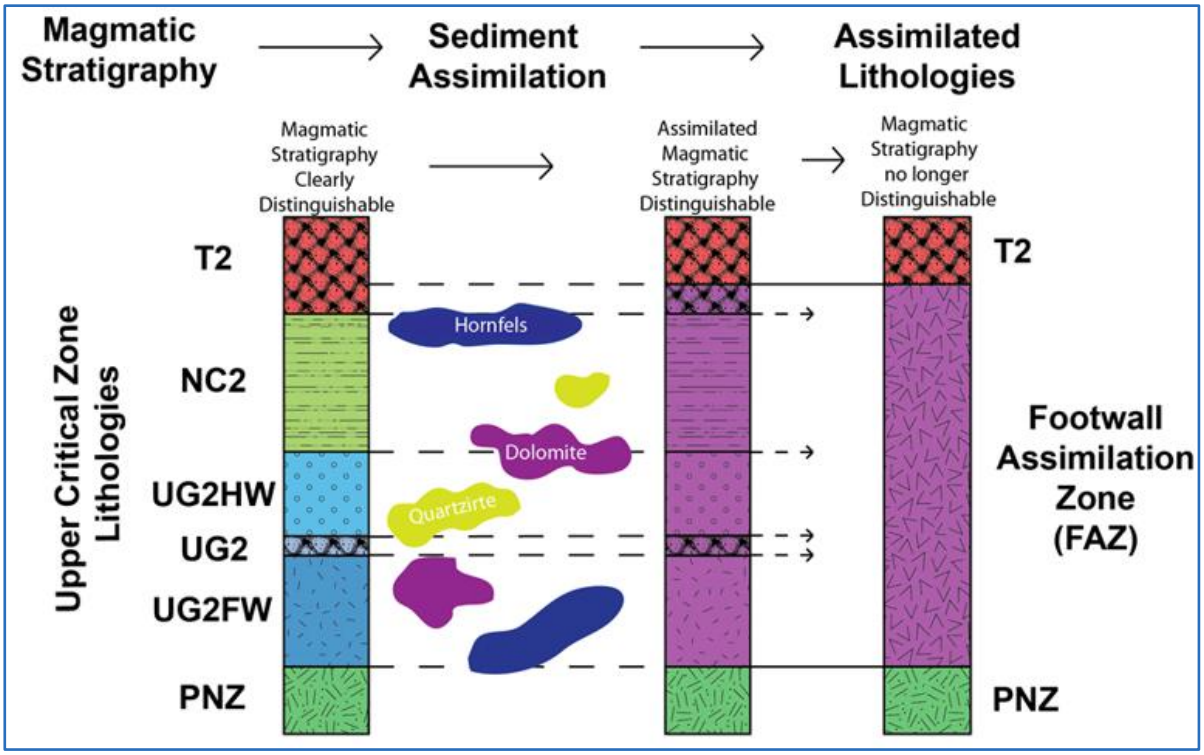


Ivanhoe, 2016

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 Eastern and Western Limbs of the BC.

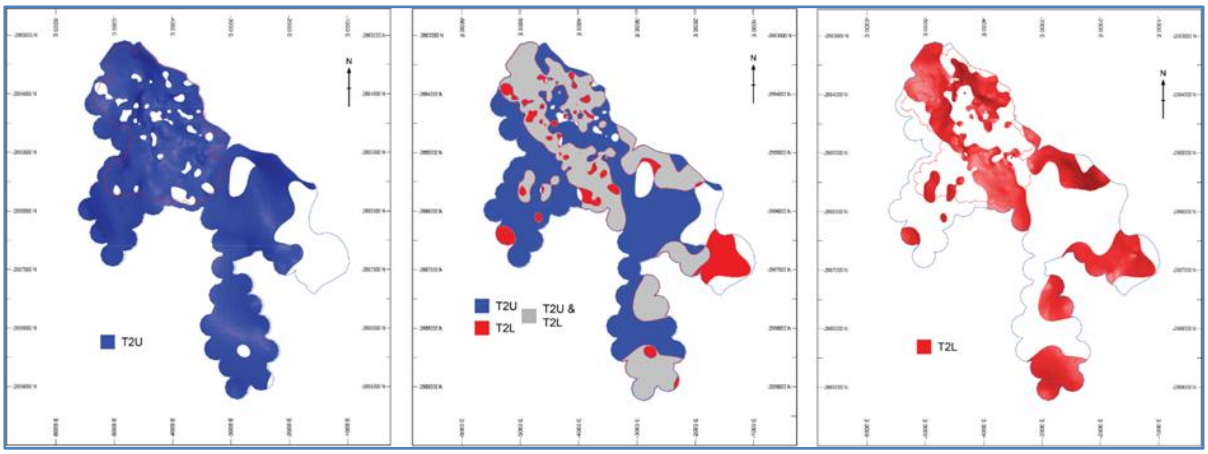
7.2.8.1 T2U and T2L

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.

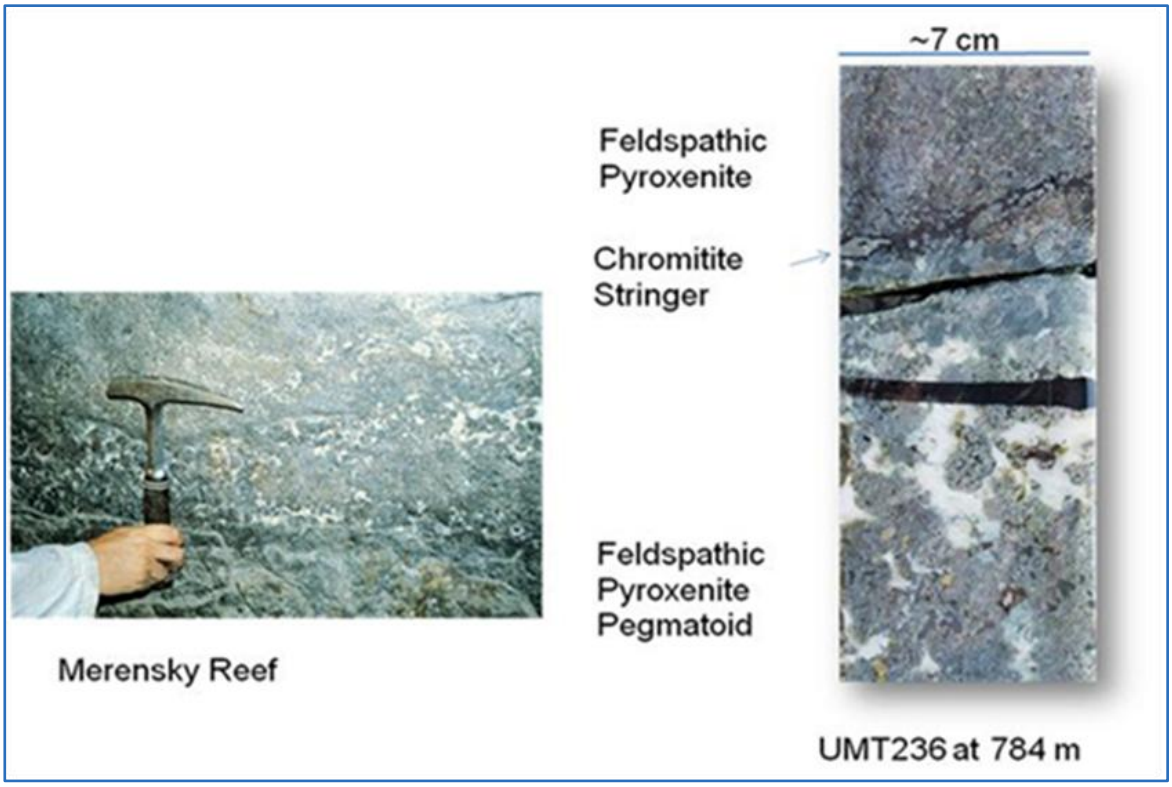
The T1MZ is therefore correlated with the Bastard Reef, and the mineralisation found above the top contact of the T2 pegmatoid, within the feldspathic pyroxenite, is correlated with the M1, as postulated by Davey (1992) and Lea (1996). This implies that the T2 pegmatoid correlates with the M2.

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	Minimum (m)	Maximum (m)	Average (m)
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 BC (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.

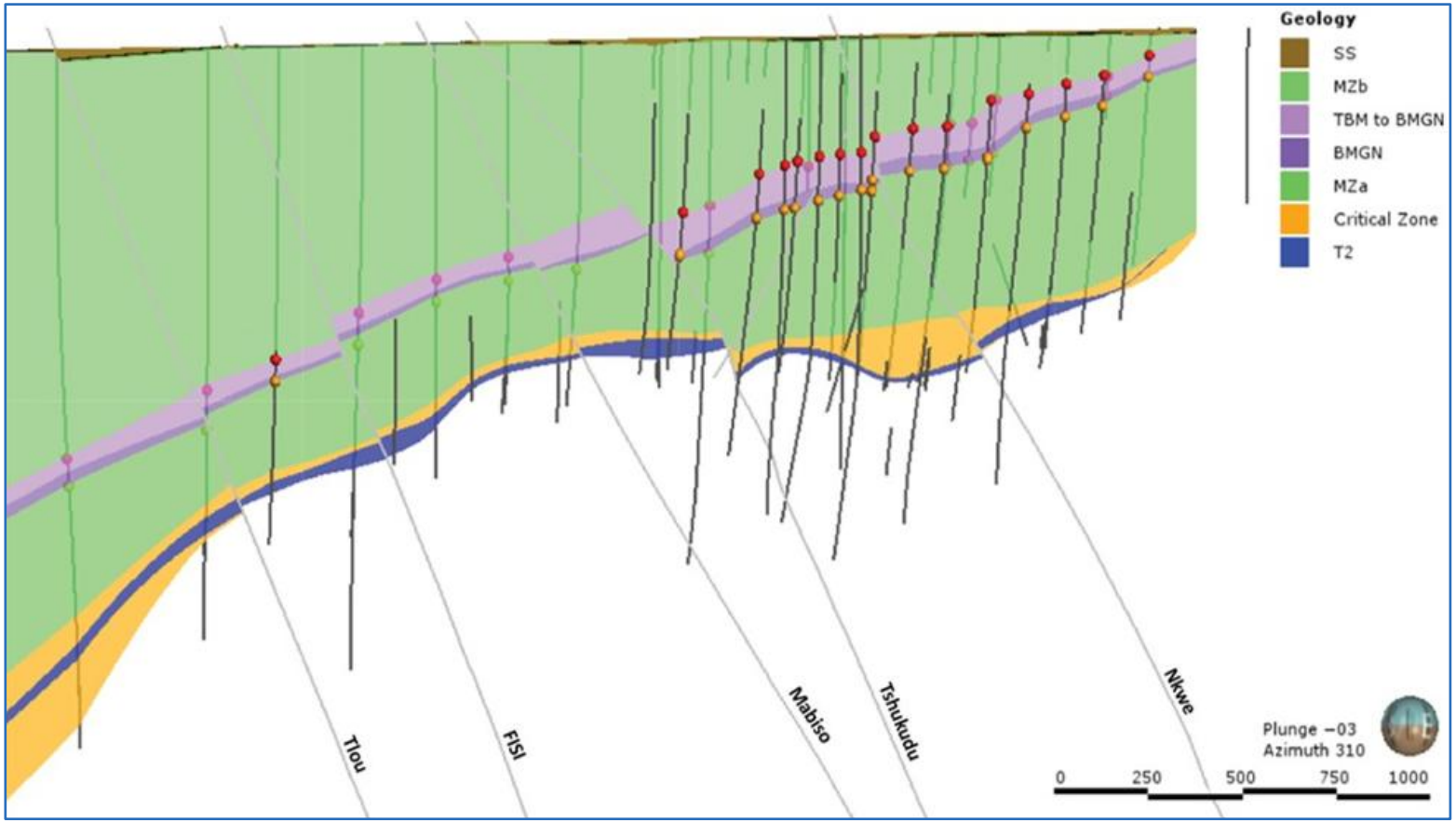
The Main Zone is broken into four units in the Project area (refer to Figure 7.11):

- 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.

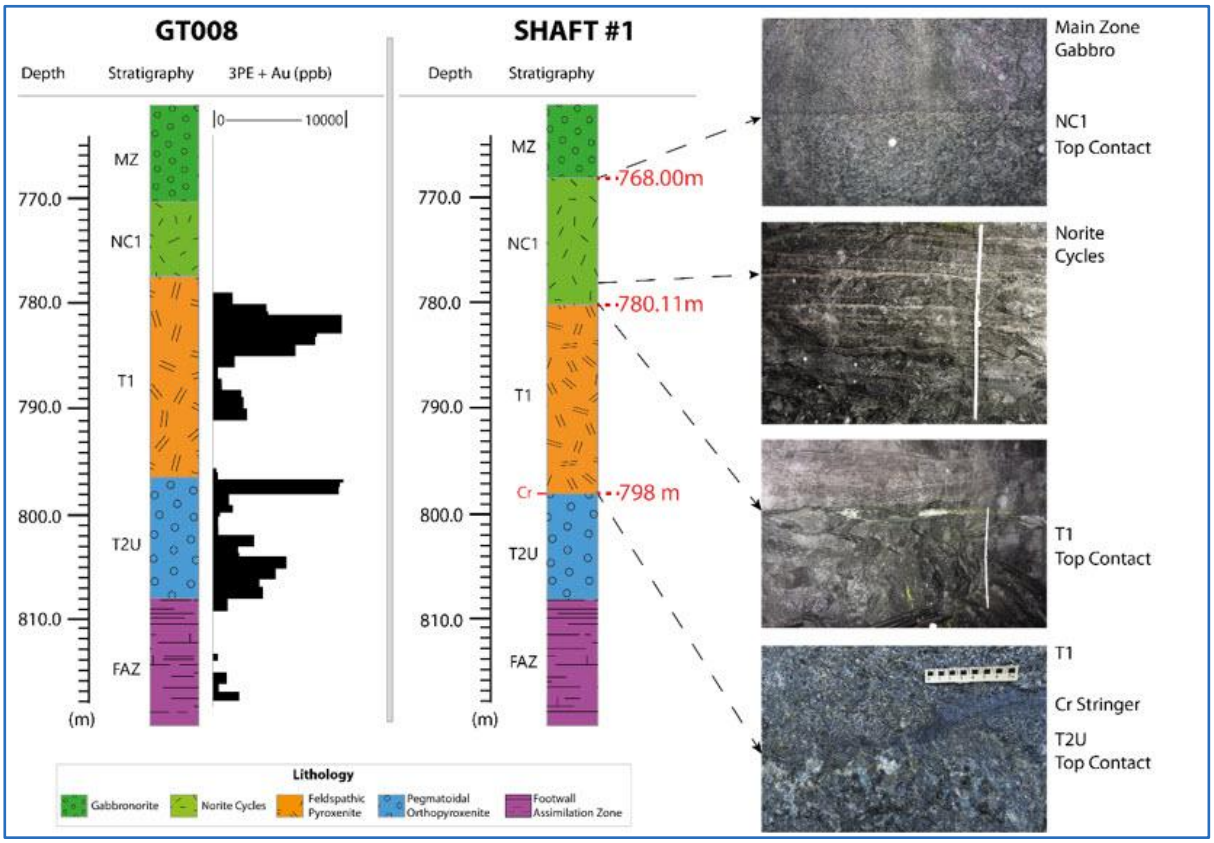
Figure 7.12 compares geology in drill hole GT008 (completed at the location of Shaft 1) and the corresponding intersection in Shaft 1. The top of the T1 was intercepted in Shaft 1 at 780.1 m compared to 777.5 m in drill hole GT008. The top of T2U was intercepted at 798 m in Shaft 1 compared to 796.5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

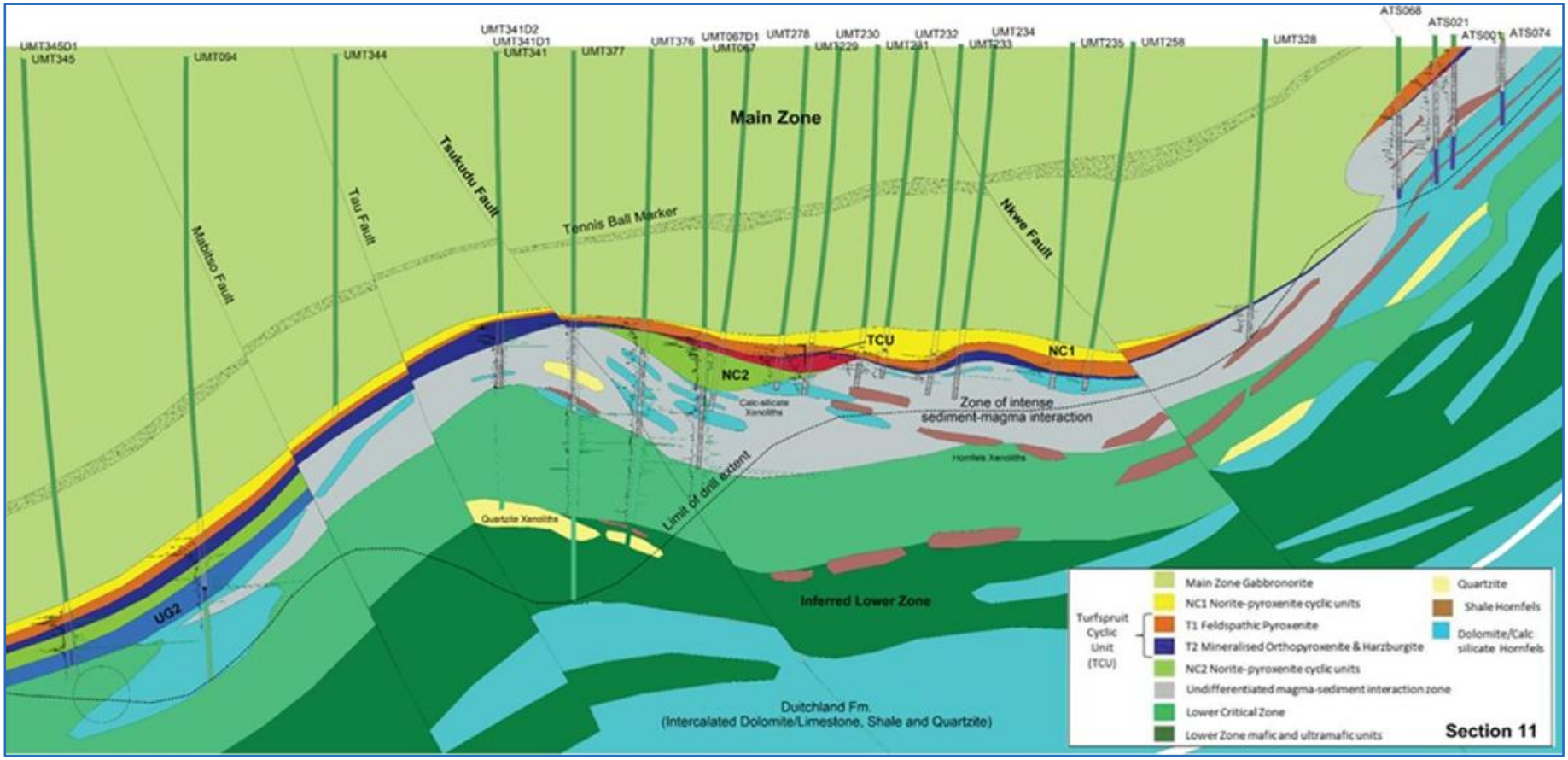
The stratigraphically lower and the updip magmatic units on the Turfspruit and Macalacaskop farms are characterised by interactions between Bushveld magma and the Transvaal Supergroup host sequence that is composed of diverse dolomite to alumino-silicate sedimentary rocks of the Duitschland Formation. 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

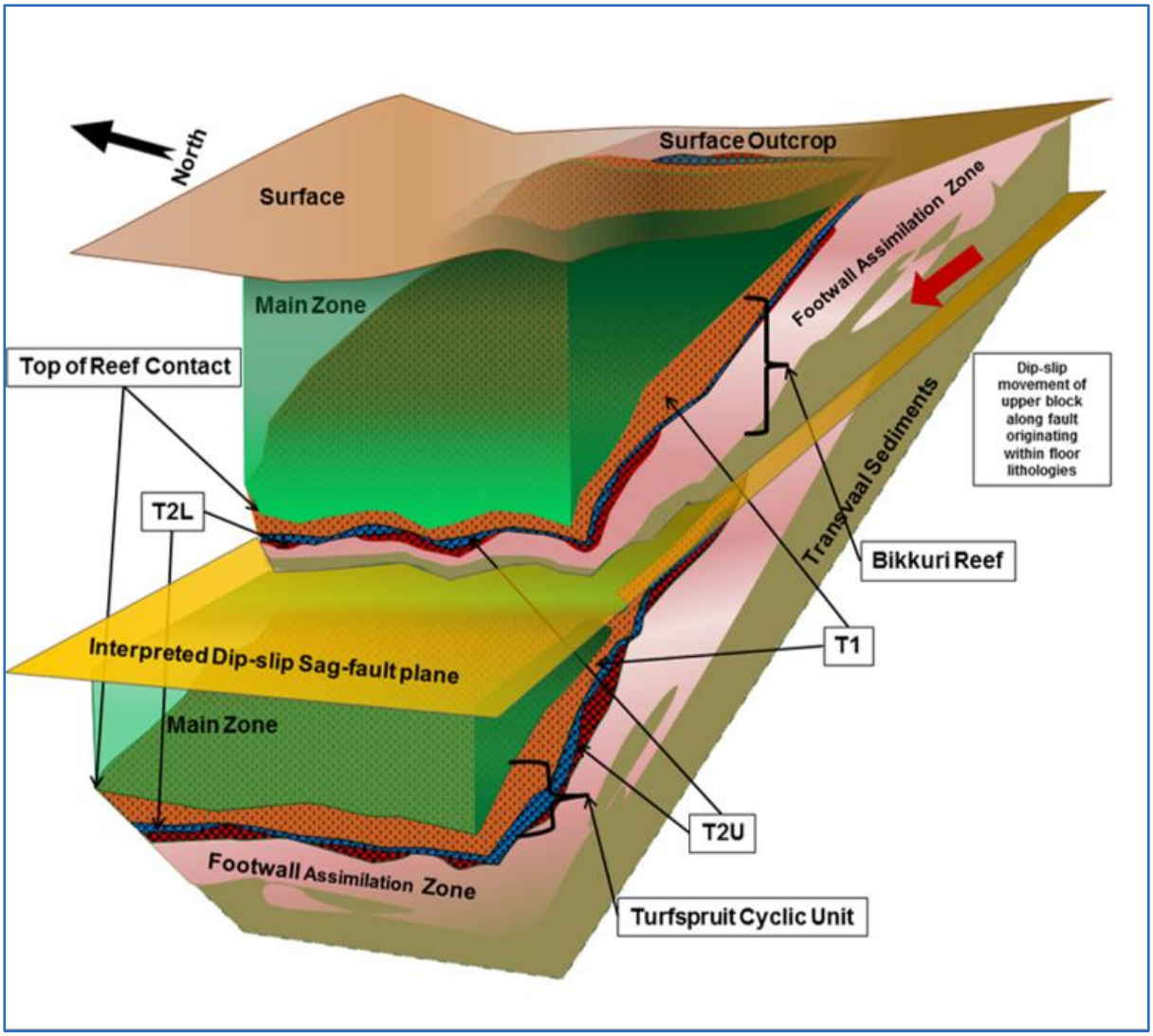
7.4 Bikkuri

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 relatively low 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 Eastern and Western Limbs 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.

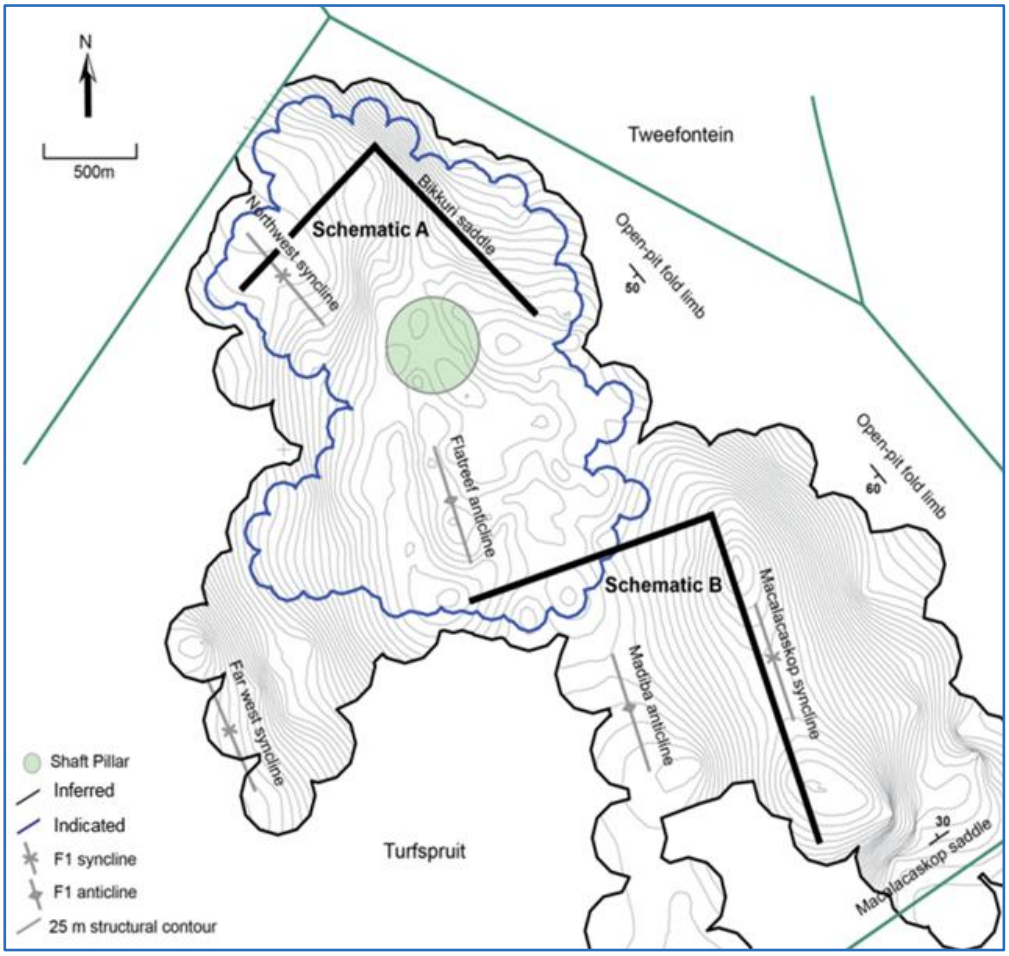
7.5.2 Project Folding

Two fold orientations have been observed, and these concur with 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 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' 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.

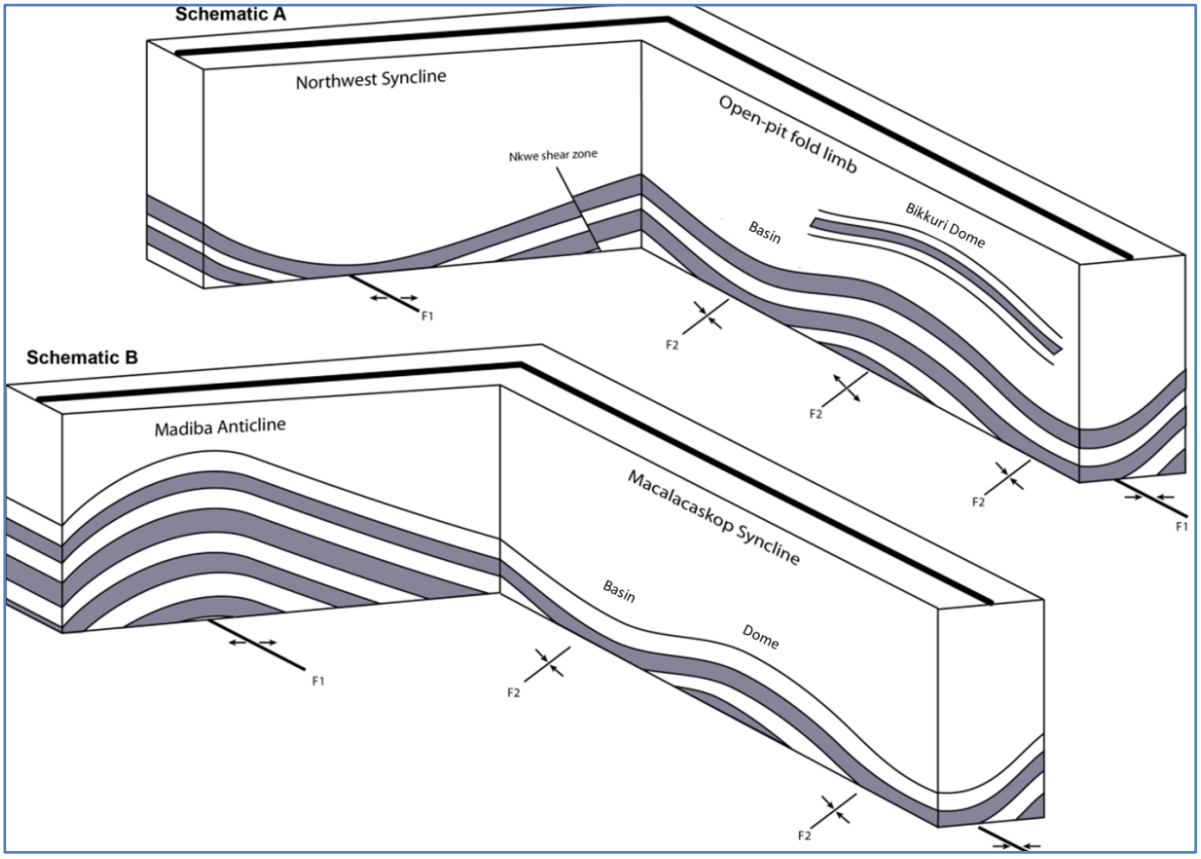
Figure 7.15 Plan View of Major F1 Fold Structures on Structural Contour Map



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

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.

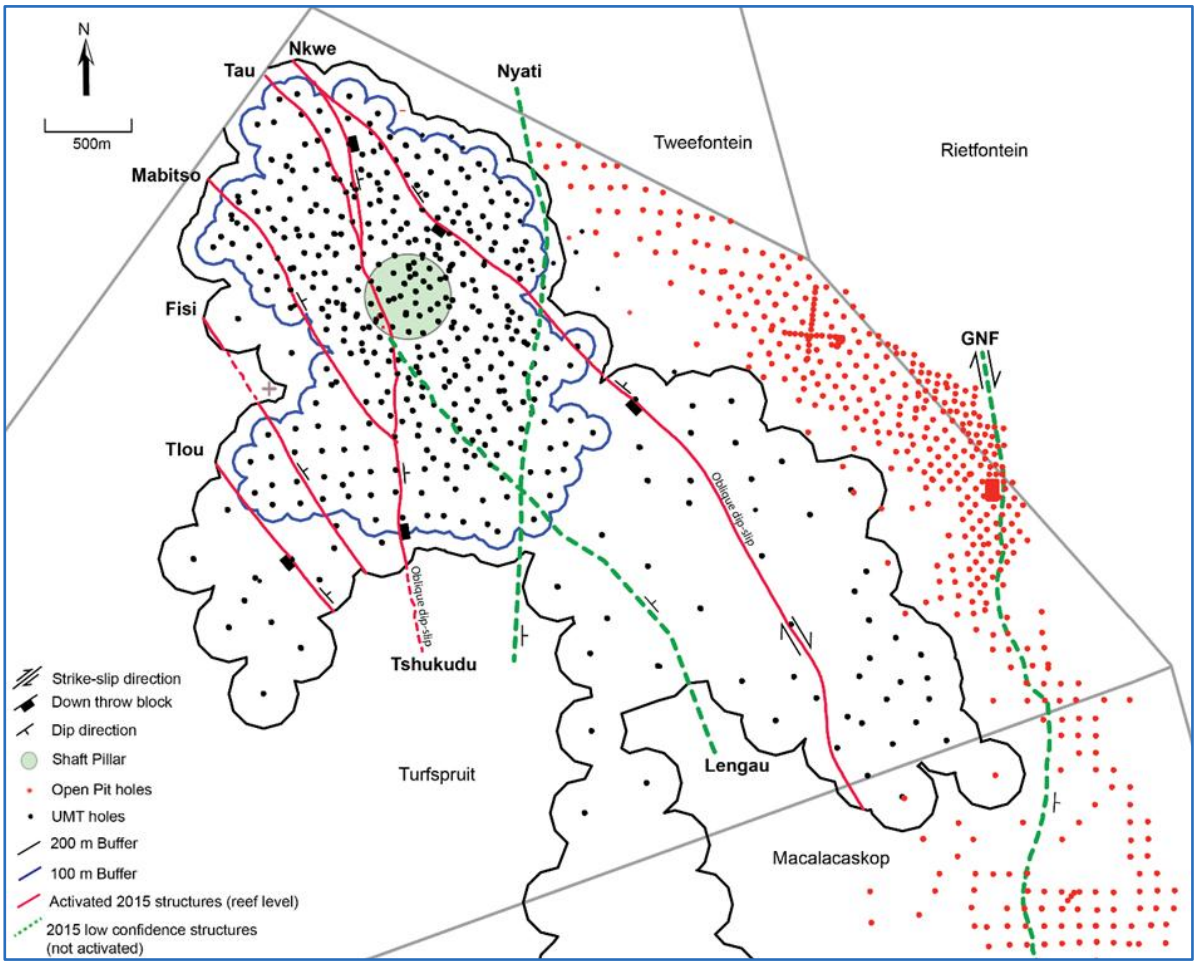
The major ductile fault structures currently recognised include:

- 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:

- North-east 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 north-west 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 north-west and dip steeply (60° to 75°) towards the north-east. 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.

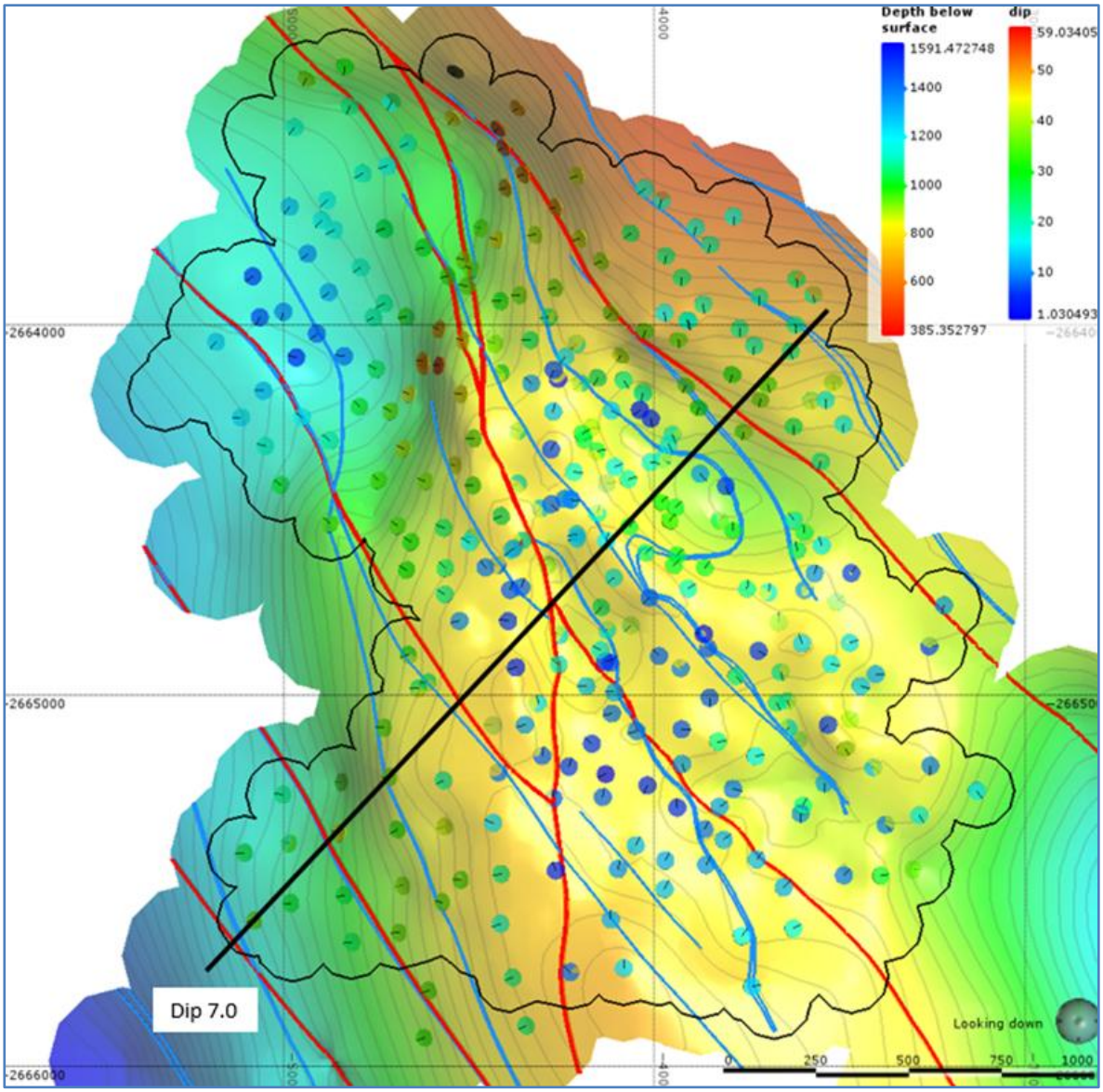
Figure 7.18 shows the locations of key dyke features in relation to the structural model at the level of the T2 horizon.

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 north-east; 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 north-east; 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 north-west.

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 BC. 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 BC (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 BC. 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 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 BC and on the Northern Limb.

Hutchison (2003), and Hutchison and Kinnaird (2005) completed work on ATS and AMK drillholes in the area of the historical 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 representativity 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 historical 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 is associated with BMS.

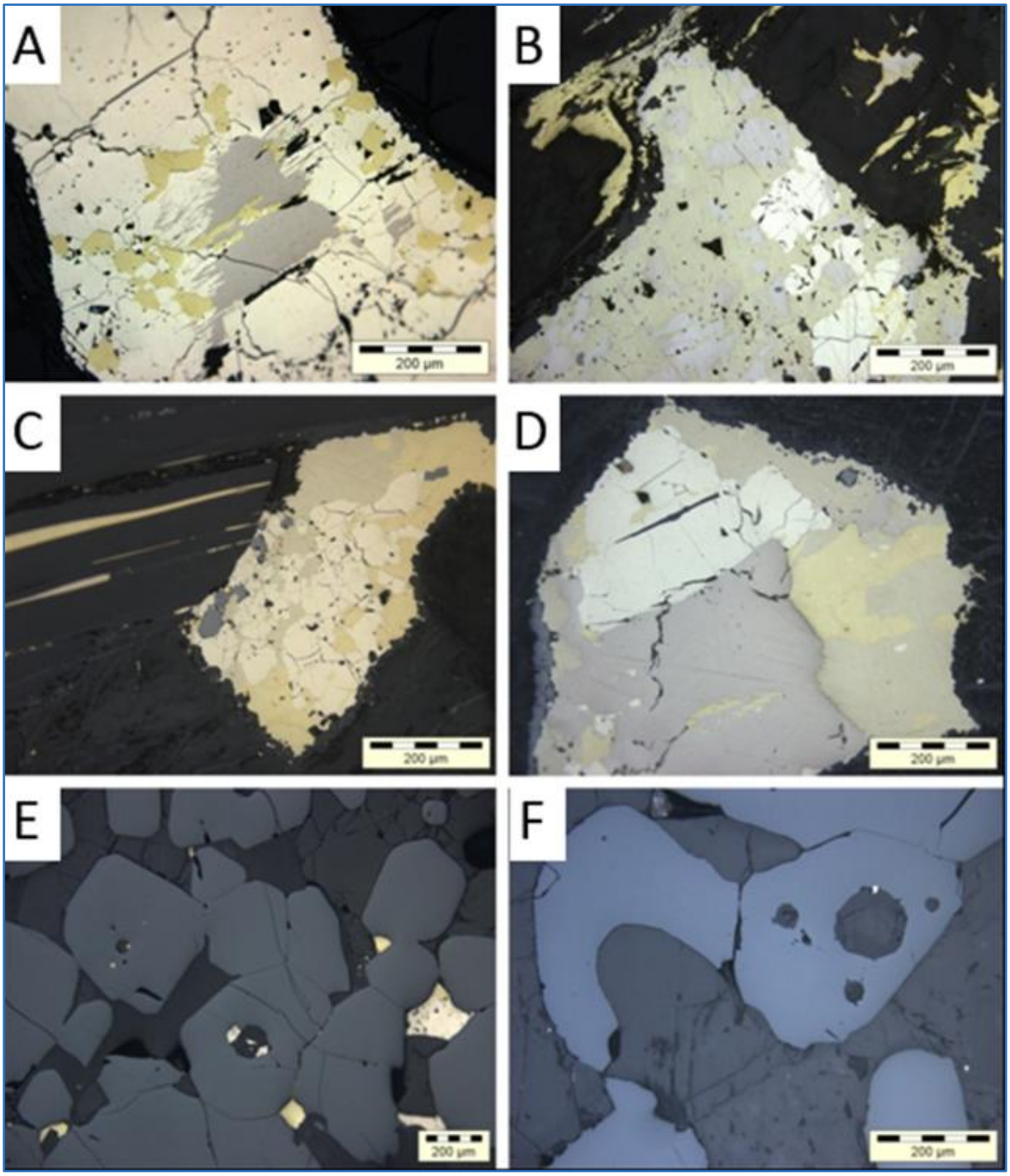
Yudovskaya (2015) determined that clear magmatic assemblages (Merensky-like trends) can be distinguished from an original assemblage influenced and overprinted by secondary effects. The zonation of PGM distribution favours in-situ crystallisation where modal PGE mineral assemblages are controlled by 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 1,000°C. 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.

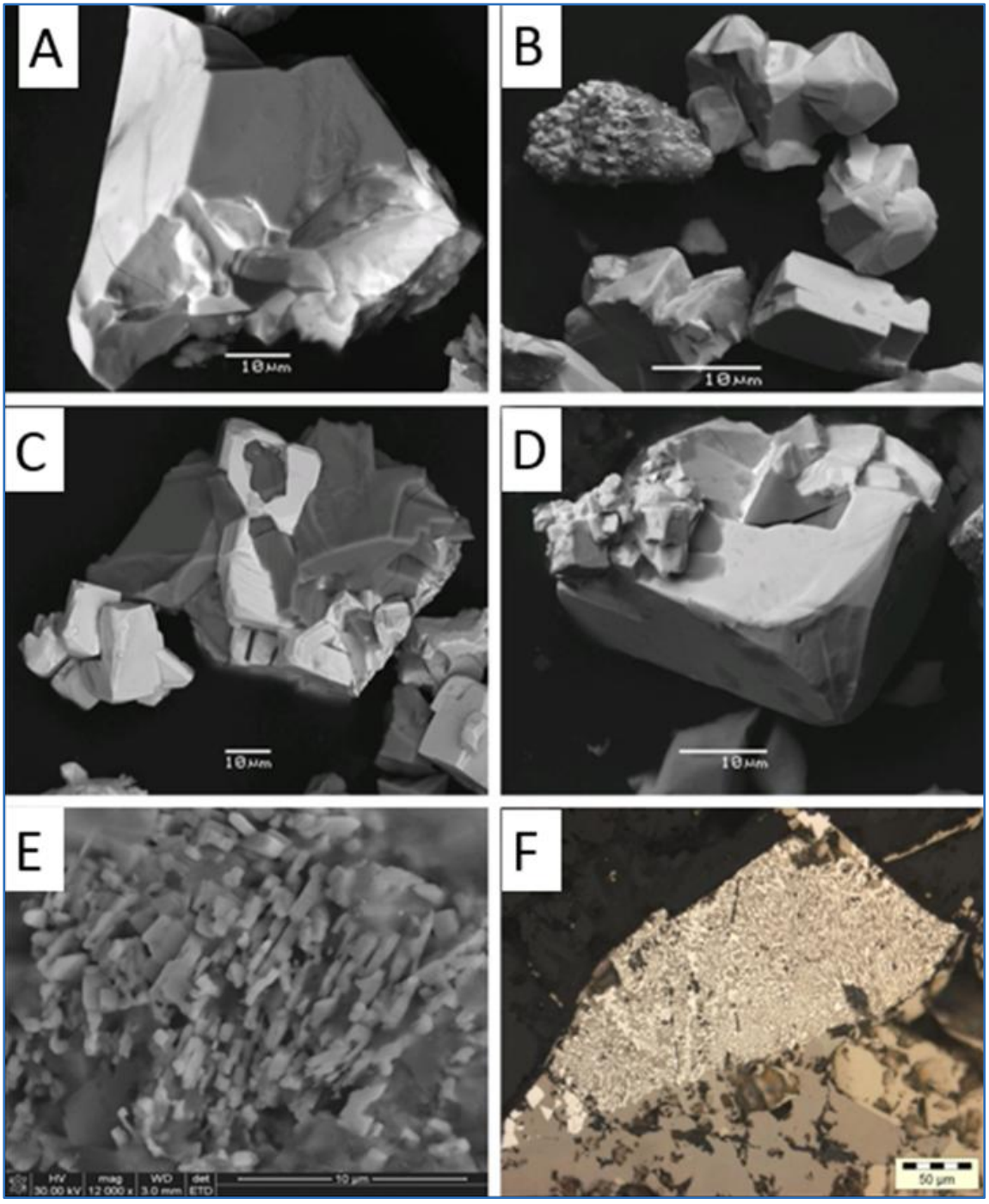
Figure 7.20 Mineral Assemblages found in the T2U



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 Group 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 Jeremy Witley, 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 source of PGMs mined from the UG2 and Merensky Reef in the Eastern and Western Limbs.
- 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 Jeremy Witley 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 Witley considers 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.

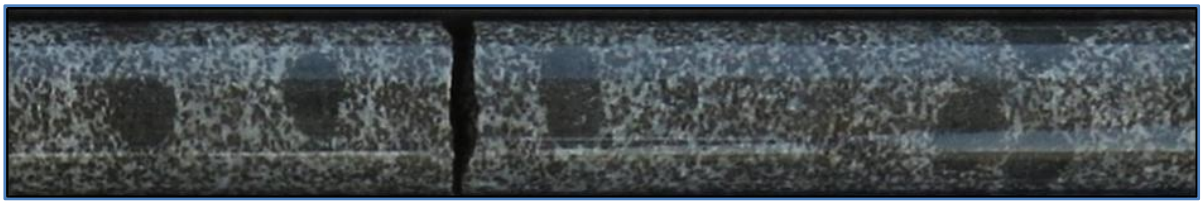
This initial exercise was expanded upon in 2013 to include near-surface information gained from close spaced drilling. The depth of weathering is controlled mainly by rock type, structure and alteration and is most pronounced along the mafic to ultramafic units and along the major fault traces at surface. 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 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.

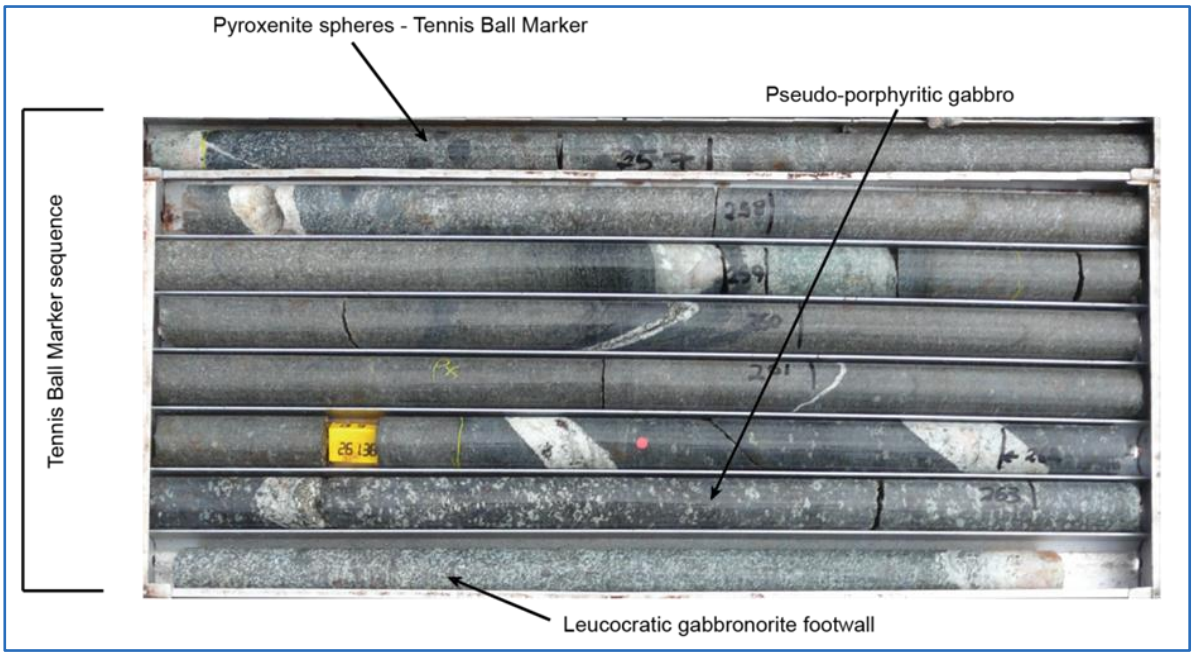
Well-defined anorthosite layering in the upper portion of the Main Zone, and the lower Main Zone layering (TBM and BMGN) are distinct and define the orientation of the layered intrusion (Figure 9.1 and Figure 9.2). This is a major distinction from the orientation of Platreef-type mineralised layers. 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 drillhole cores 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

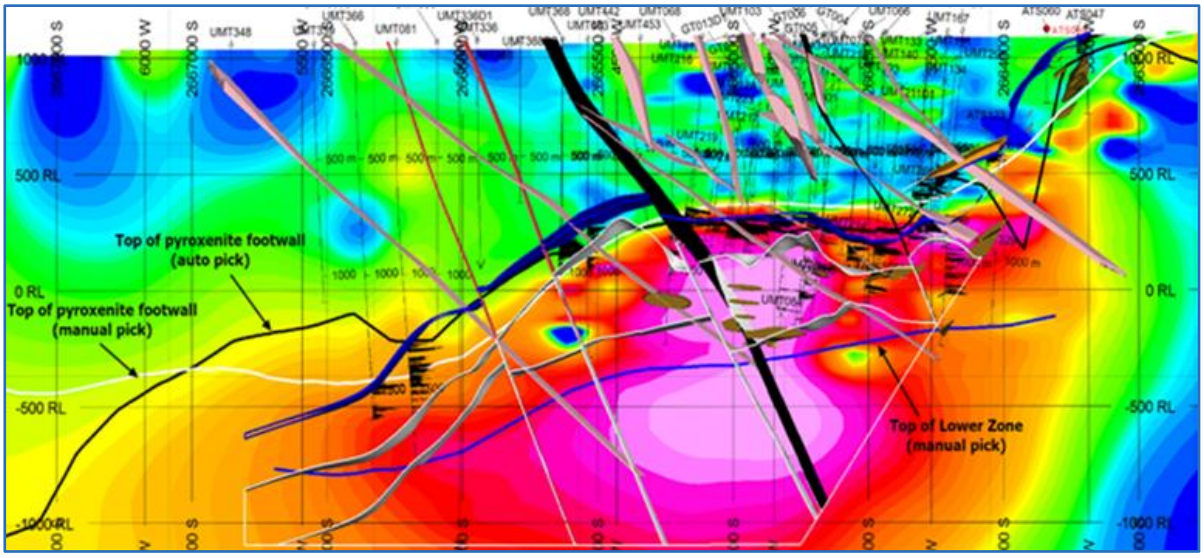
Geophysical survey methods at the Platreef Project have included aeomagnetics, gravity gradiometer and a number of downhole geophysical methods including caliper; self-potential (SP)/point resistance (PR); electrode-array-focussed resistivity (EAL); magnetic susceptibility (Msus); temperature/conductivity; 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 Q4'13 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.

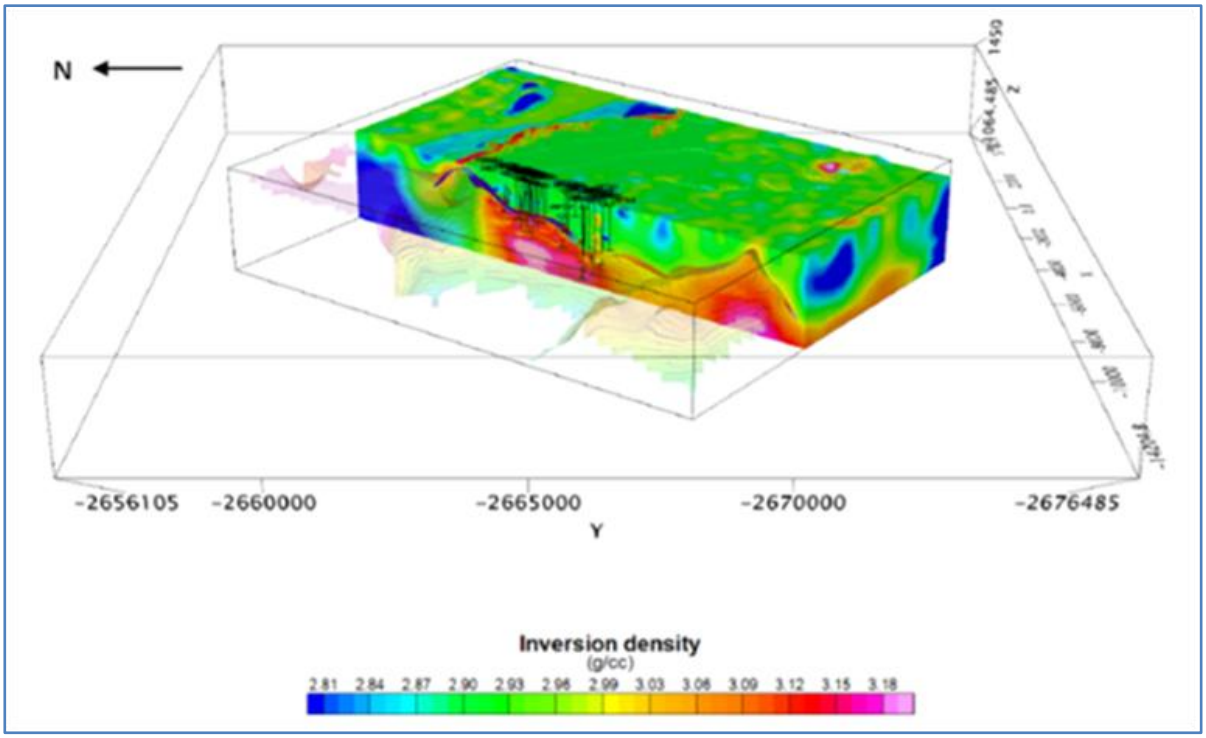
In Q1'15, Velseis (Pty) Ltd reprocessed the 3D seismic data acquired by CGG. The result of this work was a depth-converted volume constrained by the VSP data. Figure 9.5 shows a cross-section with a depth-converted seismic image showing the correlation between the xenoliths (drillhole discs) and strong 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)



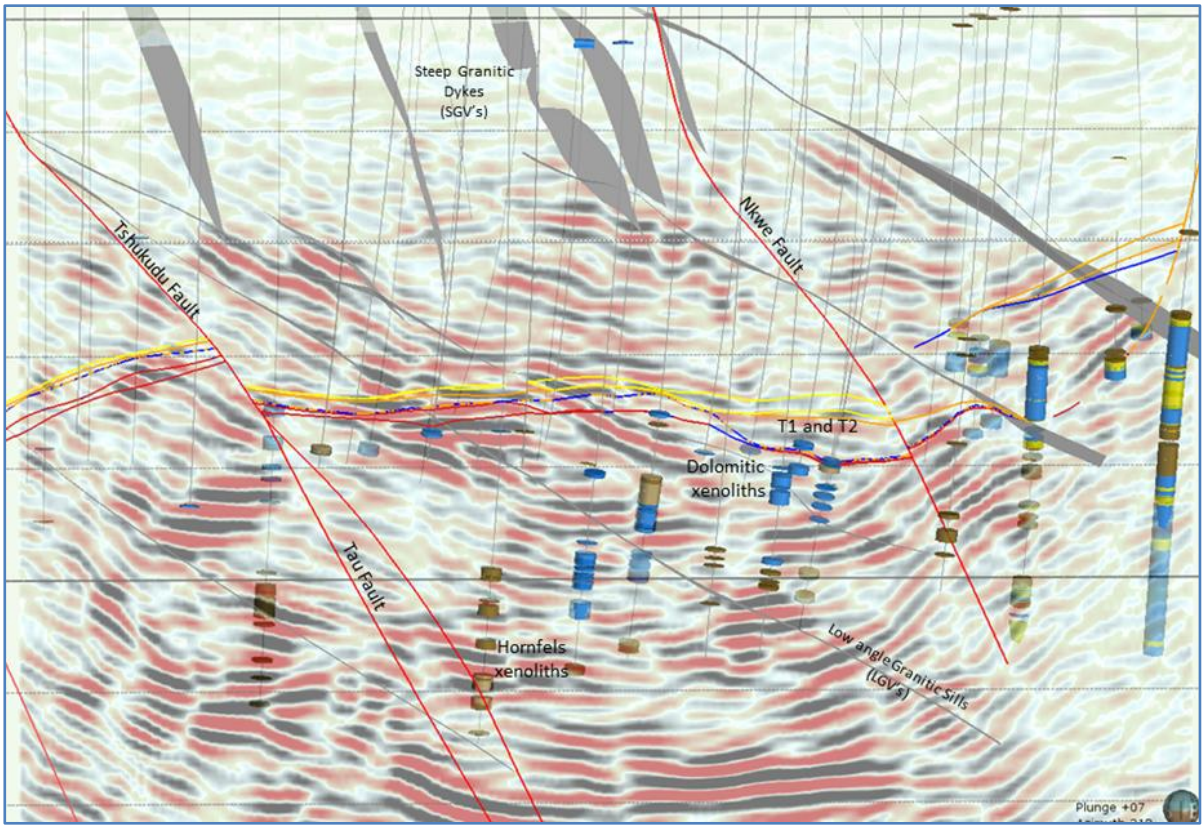
Ivanhoe, 2016; Dip Section 10 indicated on Figure 9.6.

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/cm^3 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.

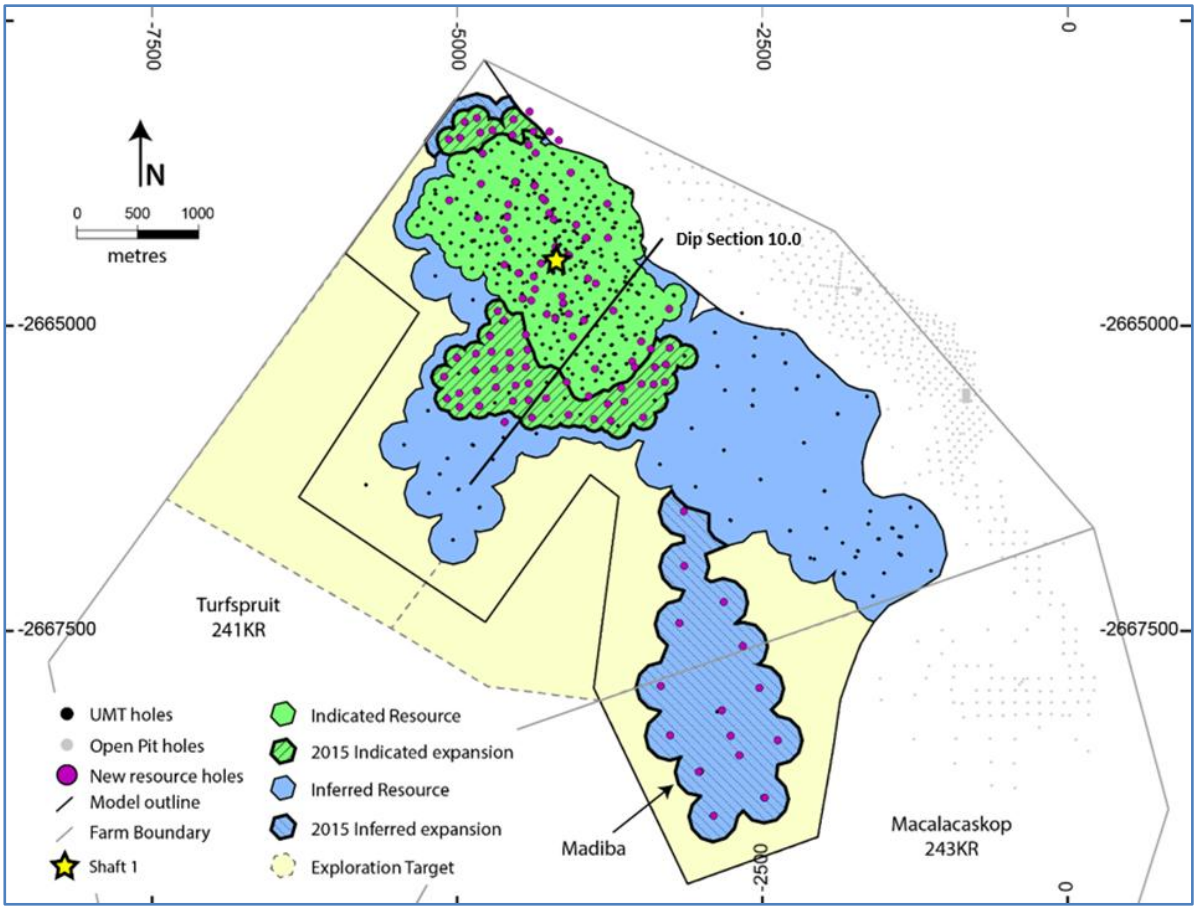
9.5 Petrology, Mineralogy, and Research Studies

Several MSc and PhD academic studies were conducted by various universities 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; Nodder 2015; Kvadsheim, 2017; Vekić, 2017; Mayer, 2018; Keir-Sage, 2019; Stephenson, 2019; and Abernethy, 2019). These major, trace and REE studies 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 Jeremy Witley, 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 February2015. 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.

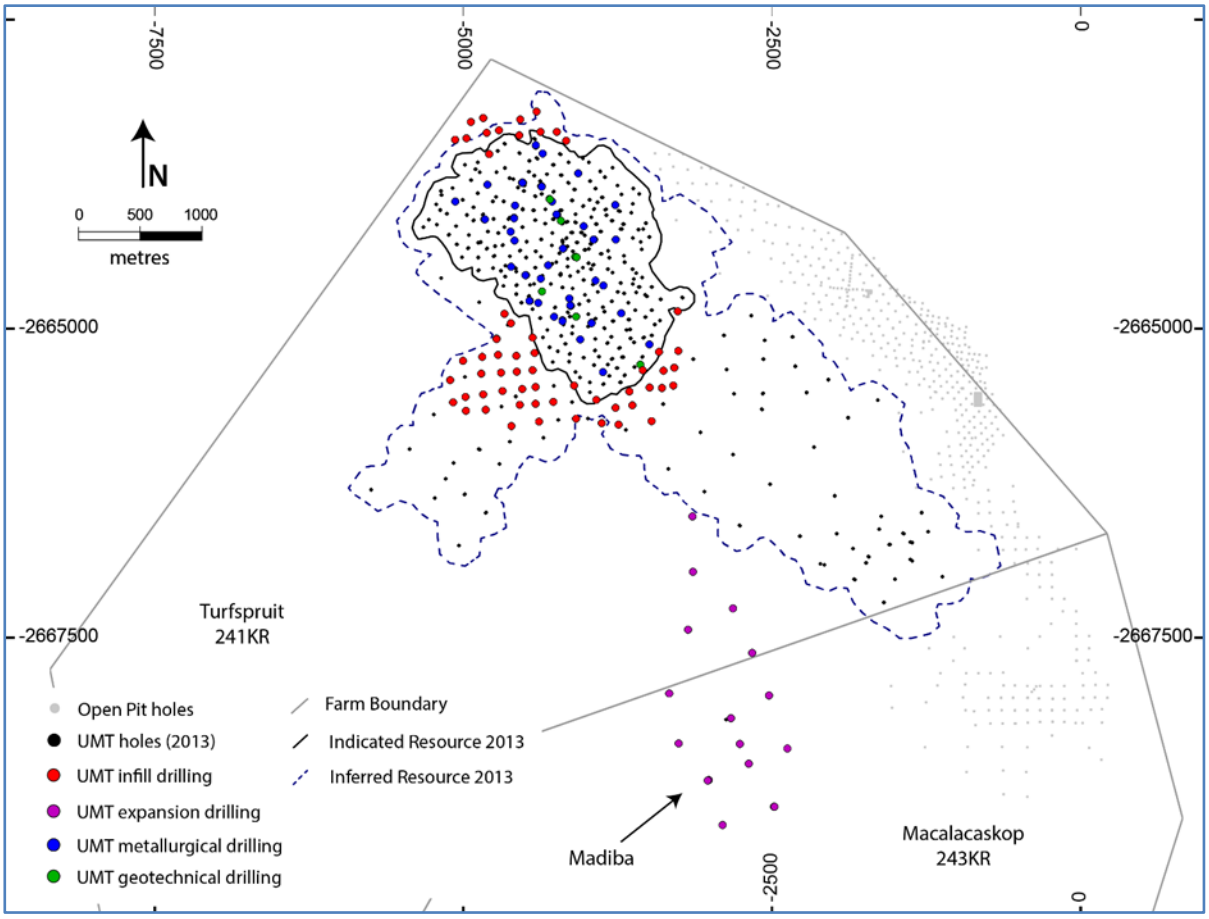
The database includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totalling 501,638 m completed by 11 February 2015. No drilling for resource estimation purposes has occurred between this date and the Report effective date; however, assay data from three 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 243 KR, 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 deposit 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 Zones 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 historical 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 prefabricated core racks for ease of access.

Mr Jeremy Witley has reviewed the local geology, including core logging and interpretations and finds 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 drillholes (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 were vertically collared and deemed to be vertical from collar to end of hole. The unsurveyed holes 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 geotechnical purposes and are less than 30 m in depth. Five drillholes were deflections with depths ranging from 28–780 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.

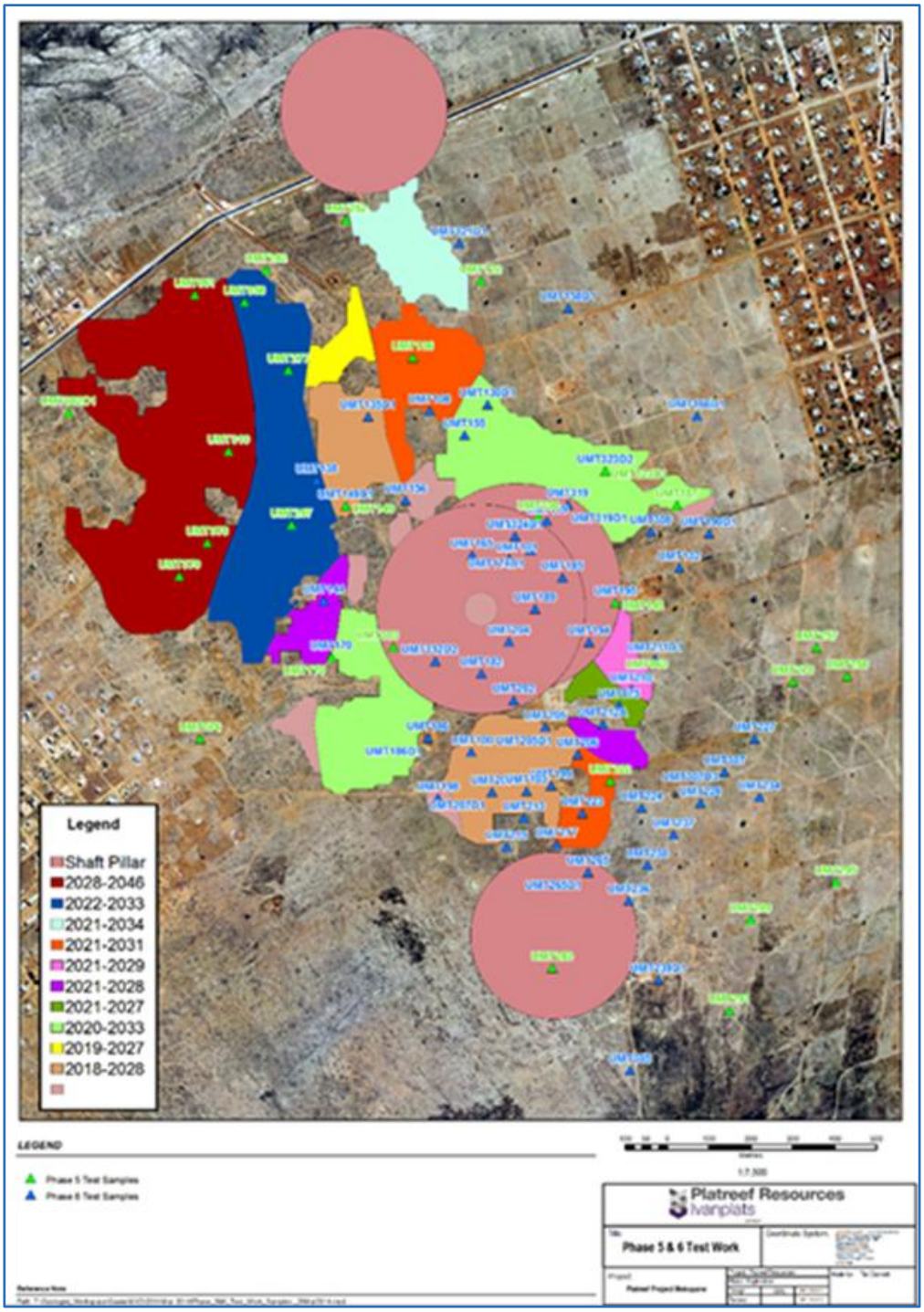
10.7 Metallurgical Drilling

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 drillhole 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

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 4.8–7.9 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. 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 and 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 MSA Comments

In the opinion of Mr Jeremy Witley, 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 historical open pit mineral resource area, vertical holes are 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, Mrs Louise Lintvelt, 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.

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 drill core data; therefore, geochemical sampling is not discussed further.

Drill core is sawn in half using a water lubricated rotating diamond impregnated core 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

11.1.1.1 AMK and ATS Sampling

AMK and AST drilling was completed to support the historical open pit mineral resource. 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).

11.1.1.2 UMT Sampling

For 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 blank samples. 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:

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

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 Relative Density by Stratigraphic Unit

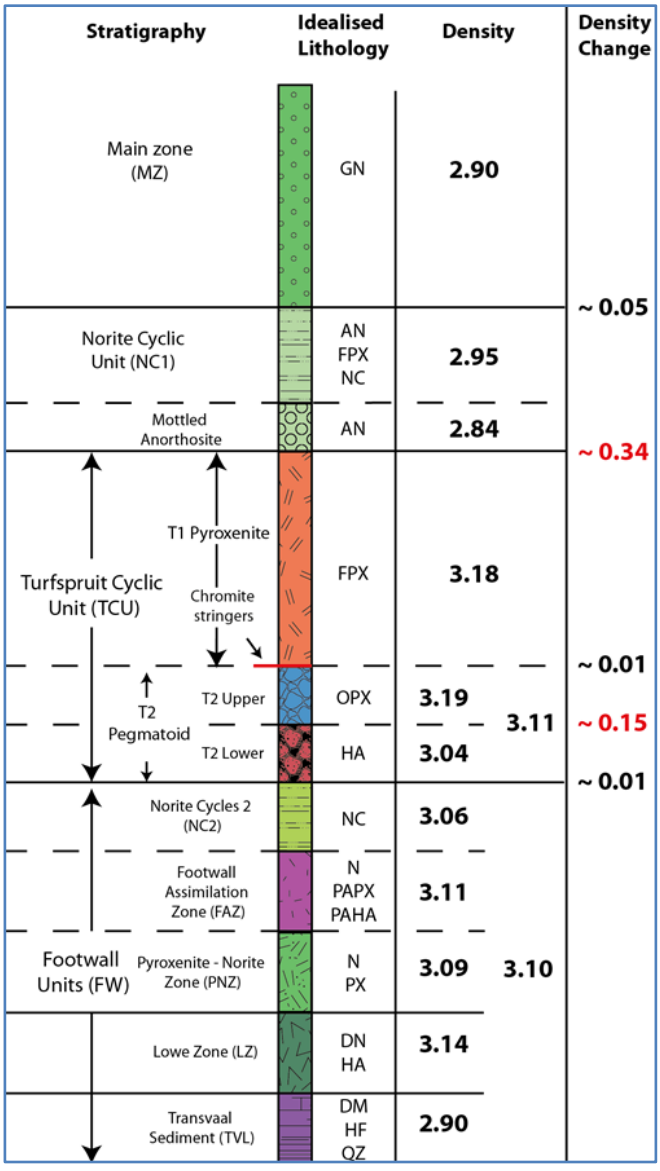
MSTRAT	Description	MSTRAT	MCODE	Number	Average	Minimum	Maximum	C.V.
Bikkuri Hanging Wall	Main Zone	BKHW	10	3,772	2.91	2.55	3.30	0.03
	Bikkuri Norite Cycles 1	BKNC1	11	244	2.98	2.62	3.26	0.04
	Bikkuri Mottled Anorthosite	BKMAN	12	31	2.83	2.63	2.95	0.02
Bikkuri	Bikkuri B1	B1	13	264	3.15	2.62	3.34	0.03
	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:

- 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 (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 (Mintek; 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 ISO 17025 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 ISO 9001.2000 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, OHSAS 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 Q3'11, 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 Q3'11, 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 has 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 previously considered 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

11.5.2.1 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.

11.5.2.2 Blanks

Blanks utilised locally sourced natural rock materials that have <10 ppb concentrations of Au, Pt, and Pd, but have Cu 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.

11.5.2.3 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.

11.5.2.4 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. 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.

11.5.2.5 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). These data are not used in Mineral Resource estimation for the underground deposit.

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 Jeremy Witley 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 12) 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 samples 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 on-site 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 reviews by previous Qualified Persons 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.

No new data have been collected since the last Wood reviews.

12.1 Current QP Site Visits

The current QP, Mr Jeremy Witley, visited the Project from 22 January to 26 January 2024. The purpose of the visit was to inspect drill core, review the sampling and logging, and gain familiarity with the Project through various workshops and discussions on the geological setting of the Mineral Resource. No exploration was being carried out at the time of the site visit, the last drilling having been completed in 2015.

12.2 Previous Site Visits

12.2.1 Site Visits by QPs During UMT Drilling

In the April 2010 site visit (Kuhl, 2010), the previous QP, Mr Timothy 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 the 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.

12.2.2 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 Parker visited the site on several occasions between 2011 and 2015. During these visits, Dr Parker reviewed geology logging, inspected core, verified collar coordinates, collected witness samples, and observed data collection programs.

12.3 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 12.5), MTS recommended 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. This data is primarily within the historical open-pit area and does not impact on the Mineral Resource reported in this technical report.

12.3.1 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.

12.4 UMT Database Reviews

Multiple reviews of the database were conducted by Mr Kuhl while still employed at Wood.

12.4.1 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.

12.4.2 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.

12.4.3 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.

12.4.4 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 at the time concluded that the drillhole database was acceptable to support Mineral Resource estimation.

12.4.5 October 2014 Data Review

Mr Kuhl 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.

12.4.6 May 2015 Data Review

Wood 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.

12.5 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 Q1'14. Mr Kuhl 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 Mr Kuhl 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 (these were drilled in the historical open pit mineral resource area).

12.6 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 QA/QC data reviews were conducted by Wood.

12.6.1 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). Ni 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 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.

12.6.2 UMT QA/QC March 2011–June 2012

Wood obtained and reviewed the available QA/QC for the period between March 2011–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 Cu and Ni 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.

12.6.3 UMT QA/QC June 2012–July 2014

Wood obtained and reviewed the available QA/QC for the period between June 2012–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.

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.

12.6.4 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–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. Cu 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 Au as obtained by Ultra Trace. Acceptable precision was defined as having an absolute relative difference at the 90th percentile of 20%. Au was observed to have a difference of 31-37%.

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.

12.7 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.

12.7.1 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.

12.7.2 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.

12.7.3 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.

12.7.4 2015 Witness Samples

Wood obtained a fourth set of 20 witness samples from the most 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.

12.8 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.

12.9 Gold Variability Testwork

Acuity (Acuity, 2015a) conducted a number of tests to investigate the high variability of Au lead collector fire assays and to recommend improvements for sample preparation and assaying work. QA/QC monitoring work has shown that Au CRM and duplicate assays typically show high variability for all grade ranges, whereas companion Pt and Pd 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 Au 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 Au 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 Au values, and the variability increase is noticeable at lower Au values. Moreover, the variability of Au generally decreased with increasing flux litharge content.
- Pt and Pd grade patterns are not as well-developed as for Au, 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 12.8) that Long (2011b) thought that harzburgite samples were already being ground to 75 µm.

12.10 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. Au, however, is slightly lower by the NiS fusion method.

12.11 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's investigation of Mintek's 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 Ni values, such as around 1,000 ppm, which is the tails assay for Ni; 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).

12.12 Comments on Section 12

Wood was 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 then at MTS, Mr Kuhl was involved in the Platreef Project since 2010. Minor problems have been identified and resolved by improving procedures at the site. It was the opinion of Mr Kuhl and the QA/QC and database specialist Mr Reid, that sufficient verification was conducted to provide support that the data collected are suitable for use as a basis for Mineral Resource estimation.

Mr Kuhl 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, although these are drillholes in the historical open pit resource area and not relevant to the current underground Mineral Resource.

Based on the results of the extensive testwork and interrogation of QA/QC data documented in Item 12 of this Technical Report, the Qualified Person is of the opinion the Pt, Pd, Au, Ni, and Cu assay results for the underground Platreef Project have sufficient accuracy and precision to support Mineral Resource estimation. The various data reviews completed demonstrated that the database is robust and suitable as an input to support Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The Platreef Resource has a long history of metallurgical testwork done by various parties and various metallurgical testwork campaigns have been conducted since October 2001 on mineralised material 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 Ni, 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.

This was followed by a series of metallurgical testwork campaigns on Platreef mineralised material over the period 2010–2022. This, named Phase 1 to Phase 9, testing included comminution characterisation, bench and mini pilot scale flotation in combination with laboratory scale dewatering testwork. The findings from this, Phase 1–9 testing were presented in the Platreef 2015 PFS, 2017 DFS and 2022 PDP FS, and can be summarised as follows:

- Prior to 2006: Testwork was predominantly conducted on lower grade platinum group element (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 HG PGE material.
- 2015 PFS: A series of metallurgical testwork campaigns were carried out on the Platreef mineralised material over the period 2010 to 2014. This testwork, named 'Phase 1 to Phase 6', included comminution characterisation, bench-scale flotation testing and laboratory scale dewatering testwork. The findings from Phase 1 to Phase 6 testing were presented in the Platreef 2015 PFS.
- 2017 DFS: The 2015 to 2016 FS metallurgical testwork conducted as part of the 2017 FS was performed on approximately 1,286 kg of HQ drill core (comminution) and 1,140 kg of quarter PQ drill core (Flotation) samples originating from the Platreef deposit. Samples were selected to represent the major ore types and ore blends expected to be processed over the life-of-mine (LOM). The aim of the testwork was to further characterise the flotation response, support flow sheet development, and to evaluate the degree of variability that could be expected across the deposit. The 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. The findings were presented in the Platreef 2017 DFS.

- 2022 PDP FS: Further testwork was conducted in 2020 and 2021 as part of the PDP FS. The aim was to evaluate the potential for inclusion of a high-pressure grinding roll (HPGR) circuit, to further characterise the flotation response of Platreef composite samples, evaluate the potential for Jameson cell technology in the cleaner flotation circuit, determine the effect on flotation response when using site water, and produce bulk concentrate samples for settling, filtration and Kell hydrometallurgical refining testwork. The PDP testwork was conducted on approximately 2.9 t of un-crushed drill core sample interval remainders from the 2017 FS variability testwork drilling campaign and two bulk samples totalling 47 t, comprised of crushed rock from surface stockpiles. The findings were presented in the Platreef 2022 PDP FS.

Further flotation and tailings dewatering testwork conducted in 2022 and 2024 as part of the current Platreef 2025 IDP included the following:

- Bench-scale flotation testwork on a 60 kg drill core composite sample remainder from the 2022 PDP FS test campaign which had been kept in cold storage. The aim was to further evaluate the potential to incorporate Jameson cell technology in the cleaner circuit.
- Bench-scale tailings settling and filtration testwork, which aimed to further characterise the tailings dewatering circuit design and performance parameters. The testwork was conducted on flotation tailings samples from the 2022 flotation testwork campaign.

Testwork has been conducted on samples representing the geometallurgical units T1 (feldspathic pyroxenite), T2U (poikilitic and/or pegmatoidal orthopyroxenite), T2L (typically olivine bearing and may vary from olivine-bearing orthopyroxenite to poikilitic harzburgite) as well as the footwall contaminated zone (CZ). The CZ zone has appropriately been reclassified as the footwall assimilated zone (FAZ). The majority of the testwork was conducted on drill core samples and also included testing of bulk samples from surface stockpiles for the 2022 PDP FS mini pilot plant (MPP) and HPGR test campaigns. The selection of samples, done in conjunction with the mining and geological teams, submitted for the metallurgical testwork are deemed to be sufficient.

Comminution testwork during both the 2015 PFS and 2017 DFS indicated that the samples can be characterised as being hard to very hard with a Bond Ball Work Index (BBWi) in the range of 19.0–24.2 kWh/t and is thus not suitable for semi-autogenous grinding milling. The comminution variability testwork showed only minor variation in hardness. The abrasion index (Ai) tests indicated that the samples could be classified as having a medium abrasion tendency.

Multiple single-pass HPGR crushing tests by Thyssenkrupp using a semi-pilot scale HPGR (SMALLWAL) were conducted on a bulk shaft intercept composite sample sourced from surface stockpiles during the 2022 PDP FS testwork programme. The HPGR testwork confirmed that Platreef sample was amenable to HPGR achieving a specified throughput of 310–338 t.s/m^{3}h at a specific energy of 0.94–1.47 kWh/t.

Based on the testwork, a three-stage crushing circuit and ball mill circuit has been recommended as the basis of design. The crushing circuit is comprised of primary Jaw crushing (within mining scope), secondary cone crushing and a tertiary HPGR circuit. The 85th percentile comminution test data were used as the design basis for the crusher and grinding circuit. The average ore hardness and abrasion index data have been used to derive the operating cost estimate.

The flotation testwork programme, over various phases, has included bench-scale, flow sheet development testwork, in combination with variability testwork (open circuit and locked cycle) using industry-standard testing techniques. Additionally preliminary Mini Pilot Plant (MPP) testwork was conducted to produce bulk concentrate for settling, filtration and Kell hydrometallurgical refining testwork.

Flotation testwork has confirmed an optimal flotation circuit feed size of 80% passing 75 µm and a flotation feed solids concentration of 30% w/w. Various flow sheet configurations and reagent regimes have been tested as follows:

- The 2015 PFS was based on an MF1 flow sheet utilising an oxalic acid and thiourea reagent regime and conventional cleaning technology.
- Further optimisation in the 2017 DFS confirmed the MF1 flow sheet using conventional cleaning technology, however the reagent regime was changed to a more conventional combination of thionocarbamate copper collector and sodium isopropyl xanthate (SIPX) as the primary collectors which is referred to as the Copper collector reagent suite.
- Preliminary MPP runs during the 2022 PDP FS indicated that sodium isobutyl xanthate (SIBX) reagent is a potential alternative to the copper collector reagent suite. This was also noted during the flow sheet development testing but was not adopted due to froth stability concerns. The mini pilot plant commissioning however, achieved adequate froth stability, but performance was not fully optimised and did not fully replicate the copper collector reagent suite performance.

The copper collector reagent regime has been more comprehensively tested and optimised and has thus been selected as the basis of design. The alternative SIBX reagent suite may be considered for further trials during full scale operation as a potential optimisation opportunity.

Bench-scale flotation testwork conducted at three independent laboratories confirmed that the type of grinding media used was critical, highlighting that the flotation feed Eh is negative when the sample is milled using carbon steel grinding media. A positive Eh prior to flotation was achieved by using high chrome grinding media which consistently returned superior flotation results compared to tests using carbon steel media.

Open circuit bench-scale flotation variability tests showed minor variation between the performance for the individual domain point samples (T1, T2L, T2U and CZ) and various ore blend composites. 3PE+Au recovery was however found to be dependent on the 3PE+Au head grade and target concentrate grade. For Platreef, as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). It was however also shown that under an excessively high mass pull regime, concentrate grades are significantly lower with limited opportunity for any gain in recovery. A 3PE+Au concentrate grade of 85 g/t could be achieved for head grades in the range 3.2–4.6 g/t where both the open circuit and locked cycle bench-scale variability testwork indicated that targeting higher concentrate grades in excess of 100 g/t (3PE+Au) resulted in reduced metal recoveries. Concentrate grades in excess of 100 g/t could only be achieved for head grades in the range 4.6–8.8 g/t.

Bench-scale Jameson cell flotation testwork (open circuit and locked cycle) at XPS in 2022, has shown that a single Jameson cell was able to replicate or better the performance of two stages of conventional cleaning. Jameson flotation technology has thus been recommended for the final HG cleaner, MG cleaner and LG re-cleaner duties.

The impact of site water has also been evaluated where comparative testwork on two composite samples using filtered site water from the Masodi grey water system achieved a similar 3PE+Au upgrade profile as the baseline tests using laboratory tap water. There was, however, notable variance in the ranges for repeat baseline tests.

Optimised locked cycle flotation testing on drill core sample blend composites with a 3PE+Au feed grade ranging from 2.4–4.9 g/t and averaging 3.8 g/t, achieved recoveries (3PE+Au) ranging from 83.1–88.7% at a final concentrate grade of 60–95 g/t. An average 3PE+Au recovery of 85.2% was achieved and the concentrate grade averaged 83 g/t 3PE+Au.

Mini pilot plant (MPP) testwork, was conducted at Mintek in June 2021 and November 2021 with the primary aim of producing bulk concentrate samples for Kell hydrometallurgical refining and concentrate de-watering testwork. Additionally, the intention was to derive process design information to supplement the design data derived from the bench-scale flotation testwork. The latter objectives were only partially achieved as the circuit was not adequately commissioned, stabilized and optimised due to several operational challenges. The results are deemed to reflect a non-optimised, preliminary commissioning performance achieving an average concentrate grade of 57 g/t at 87% recovery (2PE+Au) for the November 2021 run on a bulk sample with a head grade of 3.5 g/t 2PE+Au.

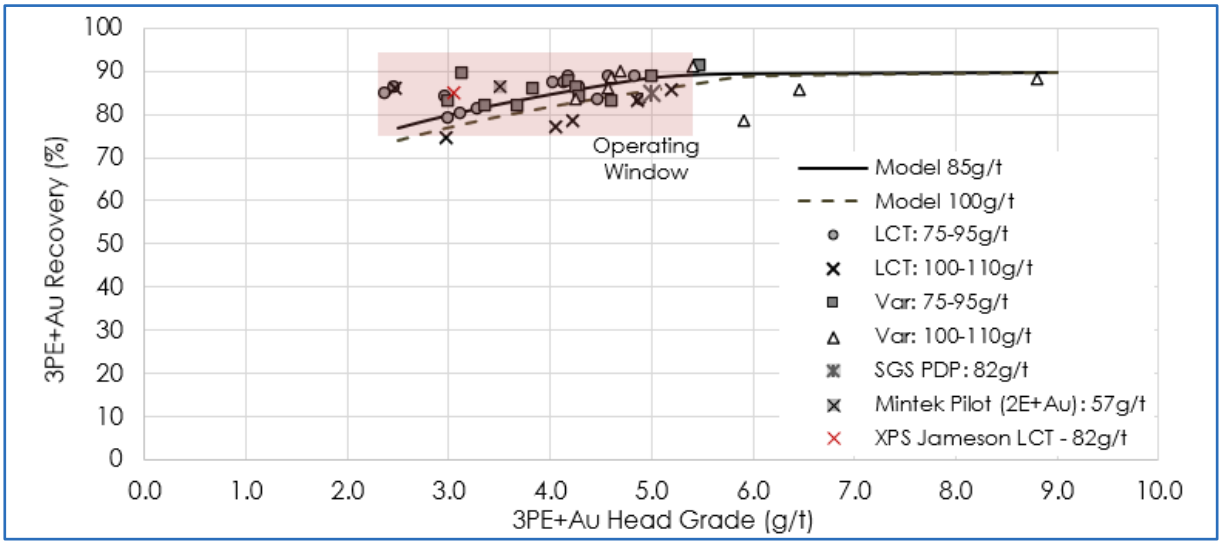
Based on the testwork findings, the flotation circuit flow sheet includes a rougher and cleaner flotation circuit to treat the mill product (P_{80} of 75µm) for recovery of 3PE+Au, Ni and Cu minerals to the final concentrate. The cleaner flow sheet reflects a split configuration treating the fast, medium and slow-floating fractions separately and incorporates Jameson Cell dilution cleaning for the final cleaning duties. A scavenger circuit will treat the scavenger rougher concentrate and cleaner tailings to ensure optimal recovery.

Settling and filtration testwork on tailings and concentrates confirmed that the tailings and concentrate samples can be dewatered using conventional dewatering technologies:

- Settling testwork on tailings and concentrates recommended an optimal thickening flux of 0.30 t/m²h for the concentrate duty and 0.40–0.50 t/m²h for the tailings duty. A thickener underflow solids concentration of 60–65% w/w was achieved for both applications.
- Concentrate pressure filtration testwork achieved a product moisture of <14% w/w with filtration fluxes averaging approximately 600 kg/m²h.
- Tailings vacuum disc and vacuum belt filtration testwork achieved optimal filtration rates of 400–978 kg/m²/hr and a cake moisture content ranging from 16–23% w/w. Optimal results were achieved using the standard belt filtration test procedure at a feed solids concentration of 70–75 % w/w.
- Tailings pressure filtration tests produced filter cakes with a moisture content ranging from approximately 9–14% w/w. An average filter flux of approximately 270–330 kg/m²h was achieved at a feed solids concentration of >60% w/w.
- Flow moisture point (FMP) and transportable moisture limit (TML) tests have also been conducted on the uncemented dewatered tailings with the estimated values reported to be 14.8% w/w and 13.3% w/w respectively.

The 2017 DFS optimised locked cycle testwork for using the copper collector reagent suite was used to derive relationships between 3PE +Au head grade, mass pull and final concentrate grade. These were used to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as reported for the 2017 DFS and summarised in Figure 13.1.

Figure 13.1 3PE+Au Recovery as a Function of Concentrate Grade



DRA, 2024

Updated metallurgical performance projections have been derived for the concentrator plant based on the IDP25 mine plan, using the recovery algorithms previously developed for the 2017 DFS. The metallurgical performance projections, when applied to the mine schedule, have indicated that when targeting a 3PE+Au concentrate grade of 85 g/t, the 3PE+Au recovery is expected to average 85.8% over the LOM, while the copper and nickel recoveries are expected to average 87.3% and 70.9%, respectively. The concentrate sulfur content is expected to range from 8–22% and average 12–15%.

13.2 Previous Metallurgical Testwork

Various metallurgical testwork campaigns have been conducted on mineralised material 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 2022, a series of metallurgical testwork campaigns, described as Phase 1–9, were carried out on the Platreef mineralised material as summarised in Table 13.1.

Most of the Phase 1–9 comminution and flotation testwork programme was undertaken at Mintek in Johannesburg South Africa, with support from SGS in Lakefield Canada. Additionally, HPGR testwork was conducted by ThyssenKrupp in Johannesburg South Africa, concentrate settling and filtration testwork was conducted by Metso South Africa, tailings settling and filtration testwork was also conducted by SGS in Lakefield Canada and Vietti Slurrytec in Johannesburg South Africa.

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.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
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 replicate during commercial operation.
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.	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.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
			Dewatering and rheology characterisation testwork.	Dynamic settling tests predicted 0.10 and 0.014 m²/t/day thickener underflow unit area (TUFUA) and hydraulic (THUA) unit areas, respectively. 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%), Au (17%) and alloys (14%). Liberated PGEs in the rougher tailings were all in the <10 µm size fraction, which are regarded as non-recoverable by flotation.

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, CU and Ni 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.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 7 (2015–2017) Managed by Ivanplats and DRA	Mintek and Vietti Slurry Technology (Tailings dewatering)	Comminution variability testing was conducted on approximately 1,286 kg of HQ drill core representing T1, T2U, T2L and CZ. Flotation testing was conducted on approximately 1,140 kg of quarter PQ drill core samples representing the geometallurgical units T1, T2U, T2L and CZ (subsequently renamed footwall assimilation zone (FAZ)). Testing was conducted on domain point samples, point sample blend composites, domain composites as well as ore blend composites representing the expected mine blend ratios.	Comminution variability testwork (CWi, Ai, BRWi, BBWi) Chemical Composition and Mineralogy. Bench-scale open circuit flotation testwork focused on further development of the MF1 copper collector flow sheet, sulfur optimisation, concentrate grade optimisation and flow sheet validation testing. This was followed by extensive open circuit variability testing and locked cycle testing using the optimised flow sheet. Tailings thickening testwork.	Comminution variability testwork during both the 2015 PFS and 2017 FS showed only minor variation in hardness and indicated that the samples can be characterised as being hard to very hard. The Bond Ball Work Index (BBWi) was in the range of 19.0 kWh/t to 24.2 kWh/t and the material is thus not suitable for semi-autogenous grinding milling. The abrasion index (Ai) tests indicated that the samples could be classified as having a medium abrasion tendency. Mineralogy on 22 variability intervals indicated that pyroxene is the predominant mineral phase present followed by minor to trace amounts of olivine, dolomite, magnetite, serpentine, clay minerals, quartz, amphibole, chlorite, mica, plagioclase, talc, pentlandite, chalcopyrite and pyrrhotite. Pentlandite and pyrrhotite were the dominant sulphide mineral phases present, with minor to trace amounts of chalcopyrite and pyrite. PGE-tellurides were the dominant PGE species, averaging 59% by volume in the samples tested, followed by lower amounts of PGE-arsenides, PGE-sulphides and laurite in minor to trace. The majority of PGEs identified reported to the 0–15 µm size class and were generally well liberated with the exception of the CZ samples which typically showed PGEs associated with relatively large gangue particles. A significant portion of slow-floating PGEs were identified, reflecting 3%–55% (by volume). Flotation testwork confirmed an optimal flotation circuit feed size of 80% passing 75 µm and a feed solids concentration of 30% w/w and showed that the MF1 flow sheet copper collector reagent regime was able to replicate the performance of the MF1 flow sheet and oxalic acid and thiourea reagent suite as adopted for the 2015 PFS. The reagent regime was thus changed to the copper collector reagent suite.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
		Tailings dewatering testwork was conducted on a flotation tailings blend sample from the Mintek flotation testwork campaign.		Eighty open circuit bench-scale flotation tests on samples representing T1, T2U, T2L, CZ and various ore blends showed minor variation between the performance for the individual domain point samples and the blend composites. The testwork showed that targeting concentrate grades >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. A 3PE+Au concentrate grade of 85 g/t was achieved for head grades in the range 3.2–4.6 g/t, while concentrate grades in excess of 100 g/t could be achieved for head grades in the range 4.6–8.8 g/t. Locked cycle testing confirmed the strong dependence between recovery, head grade and target concentrate grade, achieving recoveries (3PE+Au) in the range of 83.1–88.7% at a smelter grade final concentrate of approximately 60–95 g/t (3PE+Au) using the optimised conditions. Tailings thickening testwork recommended the use of Magnafloc 5250 flocculant at a dosage rate of 30 g/t. The underflow solids concentrations decreased from 66 to 57% (w/w) as the solids flux rate was increased from 0.3 to 0.9 t/m^2/h. A thickening flux of 0.4–0.5 t/m^2/h to target an underflow solids concertation of >65% w/w was deemed to reflect the optimal operating conditions. The maximum mud bed solids concentration that could be achieved in the bench-top dynamic unit after a 24 hour period was 71.7%m for high-rate thickening. The results were in good agreement with the tailings settling testwork at SGS during the PFS.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 8 (2020) Managed by Ivanplats and DRA	SGS Lakefield	Approximately 70 kg PDP composite sample comprised of 22% T1, 47% T2U, 26% T2L and 5% CZ comprised of un-crushed PQ quarter drill core remainders from the 2017 DFS variability testwork drilling campaign.	Bench sale open circuit and locked cycle flotation testwork to characterise the flotation response of the 2020 PDP composite sample, evaluate the flow sheet performance for a two of potential toll processing plant options and investigate the potential application of Jameson flotation cells for the final concentrate cleaning duty.	A bench-scale locked cycle flotation test for the optimised Platreef flow sheet as per the 2017 DFS achieved a 3E+Au recovery of 85% at a smelter-grade final concentrate of approximately 82 g/t. A Cu recovery of 91% was achieved, while for Ni, a recovery of 71% was achieved. This result is aligned to the performance achieved in the 2017 DFS, albeit on the lower end of the range. Comparative MF2 bench-scale locked cycle flotation tests to replicate the flow sheet of a potential toll processing plant, which was under consideration at the time, achieved a 3E+Au recovery of 85% at a smelter-grade final concentrate of approximately 61 g/t. A Cu recovery of 97% was achieved, while for Ni, a recovery of 76% was achieved. A further locked cycle test using a modified flow sheet to target a lower mass pull achieved a 3E+Au recovery of 86% at a smelter-grade final concentrate of approximately 113 g/t. A Cu recovery of 90% was achieved and a Ni recovery of 74% was achieved. The testwork programme also included two Jameson cell scouting tests which were inconclusive due to the use of an outdated test procedure that did not include adequate frother addition.
Phase 9 (2021) Managed by Ivanplats and DRA	Mintek (Flotation) Thyssen Krupp (HPGR)	A high grade core composite (0.84t) comprised of 17% T1, 39% T2U, 35% T2L and 9% CZ. A LOM grade core composite (2.1t) comprised of 24% T1, 40% T2U, 29% T2L and 8% CZ.	Bench sale open circuit and testwork to characterise the flotation response of both the core composite and bulk samples and evaluate the effect of using site water.	The optimal open circuit flotation tests results achieved an open circuit 3PE+Au recovery of 70% to 80% at a concentrate grade ranging from 77 to 96 g/t. These open circuit flotation tests achieved upgrade profiles within the expected ranges observed in the 2017 DFS and 2020 SGS testwork but were noted to fall on lower end of the range. Comparative open circuit flotation testwork the HG and LOM core composite using filtered site water from the Masodi grey water system achieved a similar 3PE+Au upgrade profile as the baseline tests using laboratory tap water. There was, however, notable variance in the ranges for repeat baseline tests.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
	MO Group (Concentrate dewatering) and Green Technical (Concentrate materials handling testwork)	Two bulk samples for which the composition was unknown weighing approximately 23 tonnes and 24 tonnes respectively. The drill core composite samples were prepared using reflect un-crushed drill core remainders from the 2017 FS variability testwork drilling campaign, while the bulk samples reflect of mined crushed rock samples from the mining shaft intercept reclaimed from surface stockpiles. Bulk Concentrates generated from the preliminary pilot plant runs were used for the concentrate dewatering and materials handling testwork.	Mini pilot testwork, aimed to produce bulk concentrate samples for Kell hydrometallurgical refining 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 testwork. The latter objectives were only partially achieved as the circuit was not adequately commissioned, stabilized and optimised due to a number of operational challenges. Multiple single-pass crushing tests on shaft intercept bulk composite sample using a semi-pilot scale High Pressure Grinding Roll (HPGR). Concentrate settling and filtration and materials handling testwork.	A similar trend was observed for Cu and Ni. The bench-scale testwork programme also included Jameson cell scouting tests to evaluate the cleaner circuit upgrade potential using Jameson cell technology. Similar to the Phase 8, SGS testwork, the tests were inconclusive due to the use of an outdated test procedure which did not include adequate frother addition. An initial mini pilot plant (MPP) commissioning run produced a final concentrate with a 3PE+Au grade of approximately 30–35 g/t. The MPP run experience numerous operational challenges, most notably inconsistent dosing of the primary copper collector (low flows) and stoppages due to power outages resulting in instability with poor 3PE+Au, Cu and Ni balance accountability. Final concentrates from this campaign were collected and batch processed in a bulk flotation cell to produce concentrate for Kell hydrometallurgical refining and concentrate de-watering testwork. A second mini pilot plant (MPP) run 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, although improved, the copper collector dosage system was still not functioning optimally, and blockages were experienced in the final cleaner flotation cells. This contributed to large variances in final concentrate mass pull. The averaged metallurgical projection data indicated a 2PE+Au concentrate grade of 57 g/t at a recovery of 87% and 5.3% mass pull. A Cu recovery of 86% was achieved, and a Ni recovery of 80% was achieved. These results are considered to reflect preliminary commissioning runs which require further optimisation to achieve the target concentrate grade of 85 g/t at reduced mass pull aligned to optimised bench-scale test results.

Testwork Description	Laboratory	Samples	Scope	Summary of Key Findings
				Due to sample mass limitations, it was not possible to conduct any further MPP optimisation runs. The HPGR testwork confirmed that Platreef samples are amenable to HPGR achieving a specified throughput of 310–338 t.s/m3h at a specific energy of 0.94–1.47 kWh/t. Concentrate bench-top dynamic thickening tests indicated a decrease on underflow solids concentration from approximately 63 %w/w to 58% as the solids flux rate increased from 0.30 t/m²/h to 0.80 t/m²/h. The overflow clarities ranged from 250 mg/l to 300 mg/l. Concentrate bench-scale filtration testwork achieved a filtration flux of approximately 600 kg/m²h for all samples tested and confirmed the potential to achieve a final concentrate moisture of <14% (w/w). The 3PE+Au grade of the samples (30 to 60 g/t) was noted as being lower than the design requirement of 85 g/t; however, the Cu and Ni grades and particle size distributions were within the expected design ranges. The flow properties of the concentrate sample were also determined using primarily a Jenike shear, density/compressibility, and a chute friction angle tester.

The findings from the Phase 1–Phase 9 testing were presented in the Platreef 2015 PFS, 2017 DFS and 2022 PDP FS. The key findings which have been used to derive the flow sheet, process design parameters and metallurgical performance projections for the concentrator plant are summarised below.

Mineralogy on the 2017 DFS variability sample intervals indicated that:

- Pyroxene is the predominant mineral phase present followed by minor to trace amounts of olivine, dolomite, magnetite, serpentine, clay minerals, quartz, amphibole, chlorite, mica, plagioclase, talc, pentlandite, chalcopyrite and pyrrhotite.
- Pentlandite and pyrrhotite were the dominant sulphide mineral phases present, with minor to trace amounts of chalcopyrite and trace amounts of pyrite.
- PGE-tellurides are the dominant PGM species, averaging 59% by volume, followed by lower amounts of PGE-arsenides, PGE-sulphides and laurite in minor to trace amounts in only a few samples.
- The majority of PGEs identified reported to the 0–15 µm size class and were well liberated with the exception of FAZ/CZ samples which typically showed PGEs associated with relatively large gangue particles. The PGE fast floating fraction varied in the range of 40 to 97% (by volume) and averaged 66%. A significant portion of slow-floating PGEs were identified ranging from 3 to 55% (by volume).

Comminution testwork has characterised Platreef samples as being hard to very hard and not suitable for SAG milling. The comminution testwork findings can be summarised as follows:

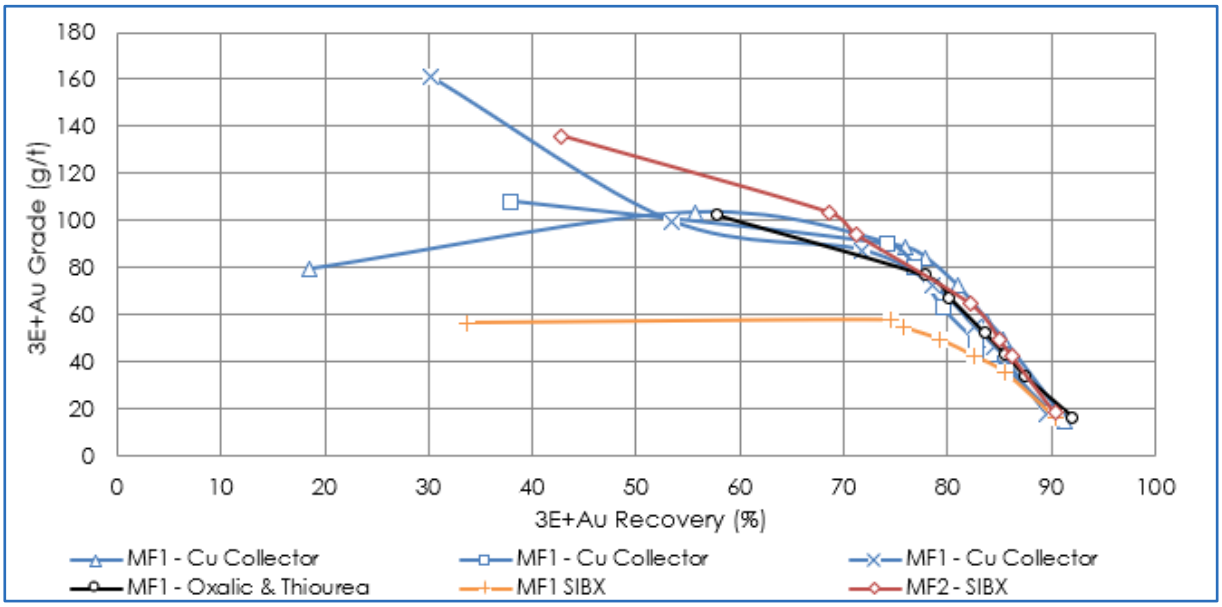
- Comminution variability testwork during both the 2015 PFS and 2017 DFS indicated that the samples can be characterised as being hard to very hard with a BBWi in the range of 19.0–24.2 kWh/t with only minor variation in hardness for the samples tested.
- The samples were classified as having a medium abrasion tendency.
- Laboratory-scale, Grindmill batch milling tests provided additional comminution characterisation data which by was used in combination with the BBWi and BRWi test results to derive the breakage parameters that have been used for the continuous ball mill simulations for design purposes.
- HPGR testwork confirmed that Platreef 2022 PDP bulk sample was amenable to HPGR achieving a specified throughput of 310–338 t.s/m^{3}h at a specific energy of 0.94–1.47 kWh/t.

Flotation testwork included bench-scale open circuit and locked cycle testwork in combination with preliminary MPP runs. The key findings can be summarised as follows:

- The optimal flotation circuit feed size and solids concentration was found to be 80% passing 75 µm and 30% w/w respectively.
- Bench-scale flotation testwork conducted at three independent laboratories confirmed that the type of grinding media used was critical. The tests 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 which consistently returned superior results compared to tests using carbon steel media.

- Flow sheet development testwork over the various study phases has considered both MF1 and MF2 flow sheet configurations and various reagent regimes comprising oxalic acid and thiourea, thionocarbamate copper collector/SIPX (copper collector) and SIBX as the primary collectors. As illustrated by the comparative grade-recovery curves for tests using conventional cleaning technology in Figure 13.2 below, the open circuit bench-scale flotation tests showed similar performance for the MF1 oxalic acid reagent suite, MF1 copper collector reagent suite and MF2 SIBX reagent suite. The MF1 SIBX reagent suite achieved an inferior upgrade profile which was attributed to the visual observation of poor froth stability. Additional trials of an MF1 SIBX reagent suite during the MPP commissioning runs showed improved froth stability relative to bench tests, but performance was not fully optimised and did not fully replicate the copper collector reagent suite performance.

Figure 13.2 Summary of Flotation Flow Sheet Options



DRA, 2024

- An MF1 flow sheet and copper collector reagent suite has been selected as the basis of design. The alternative SIBX reagent suite may be considered for further trials during full scale operation as a potential optimisation opportunity.

- Open circuit bench-scale flotation variability test data has shown minor variation between the performance for the individual domain point samples (T1, T2L, T2U and CZ) and various ore blend composites. 3PE+Au recovery was however found to be dependent on the 3PE+Au head grade and target concentrate grade. For Platreef, as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). A 3PE+Au concentrate grade of 85 g/t could be achieved for head grades in the range 3.2–4.6 g/t where both the open circuit and locked cycle bench-scale variability testwork indicated that targeting higher concentrate grades in excess of 100 g/t (3PE+Au) resulted in reduced metal recoveries. Concentrate grades in excess of 100 g/t could be achieved for head grades in the range 4.6–8.8 g/t.
- Comparative testwork on two composite samples using filtered site water from the Masodi grey water system achieved a similar 3PE+Au upgrade profile as the baseline tests using laboratory tap water. There was, however, notable variance in the ranges for repeat baseline tests.
- Optimised locked cycle flotation testing on drill core sample blend composites with a 3PE+Au feed grade ranging from 2.4–4.9 g/t and averaging 3.8 g/t 3PE+Au, achieved recoveries (3PE+Au) ranging from 83.1–88.7% at a final concentrate grade of 60–95 g/t. An average 3PE+Au recovery of 85.2% was achieved and the concentrate grade averaged 83 g/t (3PE+Au).
- Comparative locked cycle tests targeting high mass pull to improve recovery, showed a marginal recovery benefit of approximately 0.6% (3PE+Au) however concentrate grade was reduced from 83 g/t to 74 g/t (3PE+Au) highlighting that under an excessively high mass pull regime, concentrate grades are significantly lower with limited opportunity for any gain in recovery.
- Due to operational challenges, schedule constraints and sample mass limitations, the MPP was not successfully stabilized and optimised. Results are thus deemed to reflect a non-optimised, preliminary commissioning runs which achieved an averaged concentrate grade of 57 g/t at 87% recovery (2PE+Au).

Settling and filtration testwork on tailings and concentrates confirmed that the tailings and concentrate samples can be dewatered using conventional dewatering technologies:

- Settling testwork on tailings and concentrates recommended an optimal thickening flux of 0.30 t/m^{2}h for the concentrate duty and 0.40–0.50 t/m^{2}h for the tailings duty. A thickener underflow solids concentration of 60–65% w/w was achieved for both applications.
- Concentrate pressure filtration testwork on the concentrate achieved a product moisture of <14% w/w with filtration fluxes averaging approximately 600 kg/m^{2}h.
- Tailings vacuum filtration testwork by Golder for the 2017 DFS backfill design, recommended a cake loading rate of 609 kg/m^2/hr, feed solids concentration of 69 wt% solids, 90s cycle time and target moisture content of 17% w/w as the design basis for disc filter sizing.

Recovery algorithms were previously derived for the 2017 DFS based on the optimised locked cycle test results for the MF1 flow sheet and copper collector reagent suite. The modelling incorporates correlations between 3PE+Au feed grade, concentrate mass pull, concentrate 3PE+Au upgrade ratio (concentrate grade relative to feed grade) and recovery to derive nominal head grade recovery relationships for Pt, Pd, Rh and Au. Copper, nickel and sulfur recoveries correlations were also previously developed based on the recovery achieved to the combined final concentrate in locked cycle testing relative to the respective test head grade.

These recovery models (as a function of feed grade and targeted concentrate grade) for the individual metals as previously developed for the 2017 DFS are summarised in Table 13.2.

Table 13.2 Platreef Concentrator Recovery Algorithms

Element/ Metal	Recovery Algorithm	Head[1] Grade Range	Max Recovery %
Platinum	$\text{Recovery} = e^{4.1501+0.15166\times\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}$	1.0–2.7g/t	90.9
Palladium	$\text{Recovery} = e^{4.8401-\frac{0.79022}{\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}}$	1.2–2.7g/t	90.7
Rhodium	$\text{Recovery} = 52.042 + 19.041 \times Ln\left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)$	0.06–0.19g/t	84.9
Gold	$\text{Recovery} = 66.424 \times \left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.1137}$	0.18–0.35g/t	80.8
Copper	$\text{Recovery} = 129.78 \times Cu\ Head\ Grade^{0.2174}$	0.13–0.30%	93.6
Nickel	$\text{Recovery} = 125.33 \times Ni\ Head\ Grade^{0.5145}$	0.25–0.53%	82.2
Sulfur	$\text{Recovery} = 75.569 \times S\ Head\ Grade^{0.328}$	0.75–1.1%	76.8

[1]The head grade ranges have been updated to reflect the range of head grades evaluated in the testwork.

13.3 Platreef 2025 IDP Testwork Samples and Scope

In addition to the previous Phase 1–9 testwork, which formed the basis of the 2015 PFS, 2017 DFS and 2022 PDP FS, further flotation and tailings dewatering testwork was conducted in 2022 and 2024 as follows:

- Bench-scale open circuit and locked cycle flotation testwork which aimed to further evaluate the potential to incorporate Jameson cell technology in the cleaner circuit. The testwork was conducted in 2022 at XPS Expert Process Solutions in Falconbridge, Canada.
- Bench-scale tailings settling and filtration testwork, which aimed to further characterise the tailings dewatering circuit design and performance parameters for the paste backfill and Dry Tailings Storage Facility (DTSF) filter plants. The testwork was conducted on flotation tailings samples from the 2022 flotation testwork campaign at XPS. The testwork for backfill plant design was conducted in 2024 by Paterson and Cooke (P&C) in Cape Town, South Africa. Additionally supporting vendor tailings filtration testwork was also conducted by Metso and FLSmidth.

The testwork for backfill plant design was conducted by Paterson and Cooke (P&C) in Cape Town. Additionally supporting vendor tailings filtration testwork was also conducted by Metso and FLSmidth.

13.3.1 Flotation Testwork

The 2022 XPS flotation testwork was conducted on a on a 60 kg drill core composite sample remainder from the 2022 PDP FS test campaign at Mintek which had been kept in cold storage. The sample head assay is presented in Table 13.3.

Table 13.3 2022 XPS Flotation Testwork Sample Measured Head Assay

Sample ID	Grade									
	3PE+Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
2022 XPS Composite	3.14	1.47	1.36	0.08	0.23	0.13	0.30	0.67	23.7	8.9

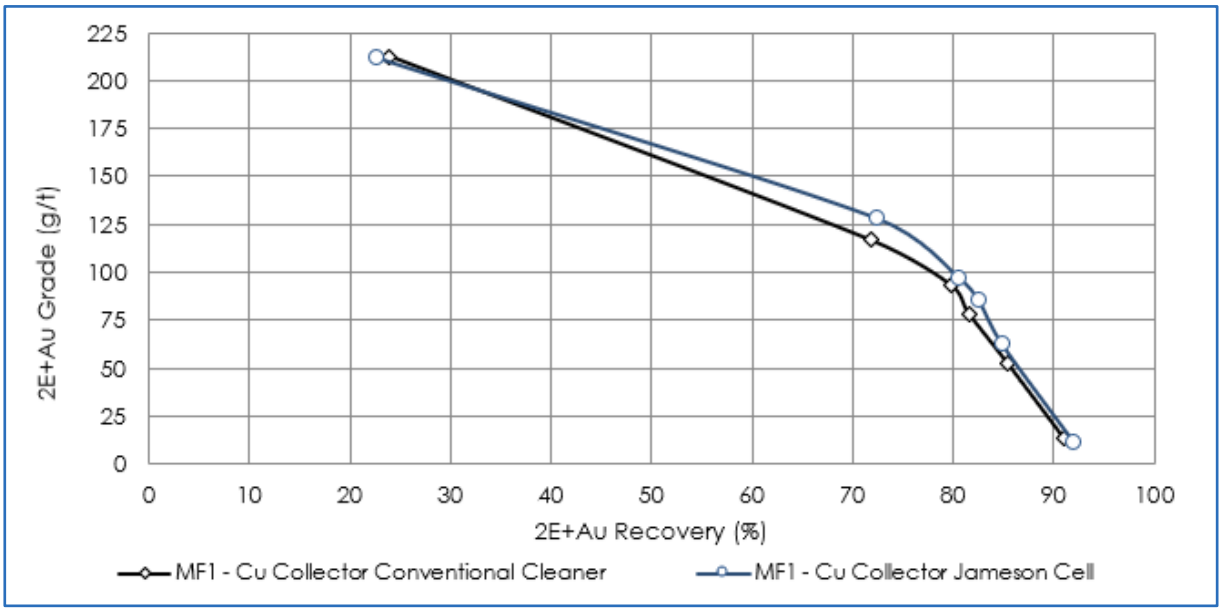
The assays were done in triplicate and averaged resulting in a measured 3PE+Au head grade of 3.14 g/t and contained 0.13% Cu and 0.30% Ni.

The testwork included a series of bench-scale open circuit flotation testing using the optimised MF1 flow sheet and copper collector reagent regime as developed for the 2017 DFS to baseline the sample performance. This was followed by Jameson dilution cleaning tests which aimed to replicate the Jameson Cell performance through “dilution cleaning” as per the specified vendor test procedure.

The cleaner cell sizes were selected to keep the cell density below 10% solids except for the scavenger cleaner. A key change to the procedure, as compared to previous Jameson cell testwork, included preconditioning process water with 20 ppm of frother prior to addition to the cell. The preconditioning was done to ensure stability of the frother through the various cleaning stages.

The open circuit bench-scale Jameson cell flotation testwork showed that a single Jameson cell was able to replicate or better the performance of two-stages of conventional cleaning as illustrated in the comparative grade-recovery curves presented Figure 13.3.

Figure 13.3 Flotation Performance Comparison Between Jameson Cells and Conventional Cleaners



Overall, the open circuit 2PE+Au recovery was comparable between the two flow sheets and the Jameson Cell dilution cleaner tests showed potential upside in the 3PE+Au concentrate grade. The base metal recoveries grade recovery curves for the optimised tests showed a similar trend however there was a wider spread in performance.

A single six-cycle locked cycle test was performed using the optimised MF1 flow sheet and copper collector reagent suite previously devolved for the 2017 DFS however the HG, MG and LG cleaner circuit incorporated the Jameson Cell dilution test procedure for the final concentrate cleaning duties.

The locked cycle test achieved steady state after 3 cycles. The outputs from cycles 4–6 outputs were used to derive a performance projection achieving a 2E+Au recovery of 84.7% at a combined final concentrate grade of 80.6 g/t. This is estimated to reflect an equivalent 3PE+Au recovery of 84.1% at a concentrate grade of 82 g/t 3PE+Au. The Cu and Ni concentrate grades were 3.7% and 6.8% respectively at 87% Cu and 70% Ni recovery. The final concentrate contained 19.2% sulfur, 24.4% Fe and 7.8% MgO.

The 3PE+Au concentrate grade and recovery of the XPS Jameson cell LCT was found to be aligned to the expected performance forecast by the recovery algorithms previously derived for the 2017 DFS. Similarly, the Cu and Ni recovery was also aligned to the 2017 DFS recovery algorithm performance projections.

The concentrate sulfur recovery of approximately 82% was above the upper end of the range (58–78 %) projected by the 2017 DFS recovery algorithms which highlighted an expected variance of ±10.0% between the modelled vs actual sulfur recovery. The LCT was noted to have a call factor of 112% for sulfur and the final concentrate sulfur grade of 19.2% falls within the range of 12–22% achieved for previous locked cycle tests at both Mintek and SGS. The high variance in sulfur grade and recovery as evidenced in testwork, is possibly due to the analytical methods employed.

Jameson cell locked cycle testwork also showed potential for a reduction in frother addition from 100 g/t to 52 g/t.

Based on the XPS testwork findings, using the updated vendor dilution cleaning test procedure, Jameson cells have been recommended for the HG cleaner, MG cleaner and LG Re-Cleaner duties.

13.3.2 Tailings Thickening and Filtration Testwork

13.3.2.1 Paterson & Cooke Tailings Thickening and Filtration Testwork

The Paterson and Cooke (P&C) tailings settling and filtration testwork was conducted on flotation tailings samples generated by XPS during the 2022 testwork programme. The testwork included material characterisation tests, bench top dynamic thickening tests, uncemented flow behaviour tests, filtration tests, bulk material handling tests, geotechnical tests, paste backfill strength tests and cemented flow behaviour tests.

The tailings sample was found to predominantly comprise enstatite (53%), plagioclase (13%), augite (11%), talc (7.9%) and muscovite (7.1%) with material characteristics as summarised in Table 13.4.

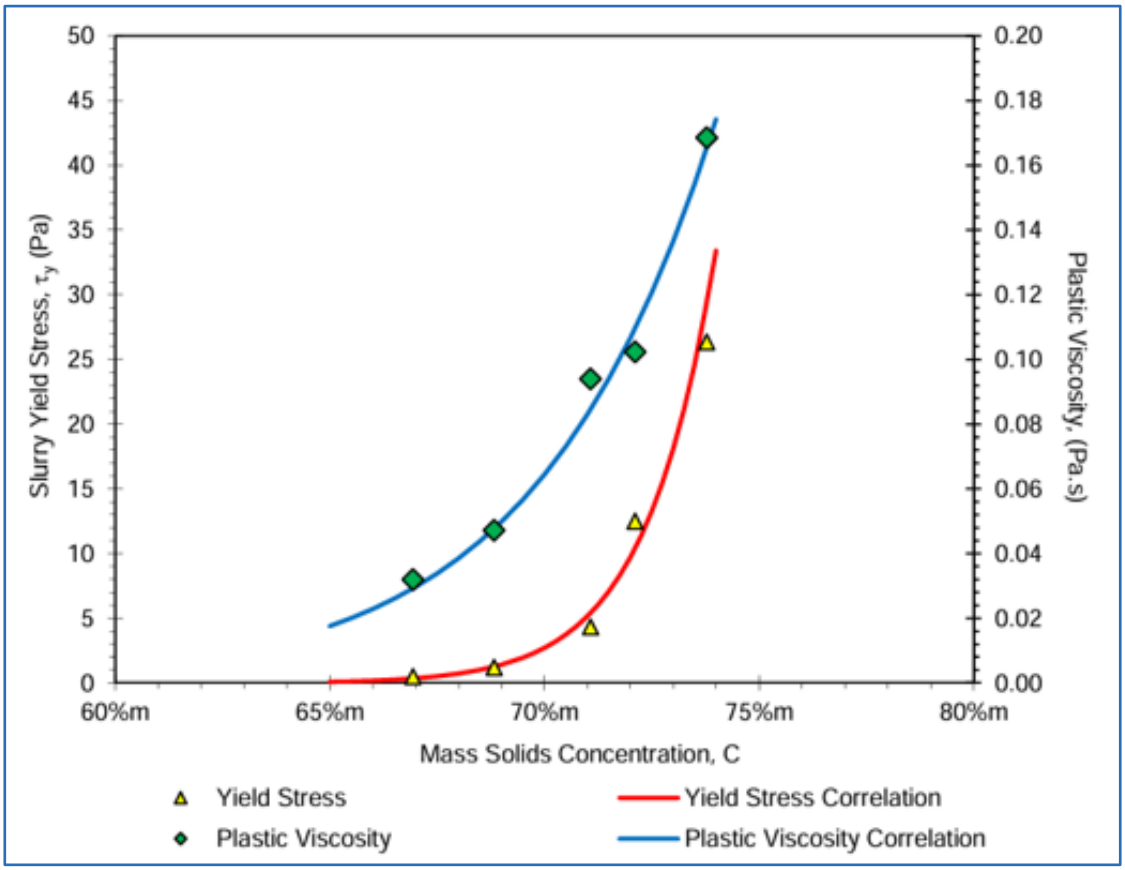
Table 13.4 Summary of Tailings Material Characterisation Test Results

Property Tested	Unit	Value
Solids Density	kg/m^3	3173
Particle Size (d_{80}/d_{50}/ d_{20})	mm	~80/40/20
Average Slurry pH at 25°C	-	7.1 (process water) ~8.4 (uncemented tests)
Average Slurry Conductivity	mS/cm	0.24 (process water) 0.26 (uncemented tests)
Average Temperature	°C	~24
Freely Settled Packing Concentration (C_{bfree})	%volume / %mass	42.1 69.8
Maximum Bed Packing Concentration (C_{bmax})	%volume %mass	50.1 76.1

Dynamic bench top thickening tests were completed on uncemented tailings slurry samples using the operating parameters previously derived from the settling testwork conducted during the original 2017 DFS. The results were consistent with the previous results achieving an underflow solids concentration of 65–68% w/w at a thickening flux of 0.47 t/m²h. The underflow solids concentration increased to 71% after 2 hours of consolidation time and remained constant at 71% after 24 hrs with a measured vane yield stress of 40 Pa.

Rheogram measurements of the uncemented, freshly thickened tailings were conducted for an underflow solids concentrations ranging from 67–74% w/w. The data was used to characterise the slurry rheology by applying the Bingham plastic model. The resulting yield stress and plastic viscosity as a function of mass solids concentration is presented in Figure 13.4.

Figure 13.4 Tailings Yield Stress and Plastic Viscosity as a Function of Solids Concentration



Paterson and Cooke, 2024

Top and bottom leaf vacuum filtration tests were undertaken to represent the cake form mechanism of a horizontal belt and disc filter respectively. The vacuum filtration tests were conducted at a feed solids concentration ranging from 60–75% w/w. Additionally, pressure filtration tests were conducted using a double-sided filter chamber unit in the horizontal position for a feed solids concentration ranging from 55–75% w/w. The results can be summarised as follows:

- Horizontal belt filtration tests at a feed solids concentration of 60–75% w/w produced filter cakes with a moisture content ranging from approximately 16–22% w/w. A filter flux of approximately 230–270 kg/m²h and moisture content of ~20% w/w was achieved for the lower feed solids concentration of 60% w/w. A higher flux of approximately 890–970 kg/m²h and moisture content of ~18% w/w was achieved at a higher feed solids concentration of 70% w/w.
- Vacuum disc filtration tests at a feed solids concentration of 70% w/w produced filter cakes with a moisture content of ~21% w/w at a filtration flux of approximately 400 kg/m²h. At a higher feed solids concentration of 75% w/w the flux increased to >800 kg/m²h and the cake moisture content was reduced to ~18% w/w.
- Vacuum disc filtration tests at a lower feed solids concentration of 60% w/w were not successful achieving high cake moistures of 25–33% w/w and a low filter flux of < 60 kg/m²h, the test results were not consistent with historical testwork, more recent comparative vendor testwork (refer Section 13.3.2.2) or the horizontal vacuum belt filtration tests at P&C. This may possibly be as a result of inefficient agitation and a bias towards finer material resulting in filter blockages.
- Pressure filtration tests produced filter cakes with a moisture content ranging from approximately 9–14% w/w. An average filter flux of 185 kg/m²h was achieved at a feed solids concentration of 55% w/w, increasing to approximately 270 kg/m²h at a feed solids concentration of 60% w/w.

The flow moisture point (FMP) and transportable moisture limit (TML) tests were also conducted with the estimated values reported to be 14.8% w/w and 13.3% w/w respectively.

Additionally, P&C conducted uncemented tailings bulk materials handling tests at a solids concentration ranging from 85–91% w/w. The uncompacted bulk density ranged from 1051–1669 kg/m³ and minimum and maximum angles of drawdown, surcharge and repose were measured as 70–83°, 17–35° and 9–22°, respectively.

13.3.2.2 Vendor Tailings Vacuum Disc Filtration Testwork

FLSmidth conducted additional vacuum disc filtration testwork to confirm the expected performance of the vacuum disc filter for the Phase 1, 0.77 Mtpa concentrator plant which is currently in the implementation phase.

The testwork, at a feed solids concentration of 55–60% w/w, produced a filter cake with a moisture content of approximately 16–17% w/w at a nominal filter flux of 650–800 kg/m²h. Comparative historical vacuum filtration testwork conducted by Goder for the 2017 DFS recommended a cake loading rate of 609 kg/m²/hr and a target moisture content of 17% w/w and a filter feed solids concentration of 69% w/w as the design basis for disc filter sizing.

13.3.2.3 Vendor Tailings Pressure Filtration Testwork

Pressure filtration testwork at the Metso Dewatering Technology Center (DTC) in Finland aimed to determine the suitability of Metso Larox® FFP filtration technology for dewatering tailings slurry at a feed solids concentration of 60% w/w.

The testwork confirmed the Metso Larox® FFP technology as suitable for the required duty, with a recommended cake loading rate of approximately 329 kg/m²h and a target moisture content of 11.9% w/w as the design basis for pressure filter sizing.

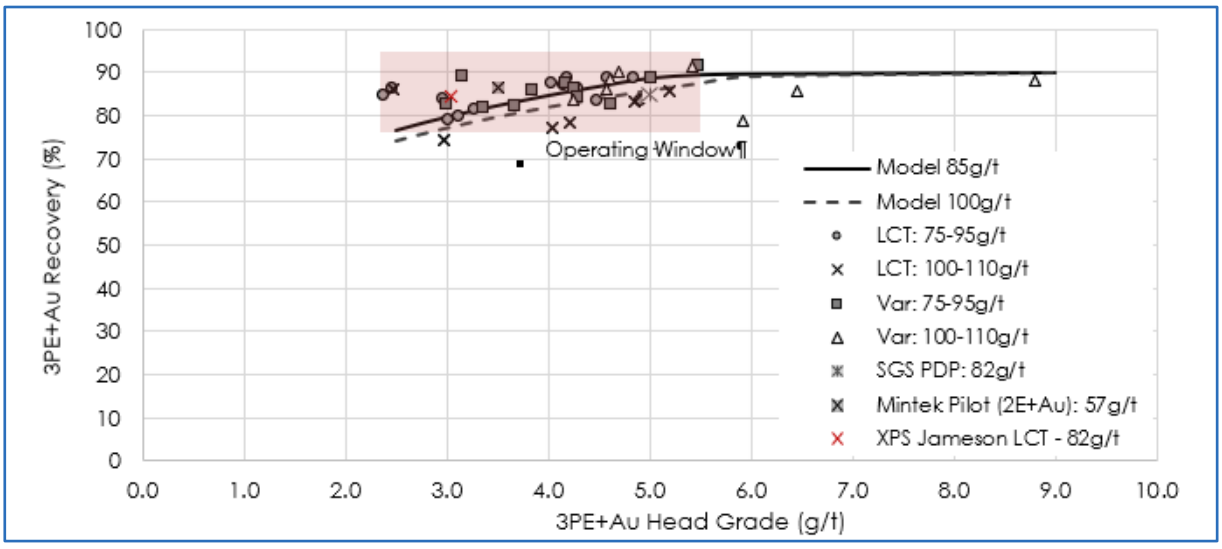
Cake discharge was reported to be good; the cake was noted to crack easily, and filter cloth washing was adequate with no evidence of cloth clogging. Filtrate was however visually hazy containing >400 mg/L solids.

13.4 Plant Recovery Estimate

The metallurgical performance projection for the concentrator has been based on the recovery algorithms previously developed for the 2017 DFS (refer Table 13.2) in combination with the expected concentrator feed grades from the IDP25 Phase 2 mine schedule at a design throughput of 4.07 Mtpa.

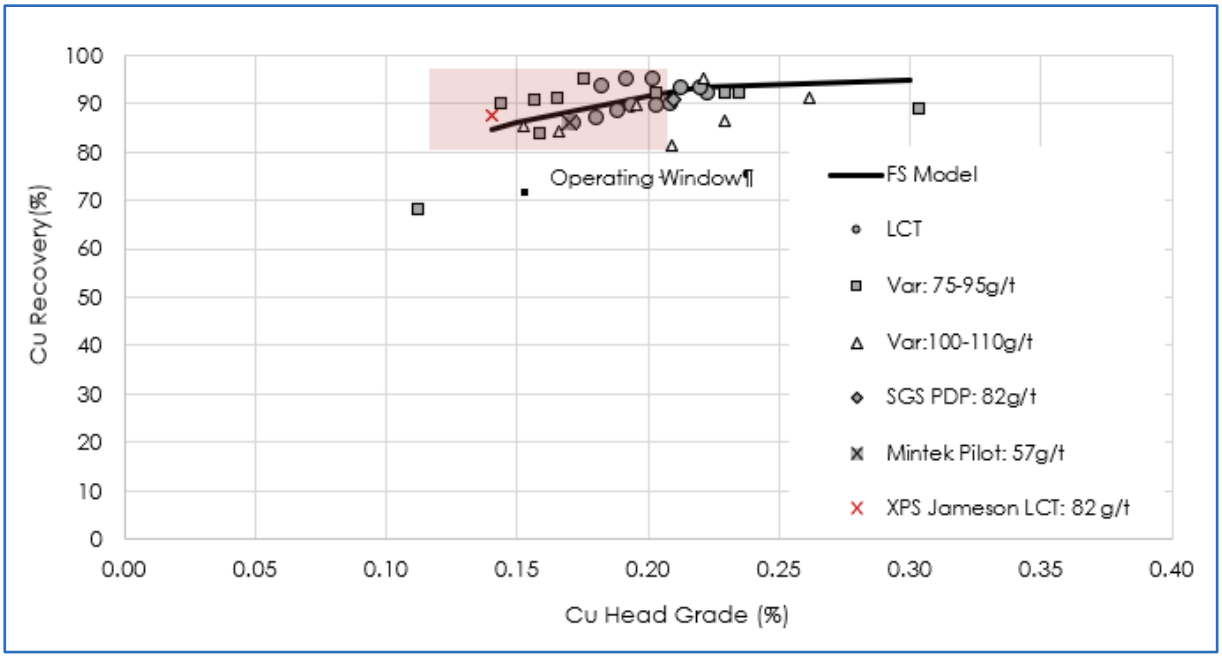
By using the recovery algorithms for each individual element, it is possible to calculate the expected overall 3PE+Au recovery based on the 3PE+Au target concentrate grade. The resulting 3PE+Au, Cu and Ni recovery models as a function of feed grade are presented in Figure 13.5, Figure 13.6 and Figure 13.7 respectively.

Figure 13.5 3PE+Au Recovery as a Function of Feed Grade



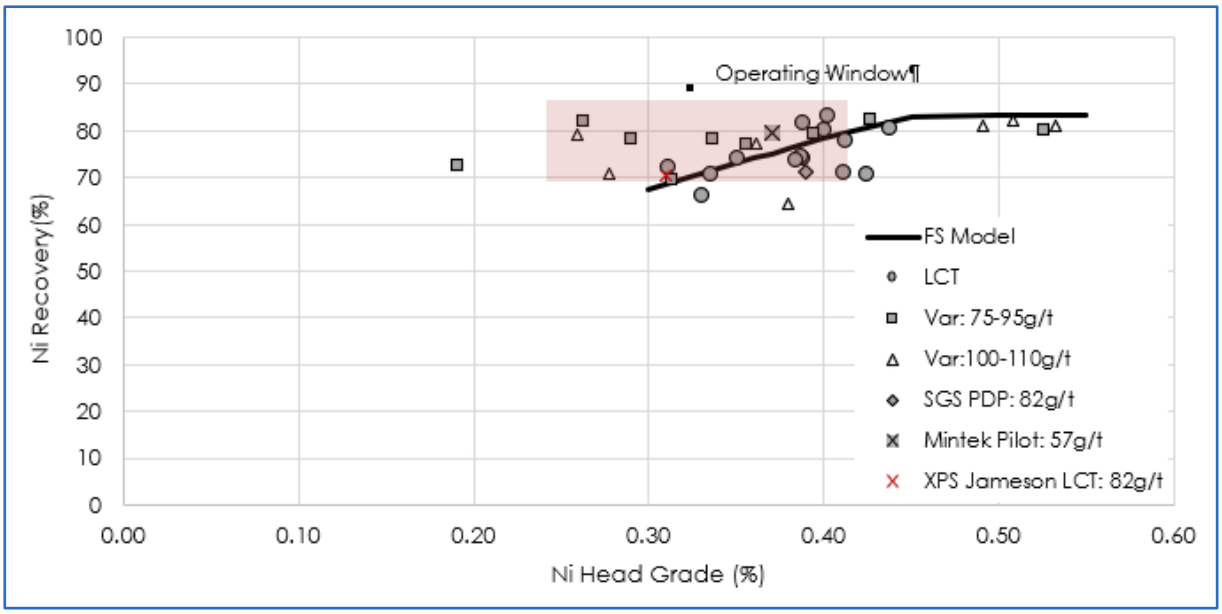
DRA, 2024

Figure 13.6 Copper Recovery as a Function of Feed Grade



DRA, 2024

Figure 13.7 Nickel Recovery as a Function of Feed Grade



DRA, 2024

The 3PE+Au, Cu and Ni recovery models have been plotted relative to the locked cycle test results and expected closed circuit recoveries derived from the open circuit variability testwork. The recovery estimates derived from open circuit variability results are not definitive

but do provide an indication of the expected variability. Variances when compared to test results are exaggerated due to that variances exist in the final concentrate grade achieved during testwork and modelled.

The results for open circuit and locked cycle flotation testwork at SGS, Mintek and XPS, post the 2017 DFS over the period 2020–2022, were found to fall within the ranges evidenced in the 2017 DFS variability testwork. This shows consistency between the various test campaigns and performance modelling using the recovery algorithms previously developed for the 2017 DFS.

13.4.1 Life-of-Mine Recovery Estimates

The recovery estimate for PGE's over life-of-mine are presented in Figure 13.8. The concentrate PGE, Ni and Cu production profile is presented in Figure 13.9.

The recoveries have been derived for a minimum target grade of 85 g/t and are based on the expected concentrator feed grades from the Platreef 2025 FS mine schedule. The modelled recoveries are steady state recoveries and do not consider any transient operations but do include a 2% recovery discount in the first six months of operation to cater for start-up and commissioning.

Figure 13.8 Modelled Steady State Life-of-Mine 3PE+Au Recovery Profile

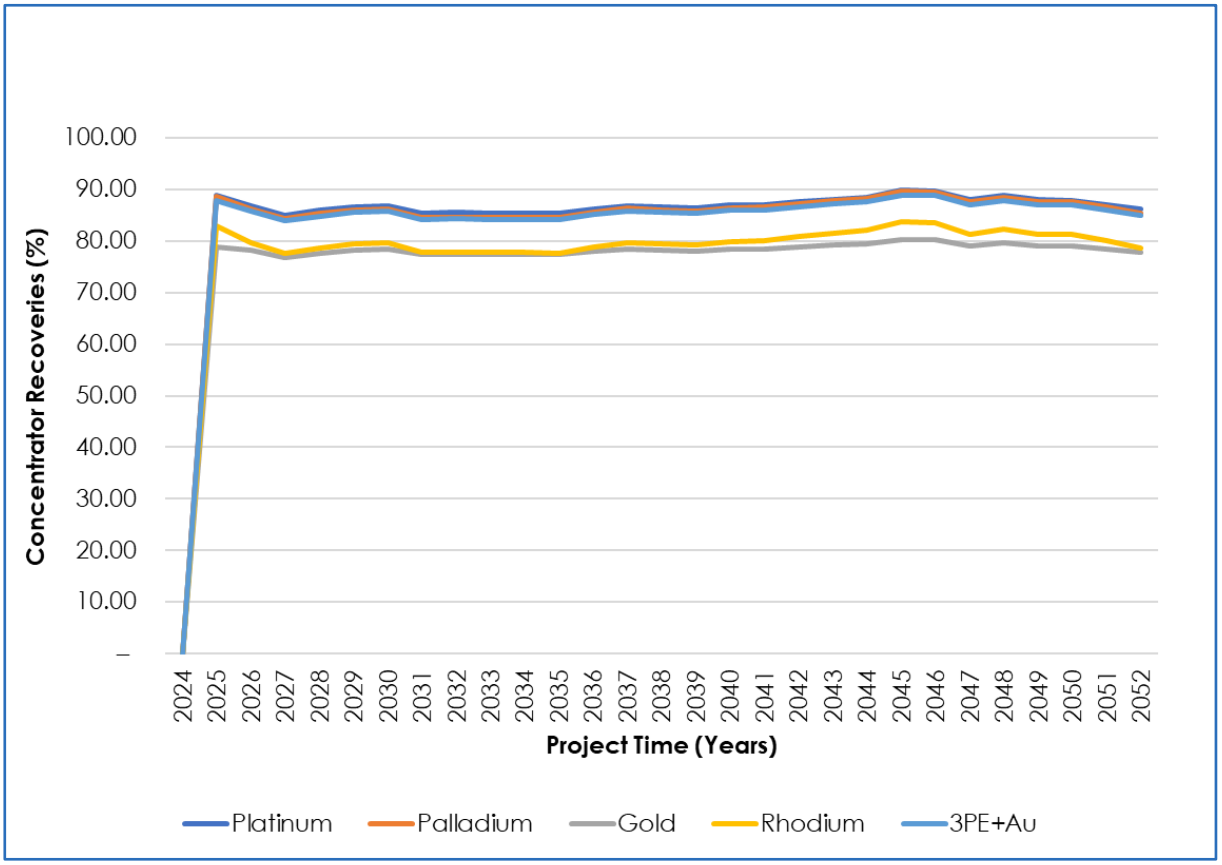
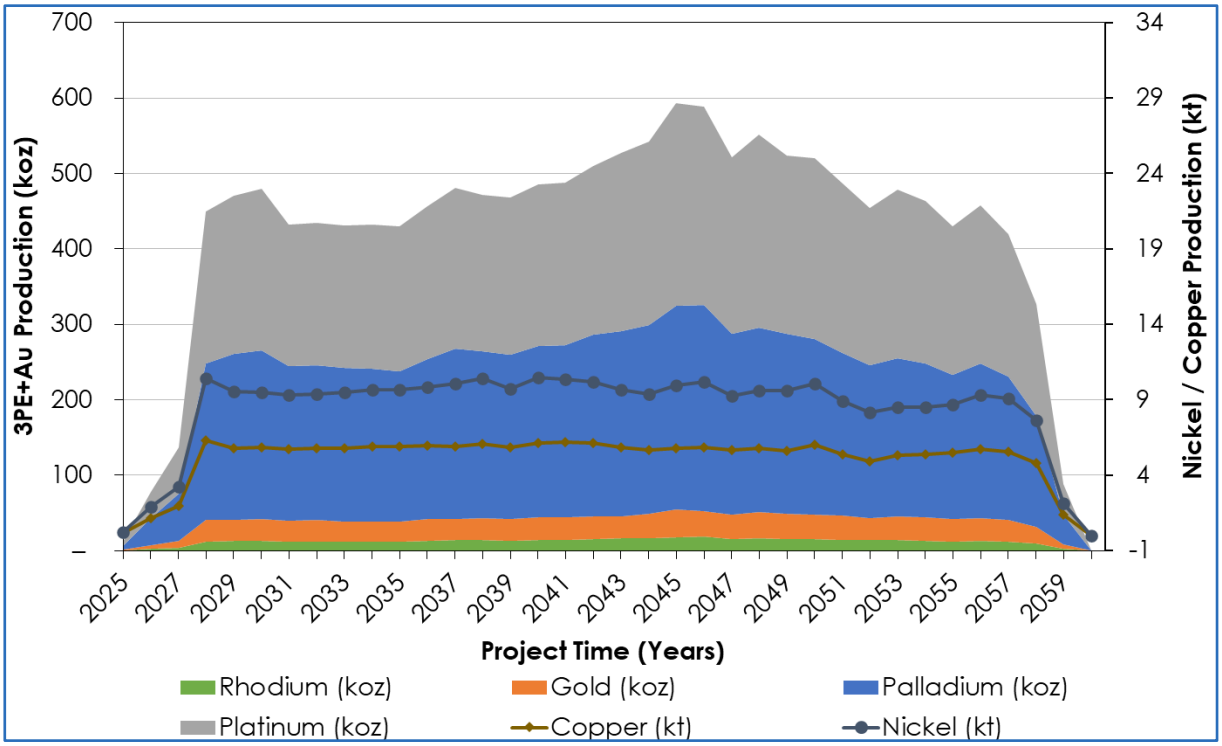


Figure 13.9 Modelled Life-of-Mine Concentrate, Cu and Ni PGE Production Profile



OreWin, 2025

13.4.2 Comments on the Recovery Estimate

The recovery algorithms previously developed for the 2017 DFS, based on locked cycle test results are considered adequate for metallurgical performance modelling.

The recovery algorithms are based on the 2017 DFS locked cycle flotation testwork on mineralised ore blends with the inclusion of footwall and hanging wall. Further to this, detailed open circuit flotation variability testing was conducted for the 2017 DFS to quantify the expected recovery ranges and to highlight the degree of variability that can be expected.

Additional open circuit and locked cycle flotation testwork was conducted at SGS and Mintek in 2020 and 2021. The results were found to fall within the ranges evidenced in the 2017 DFS variability testwork albeit predominantly on the lower end. Further open circuit and locked cycle flotation testwork by XPS in 2022 achieved results aligned to the 2017 DFS variability testwork and showed that a single Jameson cell can replicate or better the performance of two-stages of conventional cleaning. Jameson flotation technology has thus been recommended for the final HG cleaner, MG cleaner and LG re-cleaner duties.

The core composites that were used for the post 2017 DFS testwork campaigns over the period 2020–2022 reflect un-crushed ½ and ¼ drill core remainders from the 2017 FS drilling campaign. The age of these samples (4–6 years) may potentially have impacted on performance additionally sample variability, assay variances, procedural variances and non-optimised mass pull may also have contributed.

Bench-scale open circuit and locked cycle flotation testing at both Mintek, SGS and XPS, has consistently shown that as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). However, it was also shown that if the circuits target an excessively high mass pull, the concentrate grades can be depressed without any notable gain in recovery. The final concentrate recovery and grade will need to be optimised during full scale operations by targeting, controlling and optimising the final concentrate mass pull.

The Mini pilot plant (MPP) experienced several operational challenges and was not fully stabilized and optimised, these results are thus deemed to reflect a non-optimised, preliminary commissioning level performance.

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 influence performance.
- Operating conditions: Laboratory operation is undertaken under controlled, 'ideal' conditions. Operational disturbances on full-scale operations such as starting and stopping of the plant undoubtedly cause loss of recovery.
- Operational skills: The bench-scale laboratory tests are supervised by 'expert' operators. In the actual plant recovery losses may occur because of poor operational practices.

To address this, 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.

13.5 Future Testwork

No additional testwork is currently planned. The Phase 1 Project is currently being executed and includes the construction of an initial 0.77 Mtpa concentrator, which will be followed by a phased capacity expansion.

Further optimisation of the flow sheet during the Phase 1 commissioning, ramp up and optimisation phase should evaluate the potential for concentrate grade optimisation and improved 3PE+Au recovery by recycling the scavenger cleaner tailings to the rougher circuit. Additionally, the potential to conduct further trials of an SIBX reagent suite may be considered.

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 2025:

- TCU Model (UMT-TCU) – The TCU Mineral Resources includes material within and adjacent to grade shells (3PE+Au) in the TCU. 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. 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. 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 provided for confidence in mineralisation continuity allowing for declaration of Inferred Mineral Resources at relatively wide drill spacing. 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).

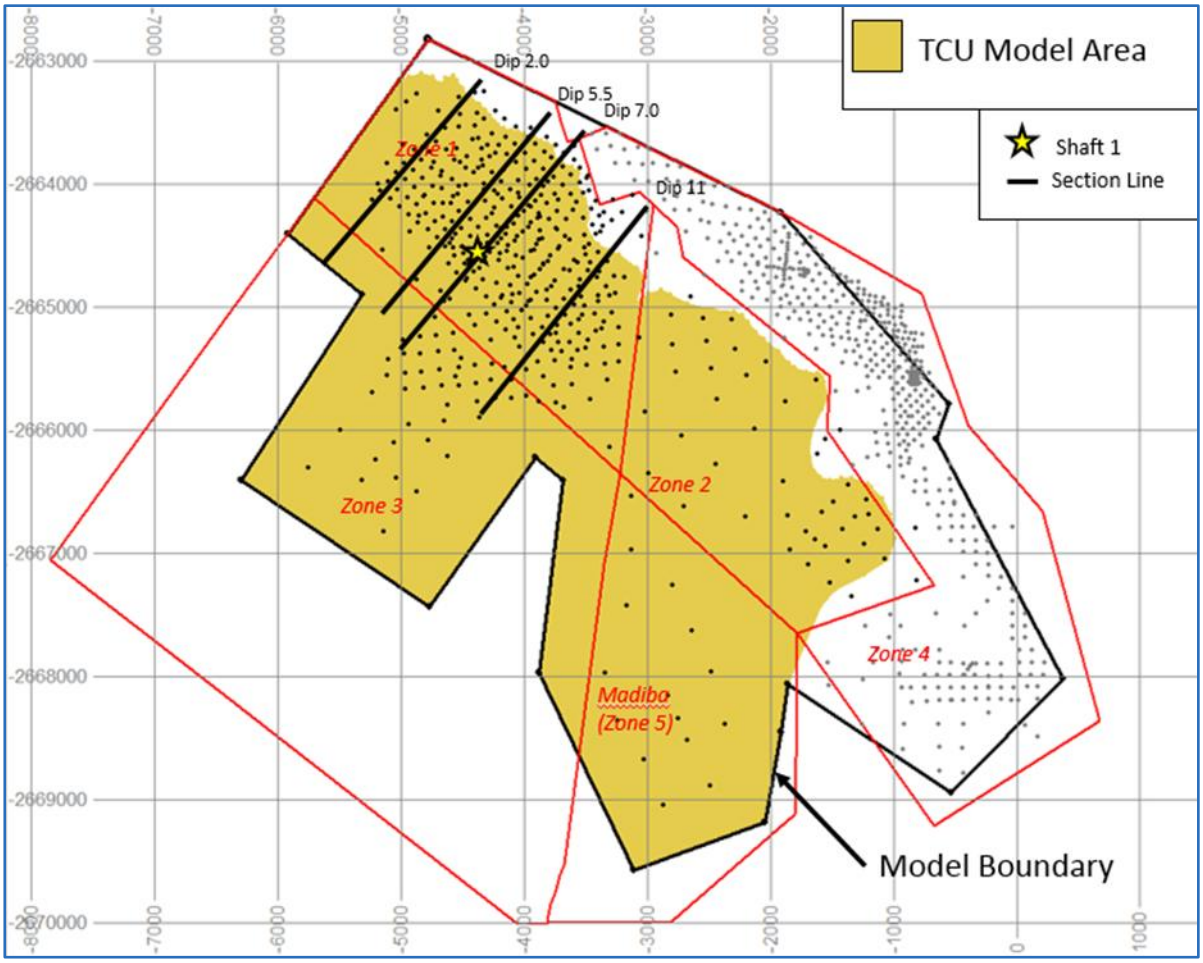
Additional drilling down-dip in Zone 3 and in the Madiba area permitted the expansion of the Inferred Mineral Resource in the UMT-TCU portion of the deposit. Additional down-dip/lateral potential could support estimation of additional Mineral Resources with further drilling. Revised geological interpretations 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).

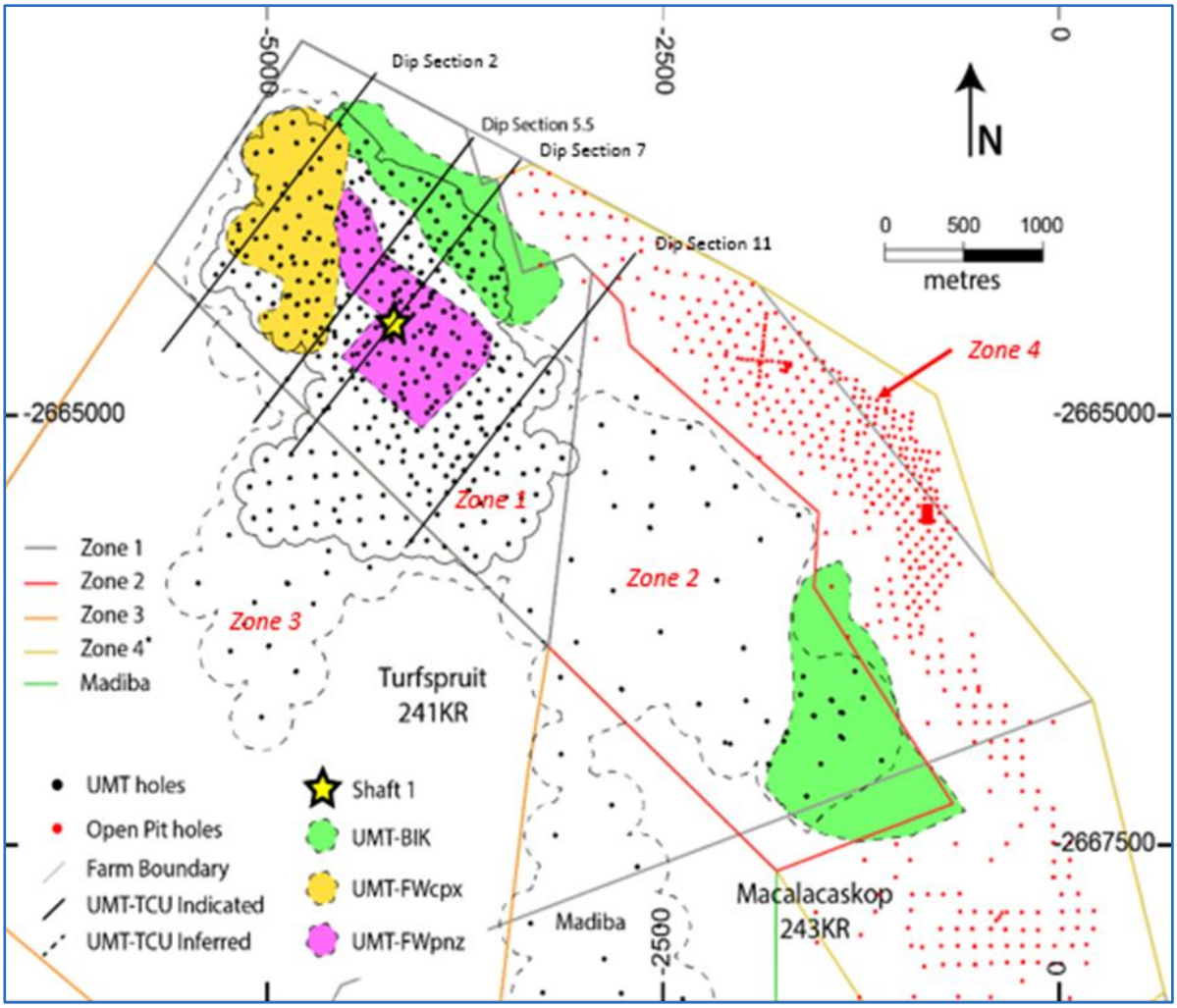
The UMT-FW model is located in Zone 1 and is situated stratigraphically below the UMT-TCU resource model (see Figure 14.2). Figure 14.3 shows the relative stratigraphic positions of the UMT-BIK, UMT-TCU and UMT-FW models.

Figure 14.1 Mineral Resource Area for the UMT-TCU



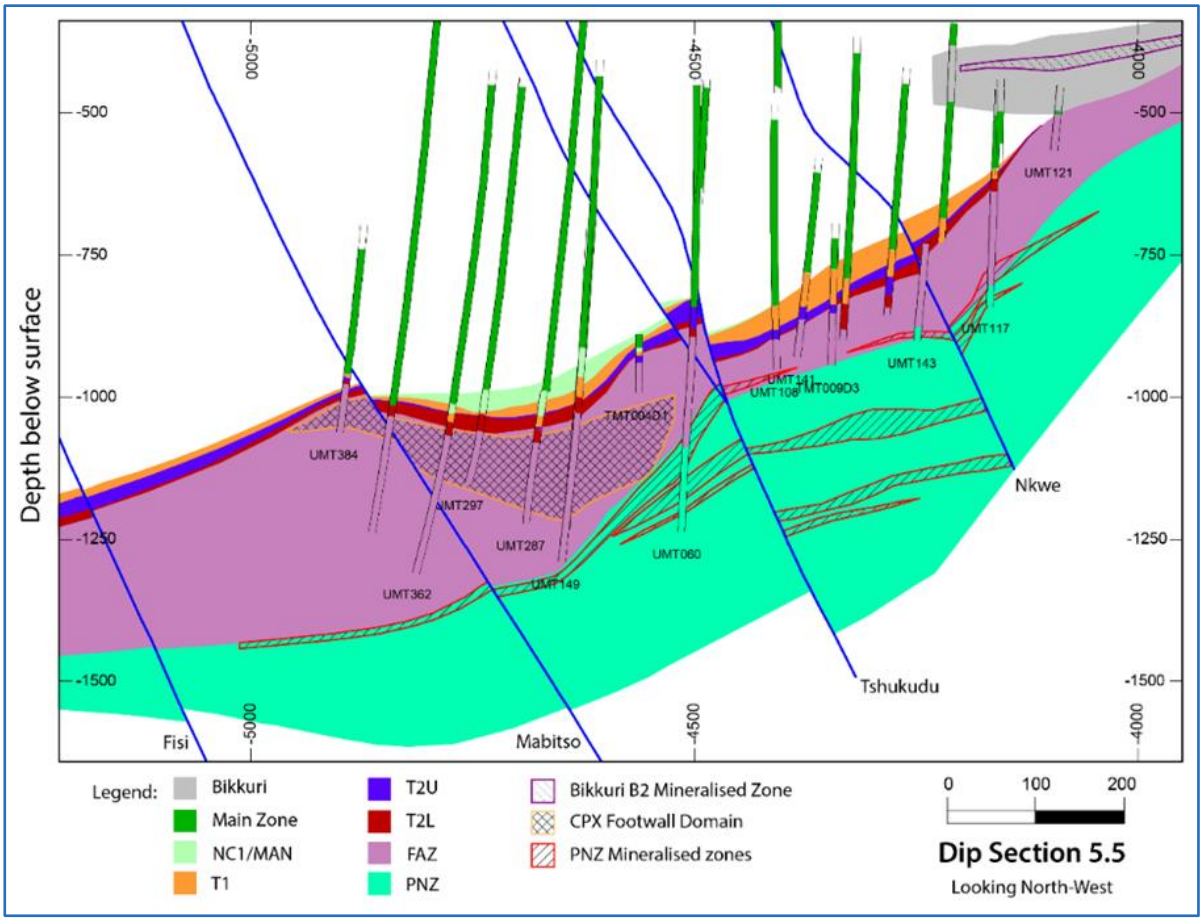
Wood, 2016

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.

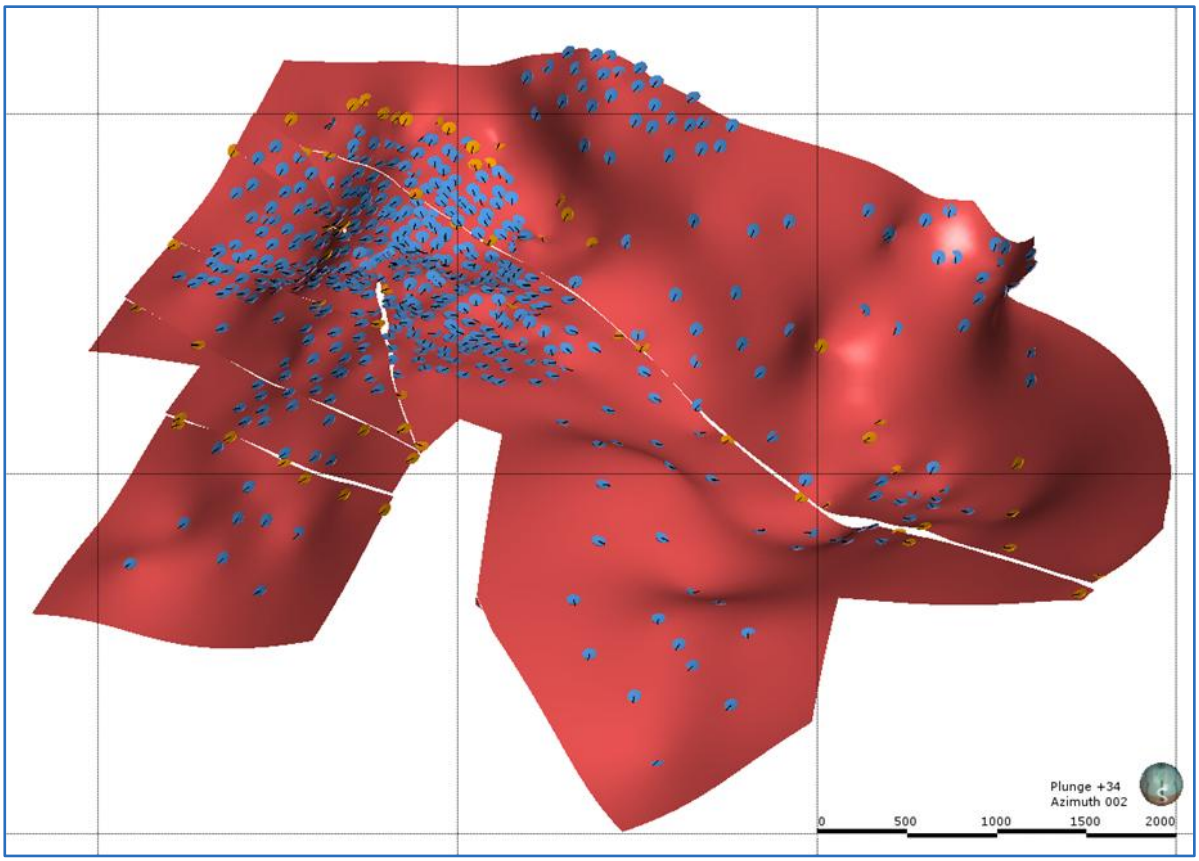
Relogging of drill core, infill drilling in Zones 1 and 2, expansion drilling in the Madiba area and a revised structural model since 2013 has resulted in modifications incorporated in the geology model.

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 14.3).

Mineralisation in the southern portions of Zone 2 that is stratigraphically above the TCU has been interpreted as like 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 FWpnz 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 7.2.8.

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 75 m below the base of the T2L. The UMT-TCU model does not extend above the 650 m elevation.

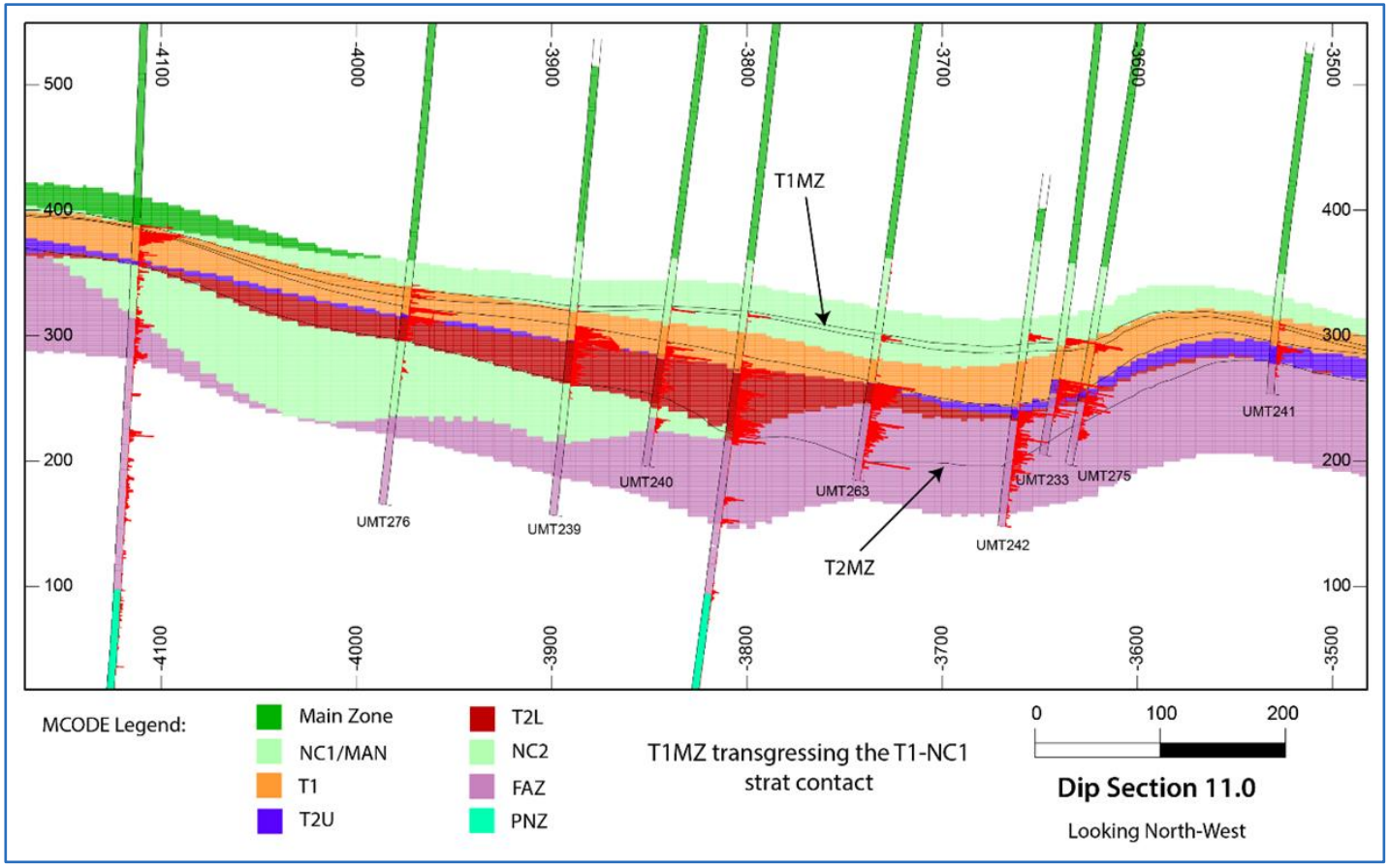
14.2.4 High Grade Shells – UMT-TCU

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 +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

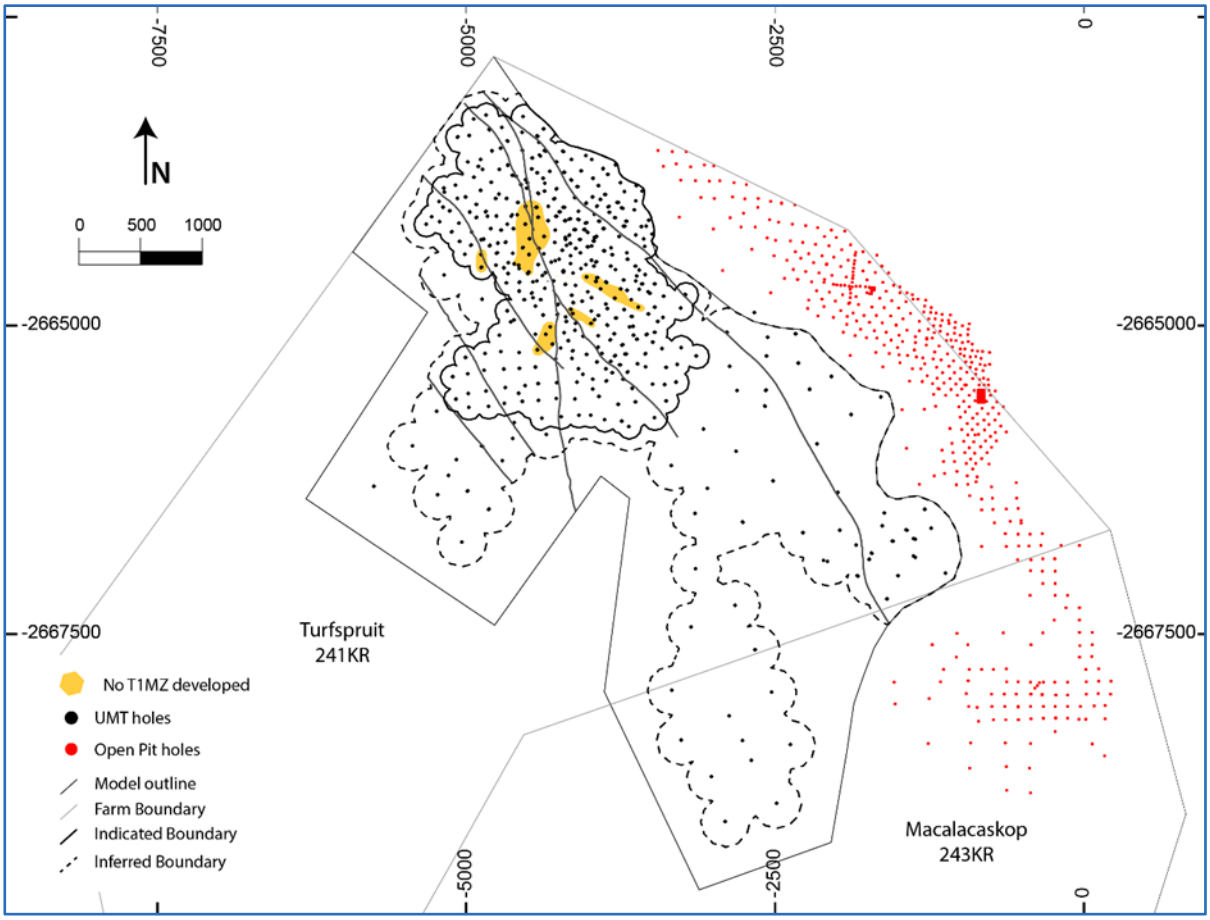
The T1MZ is interpreted to transgress the T1-NC1 stratigraphic boundary in the southern portions of Zone1 and into Zone 2 (see Figure 14.5). This transgression is localised and occurs in response to thickening of the T1 and NC1 units and the development of weak cyclicity within the T1. This relationship 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).

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

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.

Table 14.2 Summary of GCODES For TCU and Bikkuri (All Elements)

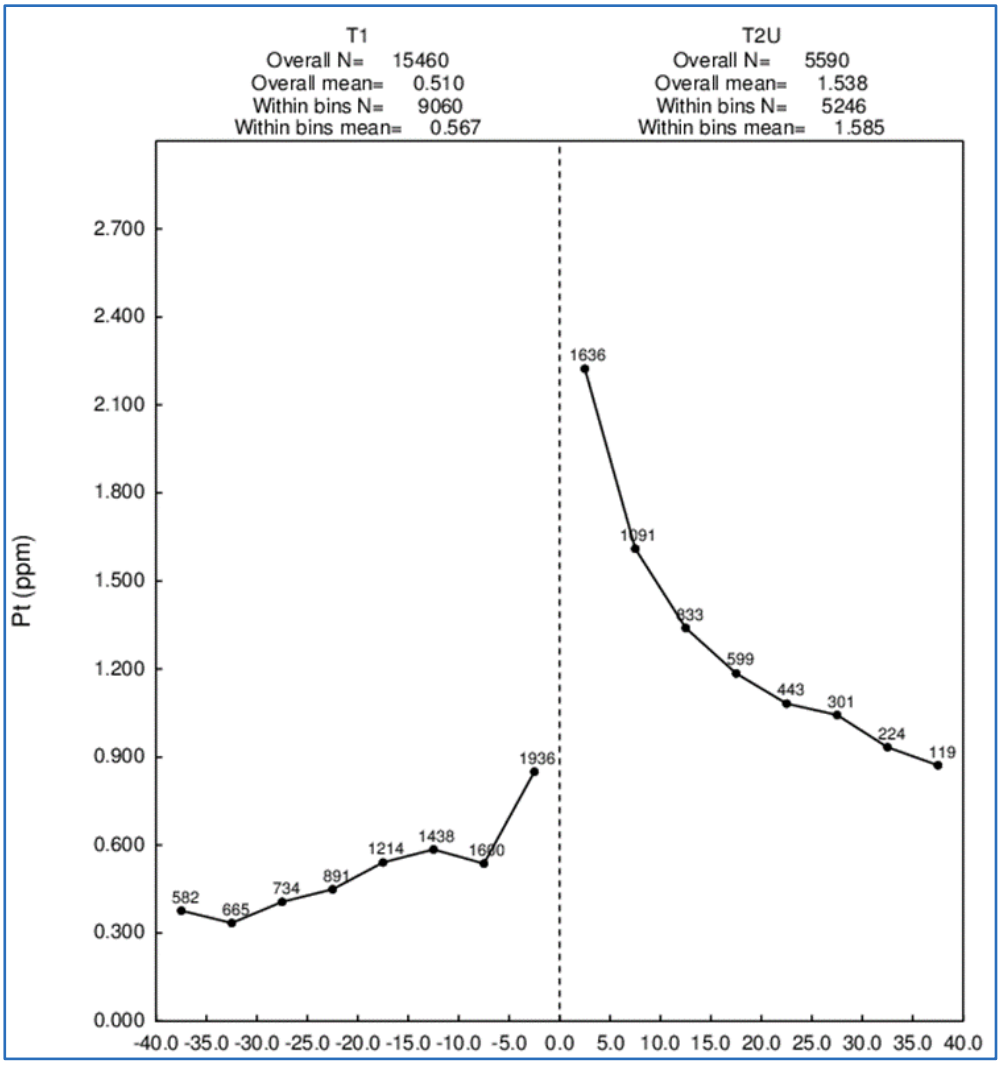
Model	Strat Unit Outside Grade Shells	Grade Shell	GCODE
TCU-BIK	NCBK, MANBK	(Outside Grade Shells)	0
		B1MZ 1g	301
		B1MZ 2g	302
		B1MZ 3g	303
	B1	(Outside Grade Shells)	1
		B2MZ 1g	401
		B2MZ 2g	402
		B2MZ 3g	403
	B2	(Outside Grade Shells)	2
	LZBK	(Outside Grade Shells)	0
UMT-TCU	MZ, NC1, MAN	(Outside 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



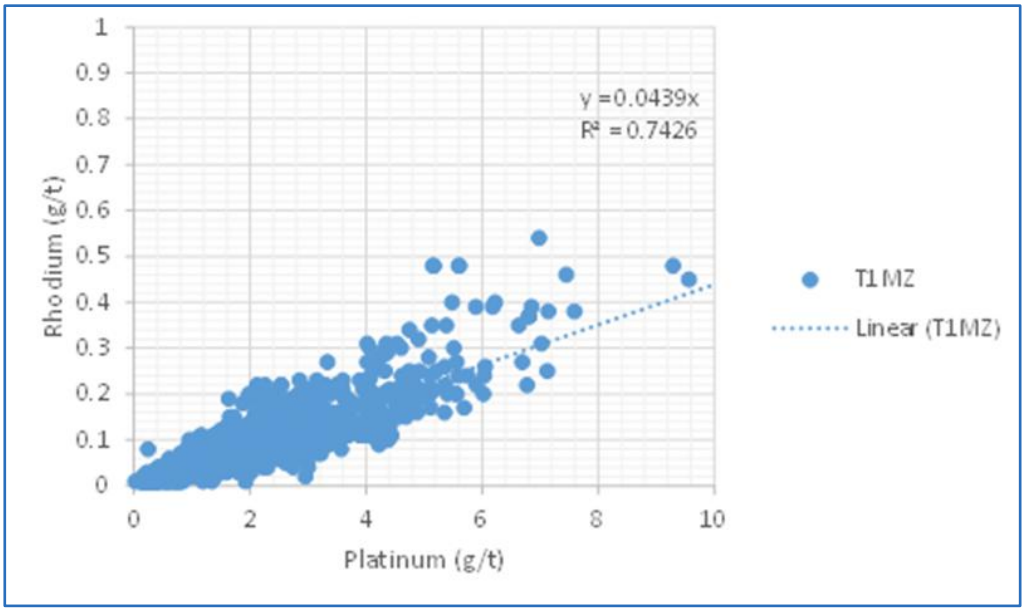
Wood, 2015

14.2.6.1 Rhodium Regressions

Rh analyses are available for most intercepts within the mineral zones. Rh to Pt regressions were constructed for samples missing Rh analysis (Parker, 2015).

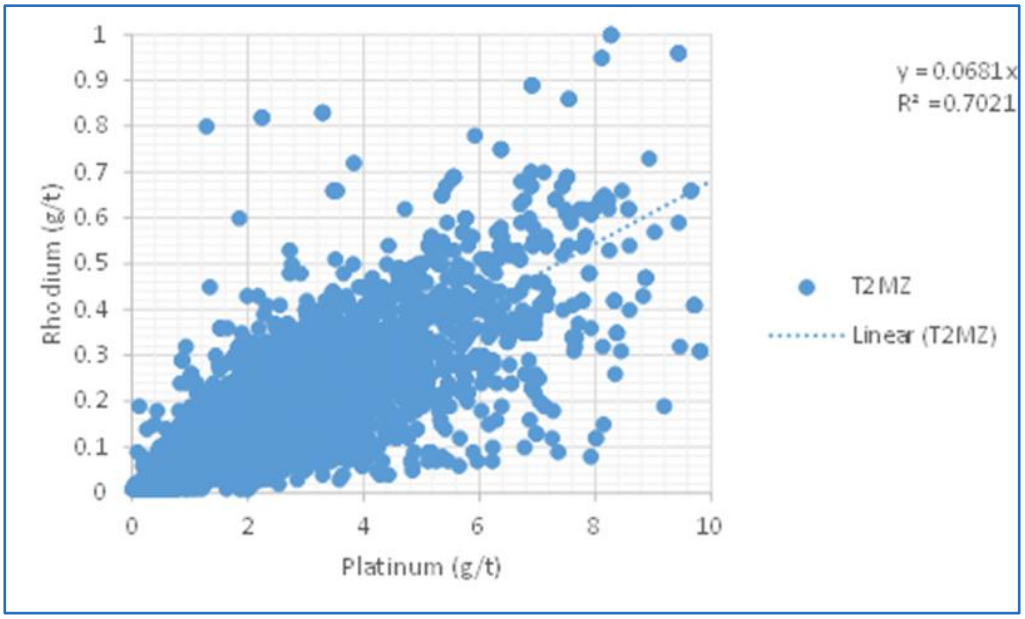
Figure 14.8 and Figure 14.9 show Rh as a function of Pt regression for the T1 and T2 respectively. Table 14.3 summarises the proportions of assays with Rh analysis within the grade shells and by stratigraphic unit. Table 14.4 summarises the lithology groups. The proportion of Rh assays exceeds 50% within the 2 g/t 3PE+Au shell and exceeds 60% in the in T2U and T2L. Table 14.5 summarises the number of missing Rh 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**	**Average Rh/ Average Pt Ratio**	**No. Data without Rh Values**	**Average Rh Value Regressed**	**Average Pt Value**	**Average Rh/ Average 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**	**Average Rh/ Average Pt Ratio**	**No. Data without Rh Values**	**Average Rh Value Regressed**	**Average Pt Value**	**Average Rh/ Average 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

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

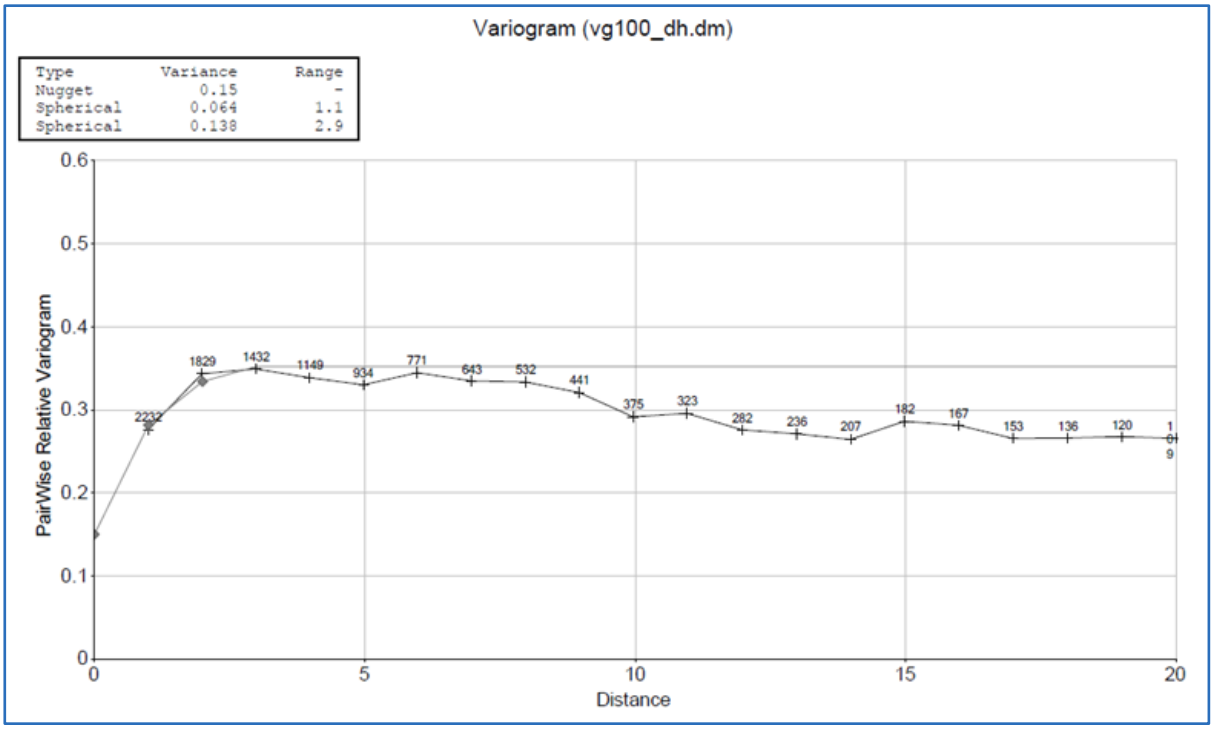
Table 14.5 Missing Rhodium Analysis by Grade Shell (All Intervals)

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

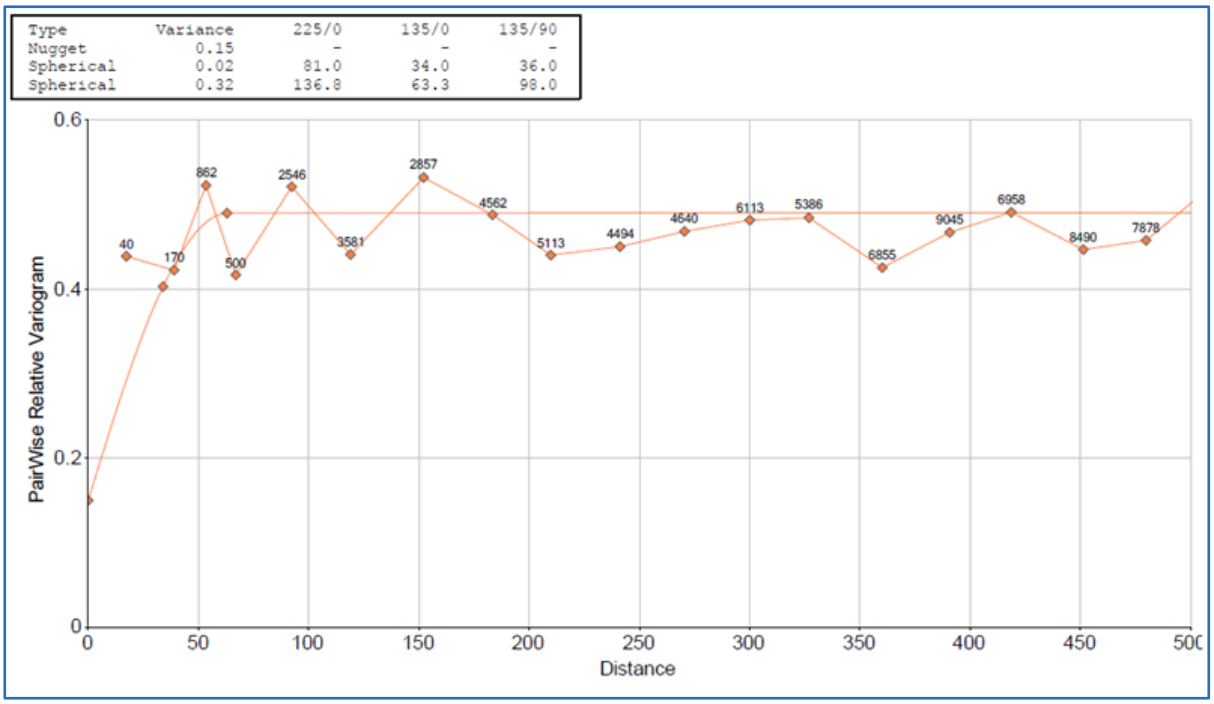
Pair wise relative variograms were completed by grade shell. Figure 14.10 shows a downhole pairwise relative variogram for Pt. Figure 14.11 shows a directional pairwise relative variogram for Pt (azimuth 135, dip 0).

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)

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	2.00
NC1/MAN	0.60	1.00	0.25	0.05	0.40	0.25	1.00
T1	2.00	1.20	0.60	0.06	0.50	0.35	2.50
T2	1.50	2.20	0.35	0.10	0.50	0.35	2.00
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	5.00
PNZ	0.25	0.50	0.06	0.15	1.00	0.65	10.00

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	7.50	10.00	-	-	-	-	5.00
T2MZ 1g/t	4.00	4.00	0.70	0.25	1.00	0.50	4.00
T2MZ 2g/t	5.00	10.00	-	-	-	-	-
T2MZ 3g/t	9.00	10.00	2.00	-	-	-	4.00
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	0.130	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.

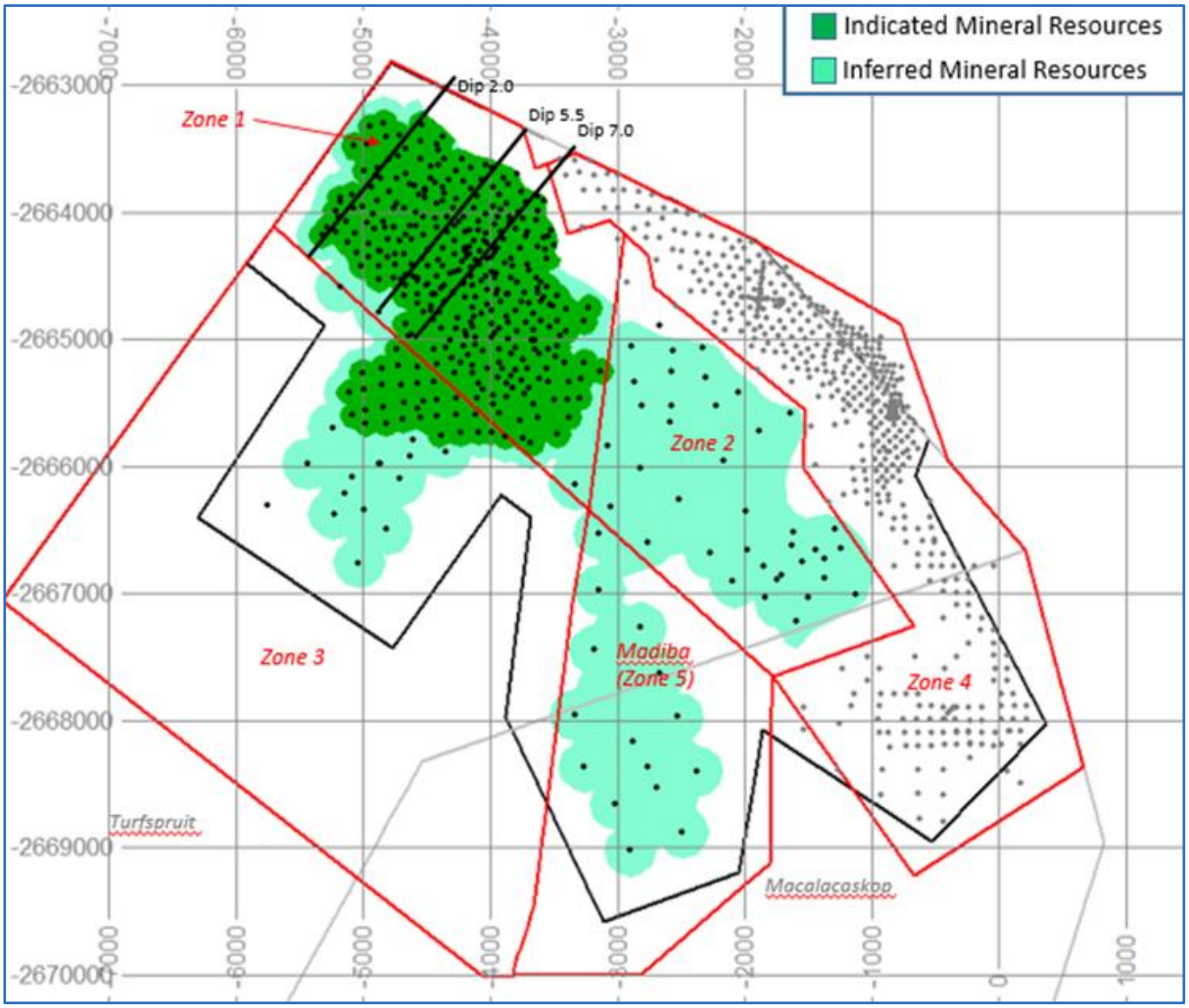
Early drilling in the UMT programme extended well below the T2MZ. Later infill drill programmes were focused on the TCU stratigraphy and were completed 20 to 50 m into the FW. This results in a wider drillhole spacing below the TCU.

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

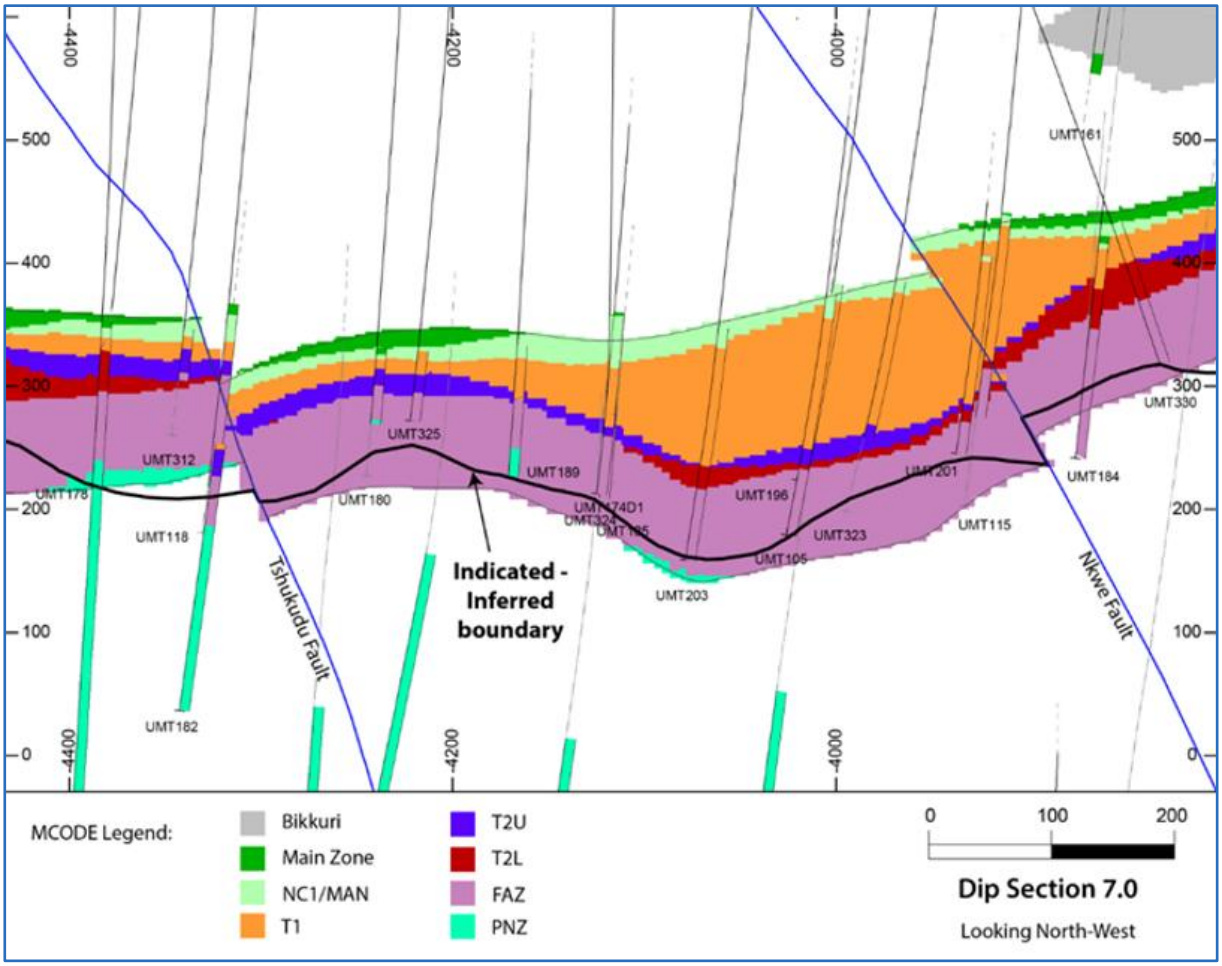
The Mineral Resource Classification for the TCU model is shown in Figure 14.12. No Measured Mineral Resources are declared. Indicated Mineral Resources are declared where closer spaced drilling has been completed (Predominantly Zone 1). Inferred Mineral Resources are declared where the drillhole spacing 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.

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.

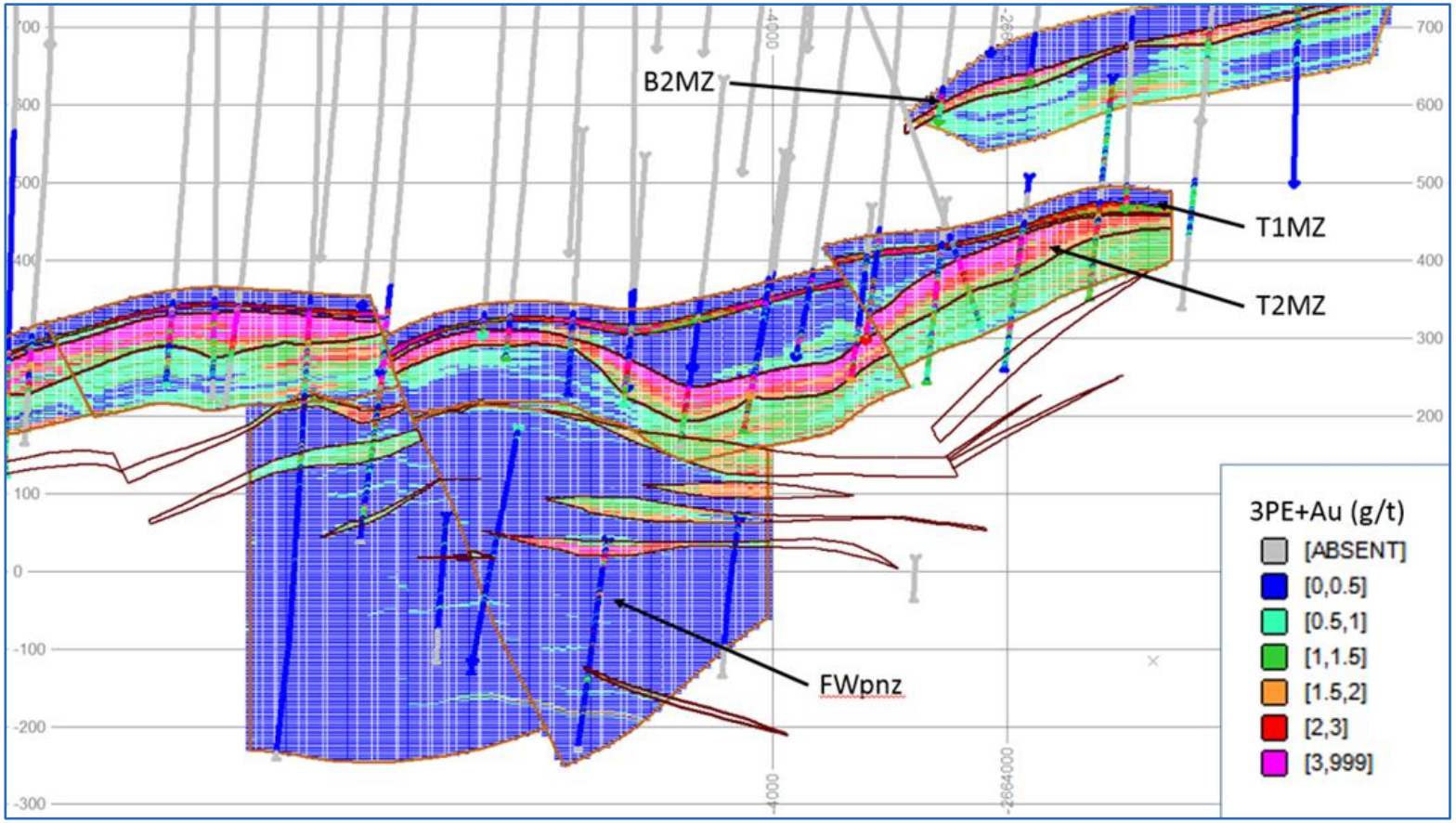
14.2.13 UMT-TCU Model Validation

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

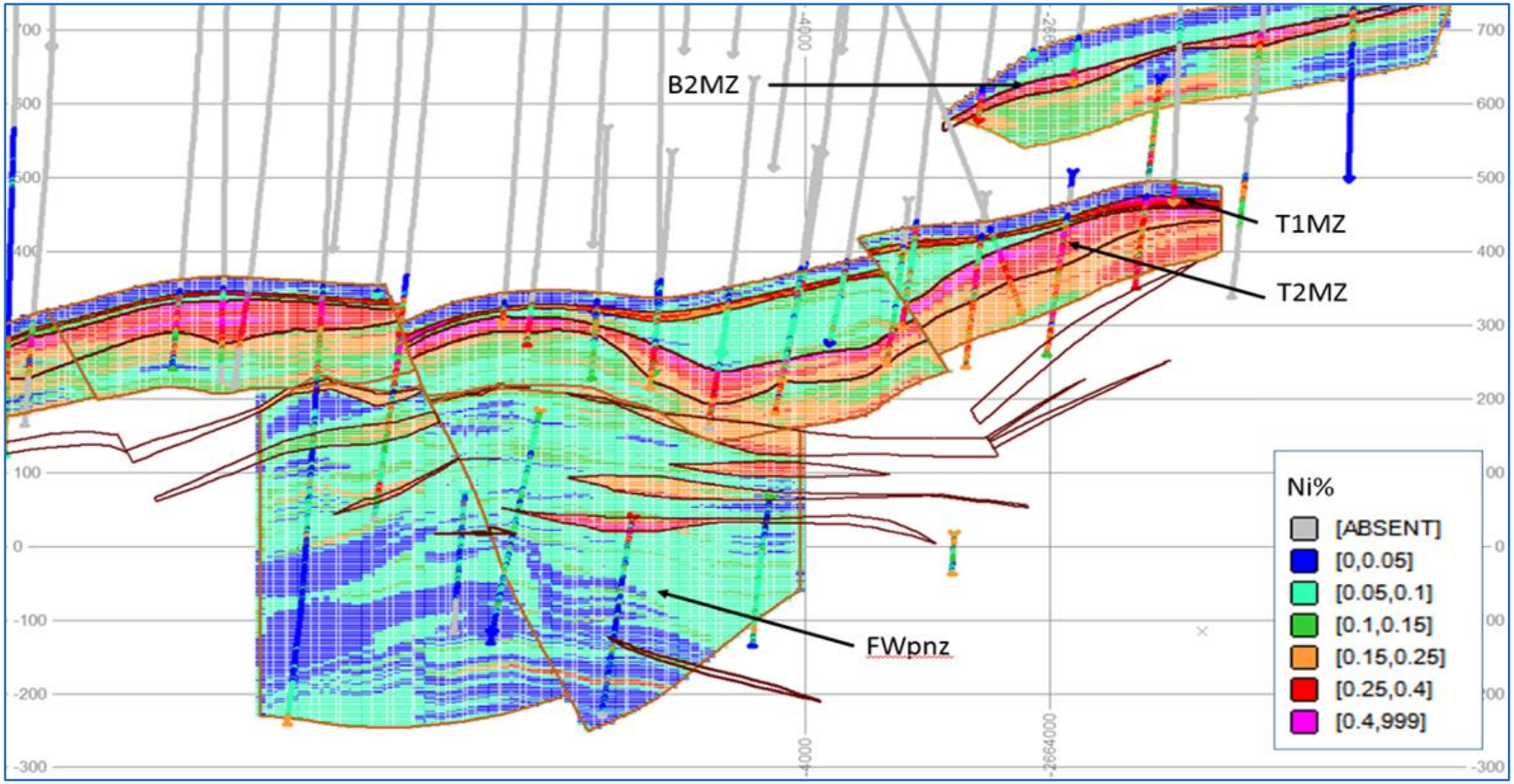
Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long-sections and level plans. In general, the block grades honoured the composite grades. Representative cross-sections for 3PE+Au are presented in Figure 14.14 (Dip Section 7.0) and Figure 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 14.14 Dip Section 7.0 Showing 3PE+Au (50 m Drillhole Projection) Looking North-West



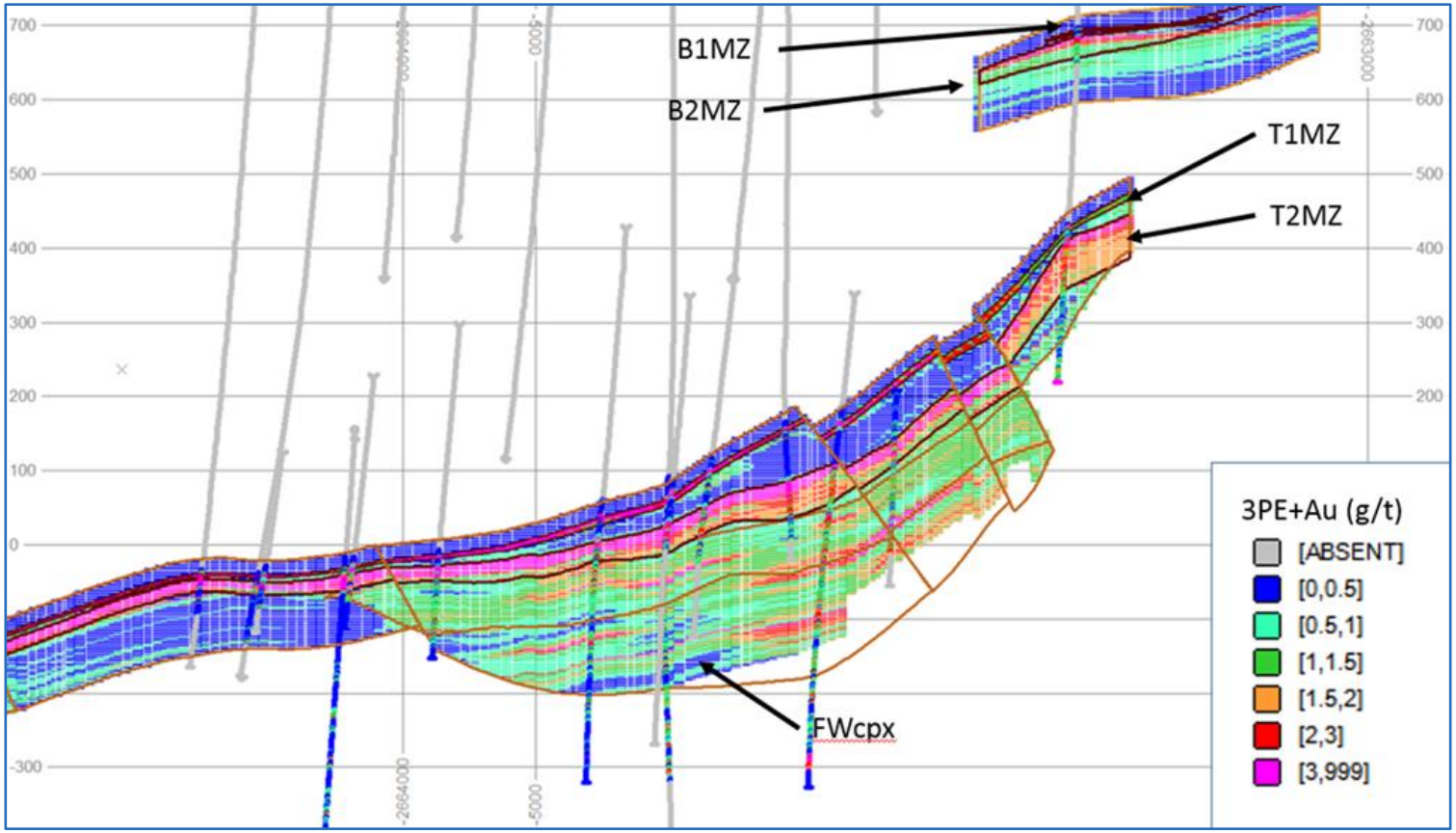
Wood, 2016; Location of Dip 7 is shown on Figure 14.2.

Figure 14.15 Dip Section 7.0 Showing Ni% (50 m Drillhole Projection) Looking North-West



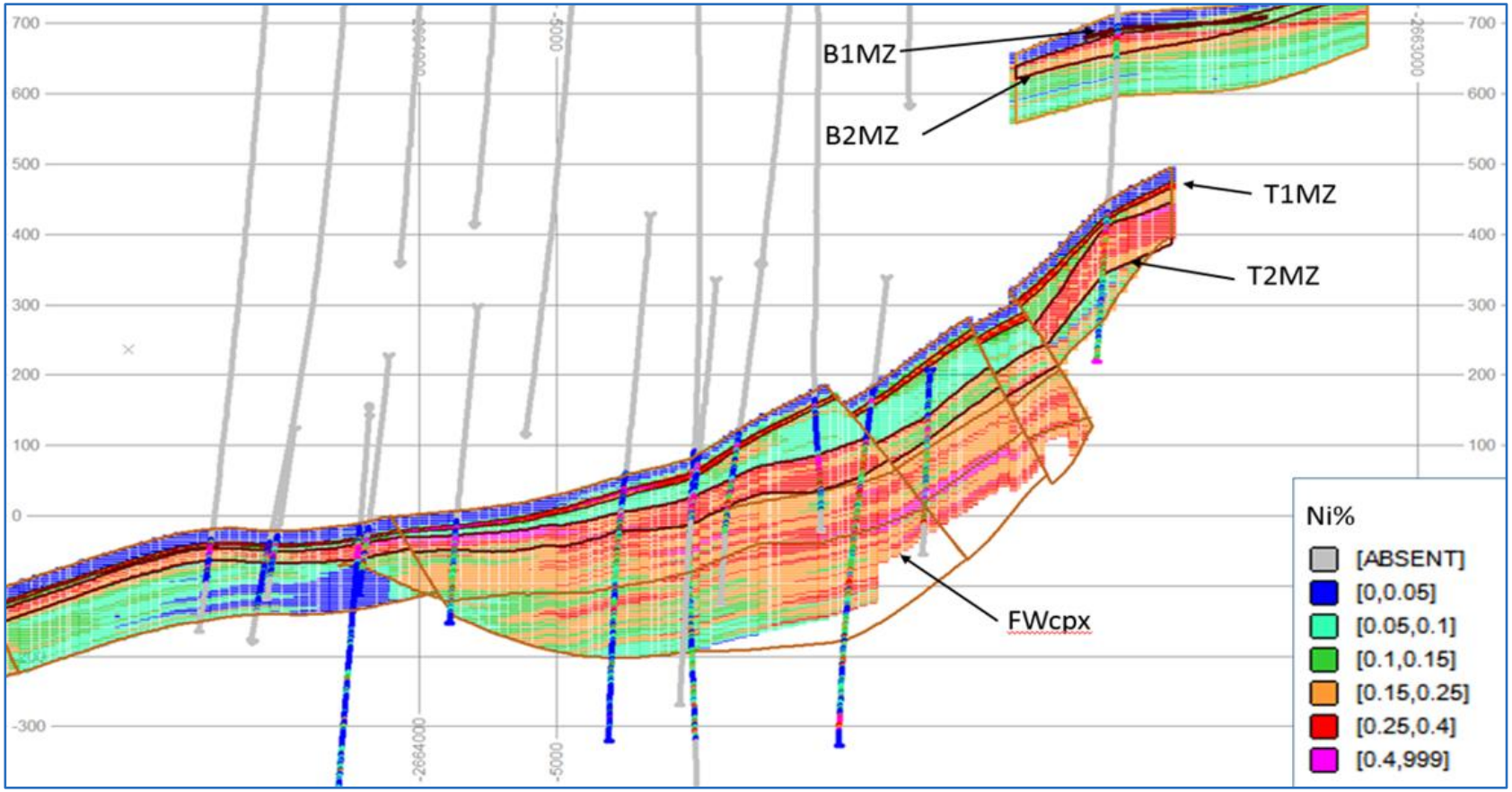
Wood, 2016; Location of Dip 7 is shown on Figure 14.2.

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.

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. There were no for T1MZ and T2MZ domains with a global bias outside guidelines of ±5% (relative) (see Table 14.12).

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.

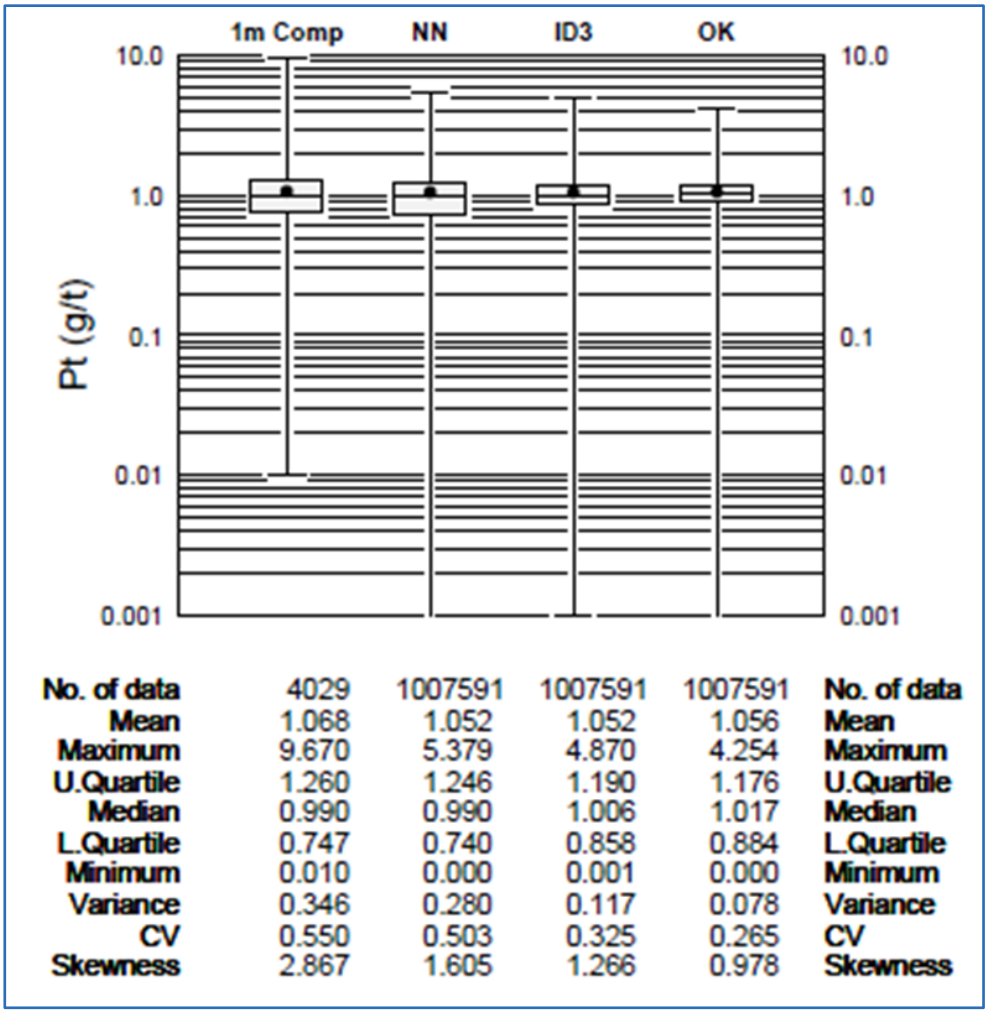
Overall, swath plots display reasonable comparisons between the ID3 estimates and their respective NN estimates; however, locally there are some differences, particularly in areas with limited drilling.

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

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%

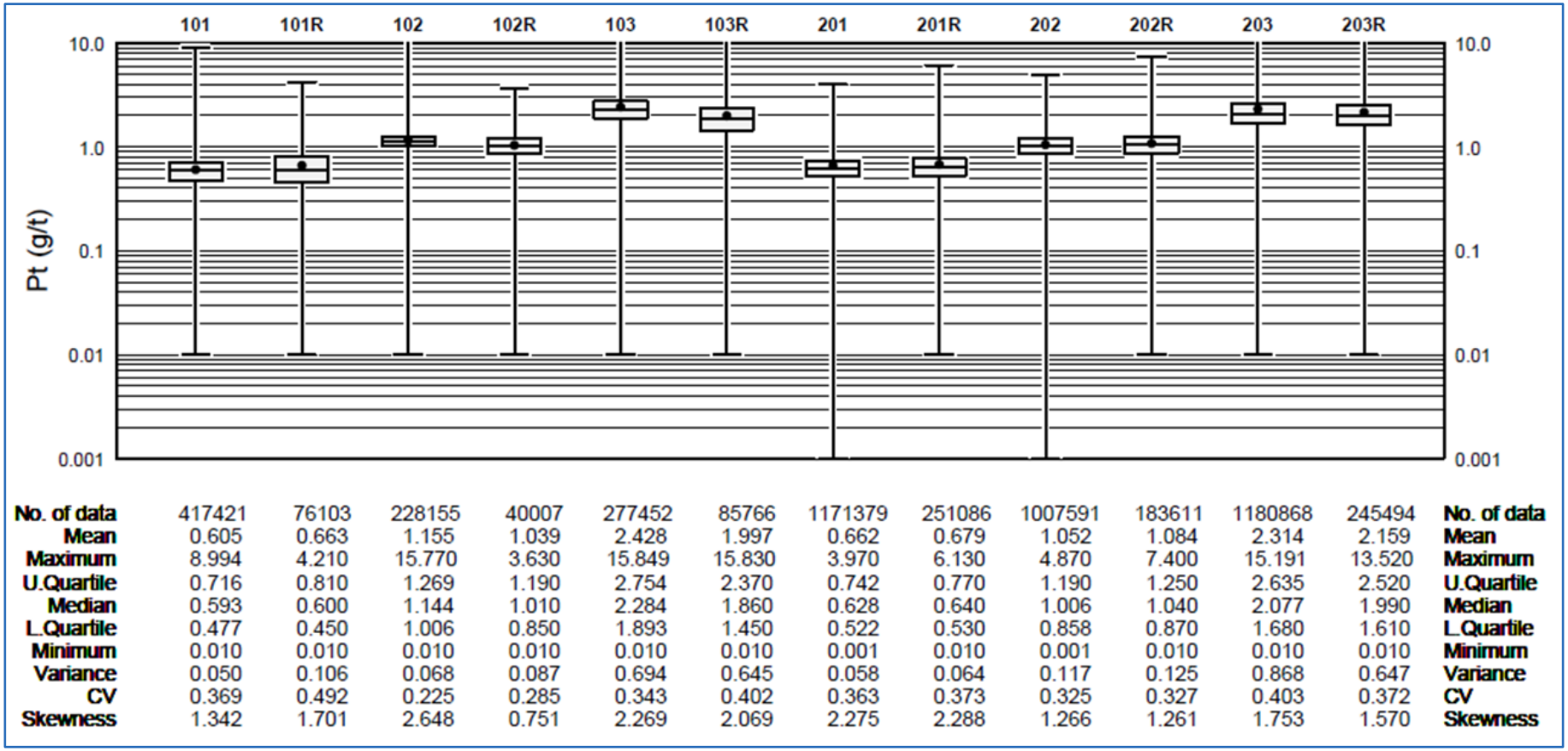
Figure 14.18 Box Plot of Pt for T2MZ 3PE+Au



	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

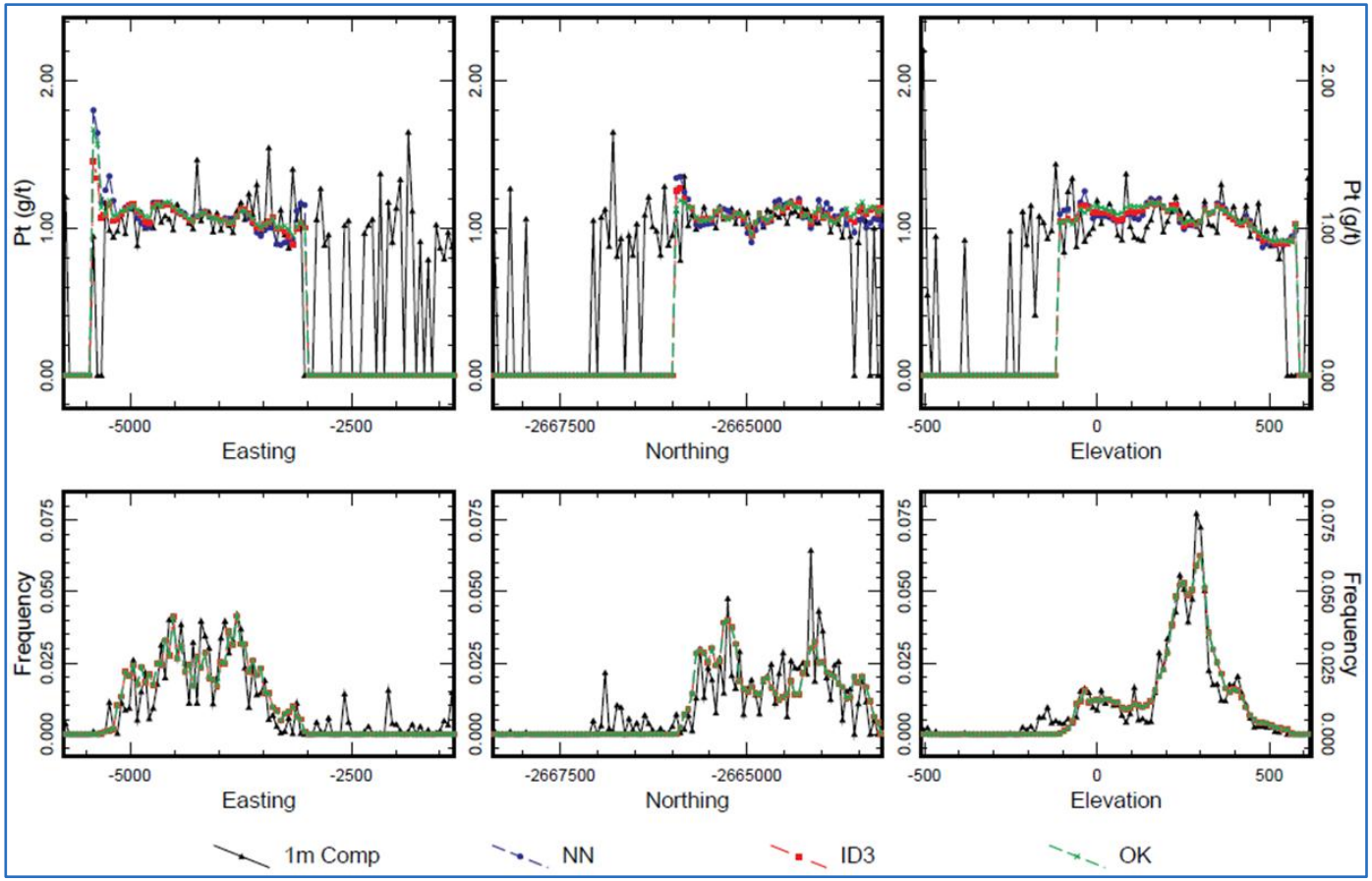
Figure 14.19 Box Plot of Pt Comparing Sub-Celled and Regularised Blocks for T1MZ and T2MZ Indicated Plots by 3PE+Au Grade Shell



	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" indicates 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

14.2.14 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 more accurate grade estimates.

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 12) 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

Rh analyses are only partially complete on the Bikkuri drillholes. Rh regressions for the Bikkuri drill data were used to address the missing Rh 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. Alternate 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%

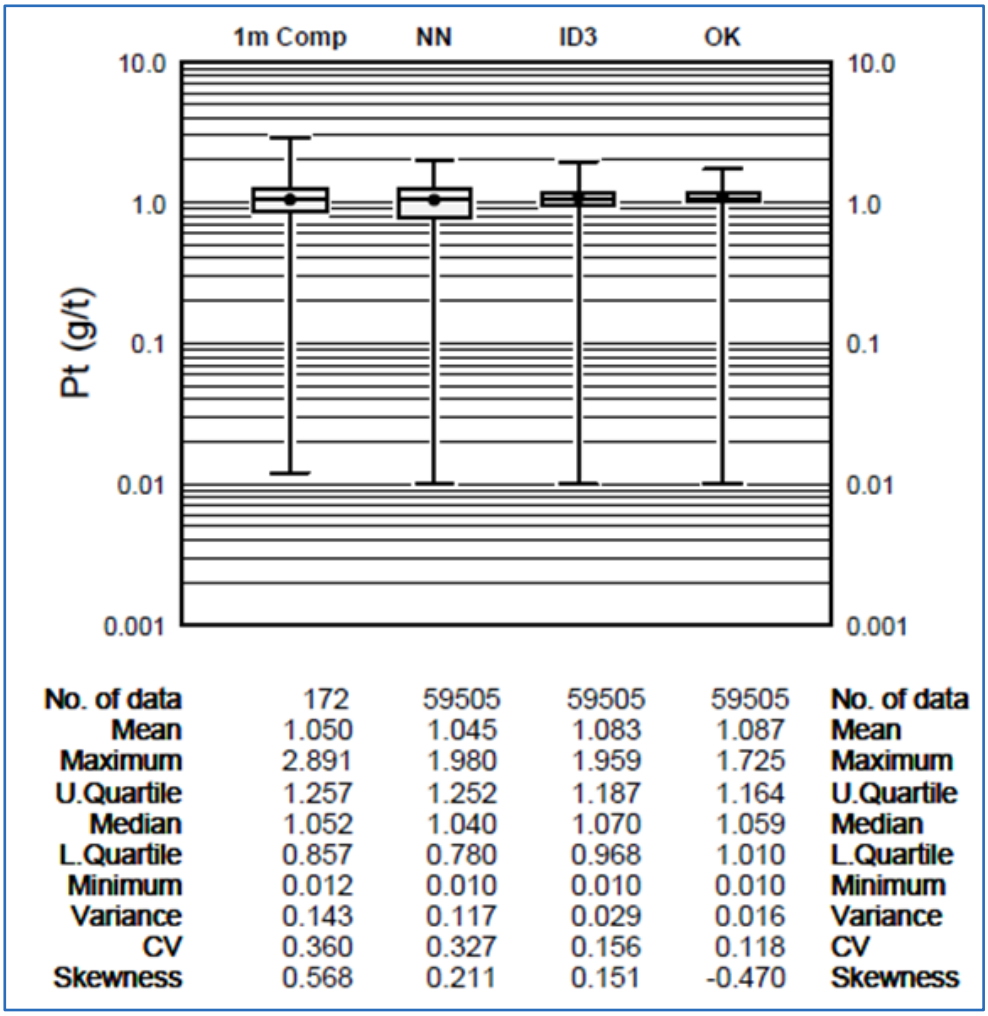
Note: Rows are shaded where the relative bias is outside (±5%).

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.

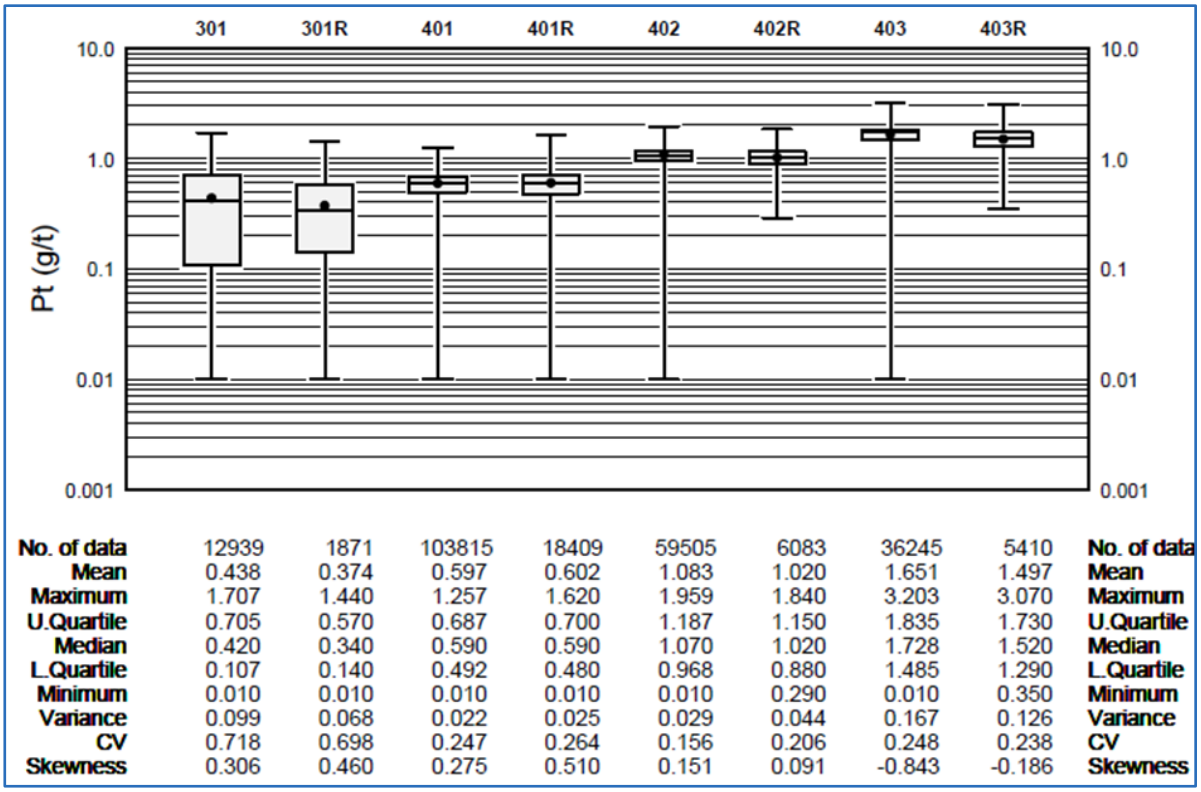
Figure 14.21 Box Plot Pt for B2MZ 2g/t 3PE+Au Figure Caption



	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

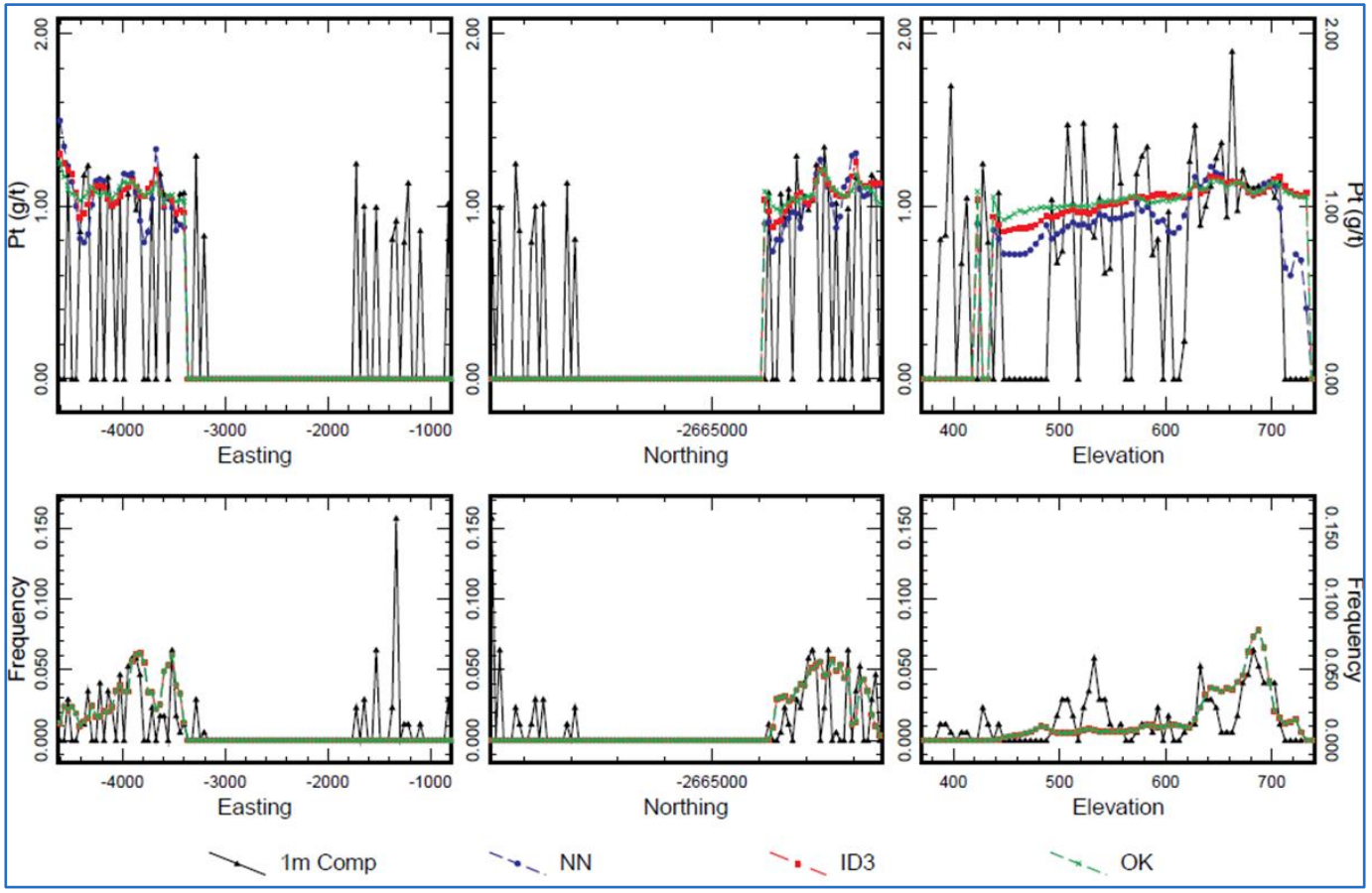
Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long-sections and level plans. In general, the composite grades were honoured in the block distributions. Representative cross-sections for 3PE+Au are shown in Figure 14.14 and Figure 14.16. 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 analyses 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 Pyroxenite-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 and 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 resulted from 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).

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

14.4.2 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 feldspar (FCPX) or olivine (OLCPX).

14.4.3 PNZ Domain

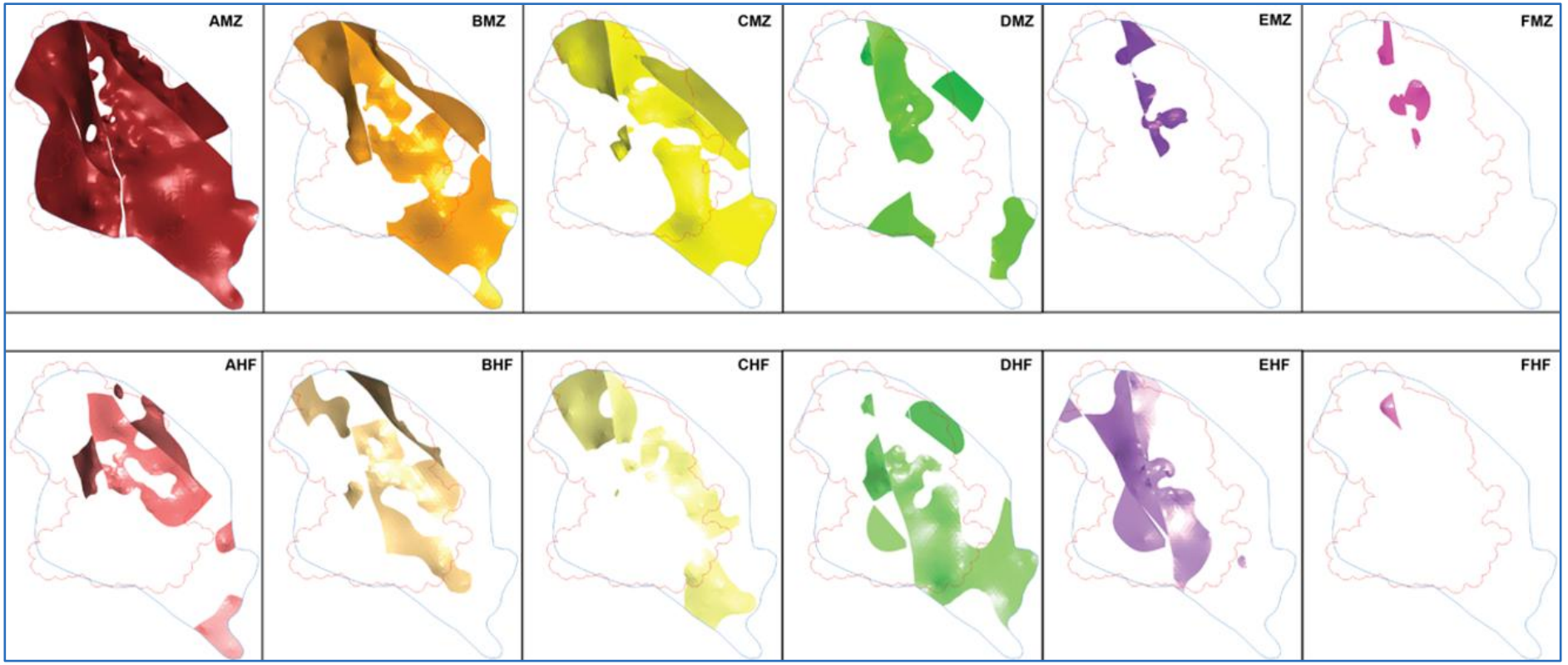
The PNZ is a unit of pyroxenite-norite composition that includes local unassimilated hornfels xenoliths. The mineralisation occurs predominantly as disseminated sulfides. Local massive sulfides are recognised at the contacts with the hornfels xenoliths. The 3PE+Au grades are commonly in the 1 g/t range, 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 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, CMZ, 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.

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 1 m 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	0.40	-	0.30	0.17	-	-
BHF	0.32	0.40	0.10	0.24	0.30	-	3.20
CHF	0.35	0.35	0.07	0.21	0.18	-	3.80
DHF	0.60	0.60	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 sub-celling (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 sub-celling 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 of 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 estimates 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. The grade estimates for the FWcpx domain are within the recommended ±5% tolerance. Generally, the grade estimates for the FWpnz domain 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-FWpnz 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

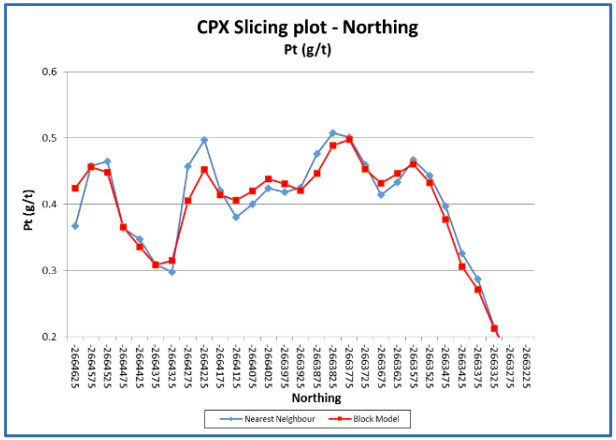
Visual validation of cross-sections for the FWcpx and FWpnz domains compared grade estimates and the 1 m composites. Figure 14.14 and Figure 14.15 shows the grade estimate for the FWcpx domain. Figure 14.16 and Figure 14.17 shows the grade estimate for the FWpnz domain.

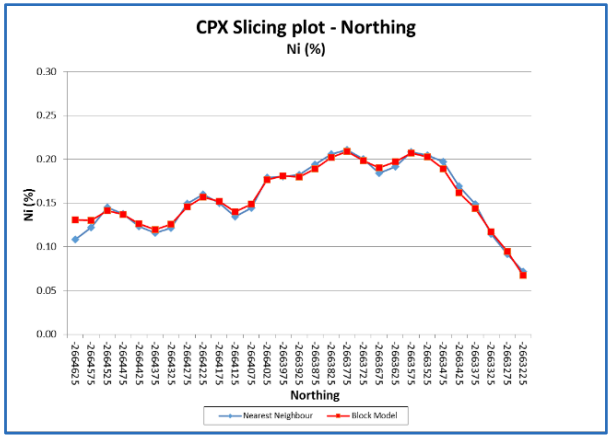
The visual inspections indicate that grade continuity is best observed at a 1.0 to 1.5 g/t 3PE+Au cut-off). At a 2 g/t 3PE+Au cut-off, mineralisation is more restricted, and additional drilling is required to fully define the mineralisation.

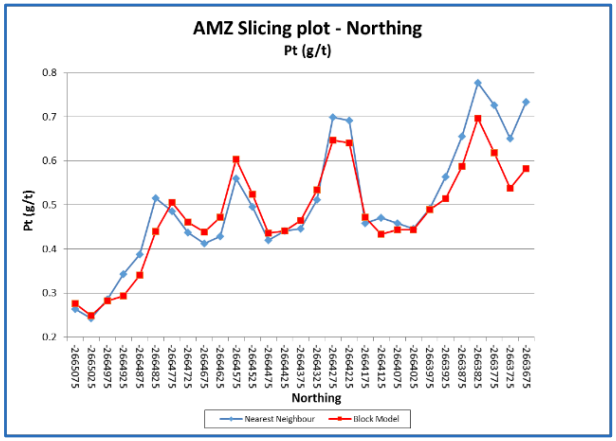
14.4.7.2 Swath Plots

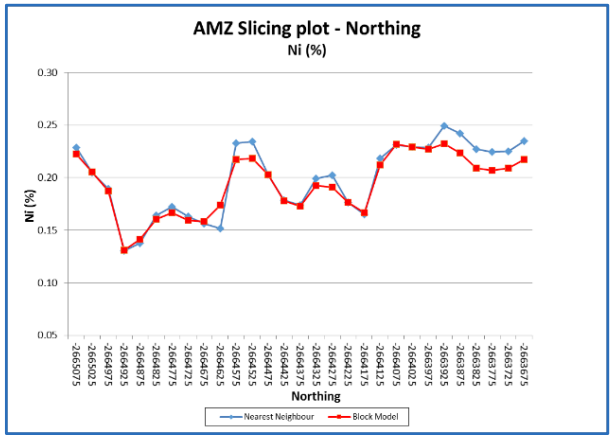
Swath plots were completed to compare grade estimation to the NN estimation and also to composites (see Figure 14.25).

Figure 14.25 Platinum and Nickel Swath Plots for CPX and AMZ (FWpnz) – 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 amount of drilling.

14.4.9 Regularisation

Upon completion of the estimation, the FWpnz and FWcpx models 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 amounts of 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

MSA undertook a conceptual analysis to assess reasonable prospects for eventual economic extraction for declaration of Mineral Resources. Assumptions were based on economic inputs used in this Platreef Integrated Development Plan 2025 (Refer to Item 22), which are based on a large ~4 Mtpa to ~10 Mtpa underground selective mining and concentrating operation.

- Commodity Prices
  - Platinum: 1200 $/t

  - Palladium: 1130 $/t
  - Rhodium: 5000 $/t
  - Gold: 2170 $/t
  - Copper: 4.25 $/lb
  - Nickel: 8.50 $/lb
- Metal Splits as derived for each mineralised zone from tabulations at 2g/t 3PE+Au
- Mining Dilution 5%
- Concentrator Recovery (average LOM)
  - Platinum: 87.2%
  - Palladium: 86.8%
  - Rhodium: 80.3%
  - Gold: 78.5%
  - Copper: 87.7%
  - Nickel: 71.6%
- Payability: 82%
  - This assumption is based on a survey made by Kramer (2012) and assumes transport to processing facilities in Rustenburg.
  - 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. MSA's conceptual analysis does not include treatment charges.
  - Platreef concentrates could also be marketed to smelters outside RSA.
- Royalty: 5%
- Onsite Operating Costs: 52 $/t

A net smelter return (NSR)was calculated based on the aforementioned assumptions using the average metal split for each mineralised zone. At a grade of 1.5 g/t 3PE+Au all mineralised zones have a NSR between 58 $/t and 69 $/t, which is above the site operating cost of 52 $/t. At a grade of 2 g/t 3PE+Au, mineralised zones have a NSR between 77 $/t and 92 $/t, which is well in excess of the site operating cost of 52 $/t.

Platinum Equivalent (PtEq) grades were calculated based on the aforementioned metal prices and concentrator recovery (other parameters are constant for each metal and are not required in the calculation). The recovered value of each metal in one tonne of rock was calculated and expressed as a percentage contribution to the total. The grade of platinum was then divided by its percentage contribution/100 to derive the PtEq grade.

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 MSA to provide likely RPEEE given on-site and additional costs. 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.

Table 14.23 shows Mineral Resources lying within and adjacent to the TCU mineralised zones. Table 14.24 provides the total Mineral Resource estimate for mineralisation lying within and adjacent to 3PE+Au grade shells for the Bikkuri Reef. Table 14.25 provides the total Mineral Resource estimate for mineralisation within the TCU-FW Model; the tabulation includes the FWcpx and the FWpnz resources.

Table 14.23 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 (%)**	**PtEq (g/t)**
3 g/t	202	2.11	2.12	0.34	0.14	4.71	0.18	0.35	7.02
2 g/t	339	1.68	1.71	0.28	0.11	3.79	0.16	0.32	5.83
1 g/t	685	1.13	1.18	0.20	0.08	2.59	0.13	0.26	4.22
Indicated Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		13.7	13.7	2.2	0.9	30.60	788	1,576	45.6
2 g/t		18.4	18.7	3.1	1.2	41.30	1,197	2,386	63.5
1 g/t		24.9	26.1	4.3	1.8	57.19	1,977	3,938	93.0
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 (%)**	**PtEq (g/t)**
3 g/t	212	1.93	1.95	0.32	0.13	4.33	0.17	0.35	6.59
2 g/t	459	1.45	1.48	0.27	0.10	3.29	0.16	0.31	5.29
1 g/t	1,213	0.91	0.96	0.18	0.07	2.12	0.13	0.25	3.68
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		13.1	13.3	2.2	0.9	29.4	802	1,625	44.9
2 g/t		21.3	21.9	3.9	1.4	48.6	1,591	3,103	78.0
1 g/t		35.4	37.5	6.9	2.7	82.5	3,472	6,579	143.7

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: 8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni, and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. 3PE+Au = platinum, palladium, rhodium and gold.
8. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by the assumed price and recovery for each metal.
9. Totals may vary due to rounding.

Table 14.24 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 (%)**	**PtEq (g/t)**
3 g/t	2	1.67	1.45	0.34	0.09	3.55	0.22	0.40	6.06
2 g/t	7	1.30	1.16	0.28	0.07	2.81	0.19	0.35	4.97
1 g/t	31	0.73	0.71	0.16	0.05	1.65	0.14	0.24	3.15
Indicated Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		0.13	0.12	0.03	0.01	0.28	12	22	0.39
2 g/t		0.29	0.26	0.06	0.01	0.62	29	52	1.12
1 g/t		0.74	0.71	0.17	0.05	1.67	99	170	3.14
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 (%)**	**PtEq (g/t)**
3 g/t	8	1.59	1.52	0.36	0.09	3.55	0.20	0.37	5.91
2 g/t	27	1.23	1.17	0.25	0.07	2.72	0.16	0.30	4.59
1 g/t	112	0.75	0.76	0.15	0.05	1.72	0.14	0.24	3.20
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		0.41	0.39	0.09	0.02	0.92	35	65	1.52
2 g/t		1.09	1.03	0.22	0.06	2.40	100	184	3.98
1 g/t		2.70	2.74	0.56	0.19	6.19	353	593	11.51

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: $8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. 3PE+Au = platinum, palladium, rhodium and gold.
8. 8. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by assumed price and recovery for each metal.
9. Totals may vary due to rounding.

Table 14.25 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 (%)**	**PtEq (g/t)**
2.5 g/t	9	1.44	1.66	0.24	0.09	3.43	0.22	0.39	5.82
2.0 g/t	20	1.15	1.34	0.19	0.08	2.76	0.19	0.34	4.84
1.5 g/t	49	0.88	1.04	0.15	0.07	2.14	0.16	0.29	3.91
1.0 g/t	105	0.66	0.81	0.11	0.07	1.65	0.13	0.25	3.18
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
2.5 g/t		0.43	0.49	0.07	0.03	1.02	45	80	1.68
2.0 g/t		0.75	0.87	0.12	0.05	1.79	84	153	3.11
1.5 g/t		1.39	1.65	0.23	0.11	3.38	169	318	6.16
1.0 g/t		2.23	2.73	0.39	0.23	5.58	304	587	10.74

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: $8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. 3PE+Au = platinum, palladium, rhodium and gold.
8. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by assumed price and recovery for each metal.
9. Totals may vary due to rounding.

14.7.1 UMT-FWcpx

Table 14.26 provides the portion of the TCU-FW Mineral Resource estimate for mineralisation within the CPX domain.

Table 14.26 Mineral Resource Estimates for the FWcpx 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 (%)**	**PtEq (g/t)**
2.5 g/t	4	1.34	1.60	0.21	0.09	3.24	0.19	0.40	5.58
2.0 g/t	10	1.08	1.29	0.17	0.08	2.63	0.16	0.34	4.62
1.5 g/t	27	0.82	1.00	0.14	0.07	2.03	0.14	0.29	3.75
1.0 g/t	58	0.64	0.78	0.11	0.06	1.60	0.11	0.25	3.04
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
2.5 g/t		0.17	0.20	0.03	0.01	0.41	16	35	0.72
2.0 g/t		0.33	0.40	0.05	0.02	0.81	34	73	1.48
1.5 g/t		0.71	0.86	0.12	0.06	1.74	80	170	3.26
1.0 g/t		1.19	1.47	0.21	0.12	2.99	144	317	5.68

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: $8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by assumed price and recovery for each metal.
8. 3PE+Au = platinum, palladium, rhodium and gold.
9. Totals may vary due to rounding.
10. The FWcpx domain includes zones of low grade mineralisation that may permit mass mining methods at a lower cut-off.
11. Mineral Resources in Table 14.26 are included in the tabulations in Table 14.25 and are not additive to that table.

14.7.2 UMT-FWpnz

Table 14.27 provides the portion of the TCU-FW Mineral Resource estimate for mineralisation within the PNZ Domain.

Table 14.27 Mineral Resource Estimates for the FWpnz 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 (%)**	**PtEq (g/t)**
2.5 g/t	6	1.51	1.68	0.25	0.08	3.52	0.24	0.39	5.94
2.0 g/t	11	1.21	1.39	0.21	0.08	2.88	0.21	0.34	5.03
1.5 g/t	22	0.94	1.09	0.16	0.07	2.27	0.18	0.30	4.13
1.0 g/t	47	0.69	0.83	0.12	0.07	1.71	0.15	0.26	3.33
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
2.5 g/t		0.27	0.30	0.04	0.01	0.63	29	45	1.15
2.0 g/t		0.41	0.47	0.07	0.03	0.99	50	81	1.78
1.5 g/t		0.68	0.79	0.12	0.05	1.64	89	148	2.92
1.0 g/t		1.04	1.26	0.18	0.11	2.59	160	270	5.04

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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
4. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
5. 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.
6. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: $8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
7. 3PE+Au = platinum, palladium, rhodium and gold.
8. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by assumed price and recovery for each metal.
9. Totals may vary due to rounding.
10. 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.
11. Mineral Resources in Table 14.27 are included in the tabulations in Table 14.25 and are not additive to that table.

14.7.3 Combined Mineral Resources

Table 14.28 provides a summary of the combined Platreef Mineral Resources for the UMT-TCU, UMT-BIK and UMT-FW models.

Table 14.28 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 (%)**	**PtEq (g/t)**
3 g/t	204	2.11	2.11	0.34	0.14	4.70	0.18	0.35	7.01
2 g/t	346	1.68	1.70	0.28	0.11	3.77	0.16	0.32	5.82
1 g/t	716	1.11	1.16	0.19	0.08	2.55	0.13	0.26	4.17
Indicated Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		13.9	13.9	2.2	0.9	30.9	800	1,597	46.0
2 g/t		18.7	18.9	3.1	1.2	41.9	1,226	2,438	64.7
1 g/t		25.6	26.8	4.5	1.8	58.8	2,076	4,108	95.9
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 (%)**	**PtEq (g/t)**
3 g/t	225	1.91	1.93	0.32	0.13	4.29	0.17	0.35	6.55
2 g/t	506	1.42	1.46	0.26	0.10	3.24	0.16	0.31	5.22
1 g/t	1,431	0.88	0.94	0.17	0.07	2.05	0.13	0.25	3.62
Inferred Mineral Resources - Contained Metal									
Cut-off 3PE+Au	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu Mlbs)**	**Ni (Mlbs)**	**PtEq (Moz)**
3 g/t		13.8	14.0	2.3	1.0	31.0	865	1,736	47.4
2 g/t		23.2	23.8	4.3	1.6	52.8	1,775	3,440	84.9
1 g/t		40.4	43.0	7.8	3.1	94.3	4,129	7,759	166.5

1. The 2 g/t 3PE+Au 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 15 February 2025 to confirm the estimates as current. Therefore, the effective date of the Platreef Mineral Resource is 15 February 2025. The Qualified Person for the estimate is Mr Jeremy Witley, Pr. Sci. Nat. 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.
3. The table shows sensitivity to cut-off and the rows are not additive.
4. Mineral Resources are reported on a 100% basis. Ivanhoe holds a 64% interest in the Project.
5. Mineral Resources are stated from approximately –200–650 m elevation (from 500–1,350 m depth).
6. 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.
7. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,200/oz, Pd: $1,130/oz, Au: $2,170/oz, Rh: $5,000/oz, Cu: $4.25/lb and Ni: $8.50/lb. It has been assumed that payable metals would be 82% from smelter/refinery of Pt, Pd and Au, 82.5% for Rh, 73.0% for Cu and 70.0% for Ni and that mining costs and process, G&A, and concentrate transport costs (average $52/t of mill feed) would be covered. The average concentrator recoveries over the life-of-mine are 87.2% for Pt, 86.8% for Pd, 80.3% for Rh, 78.5% for Au, 87.7% for Cu, and 71.6% for Ni. Mining dilution 5%.
8. 3PE+Au = platinum, palladium, rhodium and gold.
9. PtEq = platinum equivalent calculated using the relative contribution to the value of Pd, Au, Rh, Ni and Cu to the total value weighted by assumed price and recovery for each metal.

10. Totals may vary due to rounding.
11. Mineral Resources reported in Table 14.28 are included in Table 14.23, Table 14.24, Table 14.25(note that Table 14.26 and Table 14.27 are included in Table 14.25) 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. Caution should be placed on these estimates as 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.

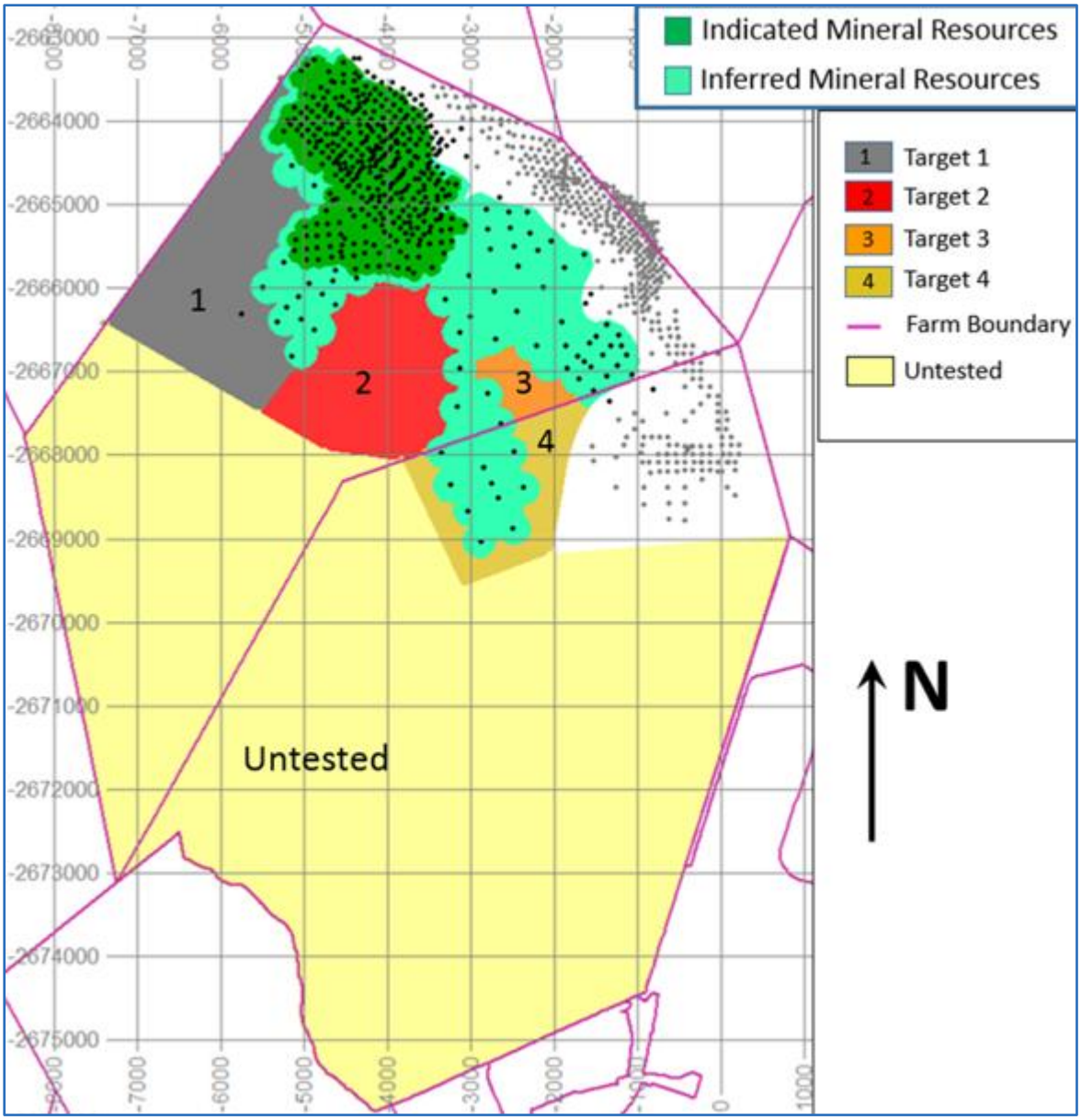
The Bushveld Complex (BC) PGE-Ni-Cu deposits have characteristics of lateral continuity over several thousands of metres. Based on this, four exploration targets have been identified (Figure 14.26). Target areas are defined based on the 2016 Mineral Resource Model, and represent currently undrilled extension 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 south-west.

Figure 14.26 Exploration Target Areas



Wood, 2016

14.9 Comments on Section 14

Mr Jeremy Witley is of the opinion that the Mineral Resources for the Platreef Project, which have been estimated using drill core 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 prices as used in the economic analysis in this Platreef Integrated Development Plan 2025. 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, 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.

It is noted 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 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.

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 the Northern Limb of the Bushveld 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.

15.2 Resource Block Model

The probable mineral reserves available for mining at the Platreef were based on the resource block model previously reported and used in the Platreef 2022 FS.

The model 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 IDP25 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 the 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 15.1 and Table 15.2.

Table 15.1 NSR20 Metal Prices

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 NSR20 Charges and Other Assumptions

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

15.3 Cut-Off Grades

A cut-off grade analysis was completed by mining method, and the resulting cut-off grade ranged from $47–$58/t. Ivanplats' operational philosophy required additional allowances for capital and profit margin to be included in the NSR cut-off evaluation. Based on BDT20 the 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.

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, 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. This was done in the studies of 2015 and 2017, although it has not been updated in this study the Platreef operation team is currently reevaluating all stope designs. Current analysis determined the Platreef 2025 FS designs are valid. Until 2029 a cut-off grade for low grade development of 1.75 g/t 3PE+Au has been used. After 2029 the cut-off grade is $130/t NSR.

15.4 Mineral Reserve

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in the Platreef 2025 FS. Only Indicated Mineral Resources have been used for determination of the Probable Mineral Reserve.

The Mineral Resource block model also includes the NSR calculated using BDT20. 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.

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 cut-off grades for each mining method, exclude capital recovery and profit margin.

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.

The Platreef 2025 FS cost estimates have been done to a feasibility study level of accuracy.

Table 15.3 shows the Probable Mineral Reserve for Platreef.

Table 15.3 Platreef Mineral Reserve as at 15 February 2025

Classification	Ore (Mt)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Proven	-	-	-	-	-	-	-	-
Probable	129.7	1.88	1.93	0.29	0.13	4.22	0.16	0.33
Total	**129.7**	**1.88**	**1.93**	**0.29**	**0.13**	**4.22**	**0.16**	**0.33**

Classification	Ore (Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (kt)	Ni (kt)
Proven	-	-	-	-	-	-	-	-
Probable	129.7	7.82	8.05	1.21	0.54	17.62	209	426
Total	**129.7**	**7.82**	**8.05**	**1.21**	**0.54**	**17.62**	**209**	**426**

1. Mineral Reserves have an effective date of 15 February 2025. The Qualified Person for the estimate is Mr 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 Pt, $815/oz Pd, $1,300/oz Au, $1,500/oz Rh, $8.90/lb Ni and $3.00/lb Cu.
4. A declining NSR cut-off of $155/t–$80/t was used for the Mineral Reserve estimates.
5. Long-term metal-price assumptions used for the feasibility study economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Totals may vary due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

16 MINING METHODS

16.1 Mine Geotechnical

The work from the Platreef 2017 FS and optimisation studies in December 2022 and January 2023 was utilised in the Platreef 2025 FS study to mitigate potential risks associated with the Platreef 2022 FS mining approach. These studies aimed to reduce risks and incorporate new technology and skills while achieving the required ramp up. A significant geotechnical change involved targeting longhole open stopes (LHOS) and drift and fill (DF) stopes above 850 m Level near the shaft system first, to expedite development to target areas. DF will be used in areas where the ore zone thickness is less than 15 m and LHOS will be used where the ore zone thickness exceeds 15 m, with stopes oriented transversely.

No additions have been made to the geotechnical database since the FS 2017 and FS 2022 reports, so the geotechnical rock mass data and laboratory testwork remain unchanged. However, the geotechnical model is presented differently from the previous studies. Geotechnical characteristics are influenced significantly by different lithologies, and structural features and this section now presents these effects more appropriately and practically. The SRK team visited underground access drives on 950 m Level in 2023 to gain an appreciation of the prevailing ground conditions.

The study included a geotechnical stability investigation of the revised mine design and sequenced layouts, evaluating stress-related interactions between excavations and support requirements. Assessment of different sequencing strategies for the drift and fill and longhole stope mining layouts were made, and recommendations were provided.

Geotechnical design parameters were derived to manage potential risks, governing stope and mine access design, including backfill and support requirements. The mine design has been reviewed in line with these geotechnical parameters.

16.1.1 Geotechnical Model

There were no additions to the geotechnical database used in the 2017 and 2022 FS reports, However, the interpretation of the geotechnical model was revised to consider the lithological and structural influences on rock mass characteristics instead of a purely statistical approach.

16.1.1.1 Geotechnical Data

In total, 83 boreholes were used in the geotechnical investigation which are listed in Table 16.1 and the locations presented in Figure 16.1. The approximate length of core logged per rock type forming each stratigraphic unit is shown graphically in Figure 16.2. From this, it is evident that the MZ, FAZ and T1 units have the largest meterage and are well represented in relation to the NC2 unit, which have the least amount of core intersections.

Table 16.1 Geotechnically Logged Boreholes

Borehole IDs						
GT007	UMT070	UMT100	UMT131	UMT160	UMT256	UMT357
GT008	UMT073	UMT102	UMT133	UMT165	UMT257	UMT357D1
GT012	UMT074	UMT103	UMT135	UMT181	UMT258	UMT358
GT013	UMT075	UMT104	UMT136	UMT182	UMT259	UMT359
GT013D1	UMT077	UMT109A	UMT138	UMT186	UMT262	UMT361
GT014	UMT078	UMT117	UMT139	UMT189	UMT265D1	UMT371
GT015D1	UMT079	UMT119	UMT142	UMT201	UMT298	UMT372
GT016	UMT080	UMT120	UMT145	UMT222	UMT299	UMT373
GT017	UMT083	UMT123	UMT146	UMT230	UMT301	UMT382
GT017D1	UMT090	UMT125	UMT147	UMT237	UMT310	UMT404
UMT068	UMT095	UMT126	UMT150	UMT244	UMT313	UMT447
UMT069	UMT096	UMT130	UMT155	UMT247	UMT319	

Figure 16.1 Plan of all the Drillholes for the Platreef FS 2017

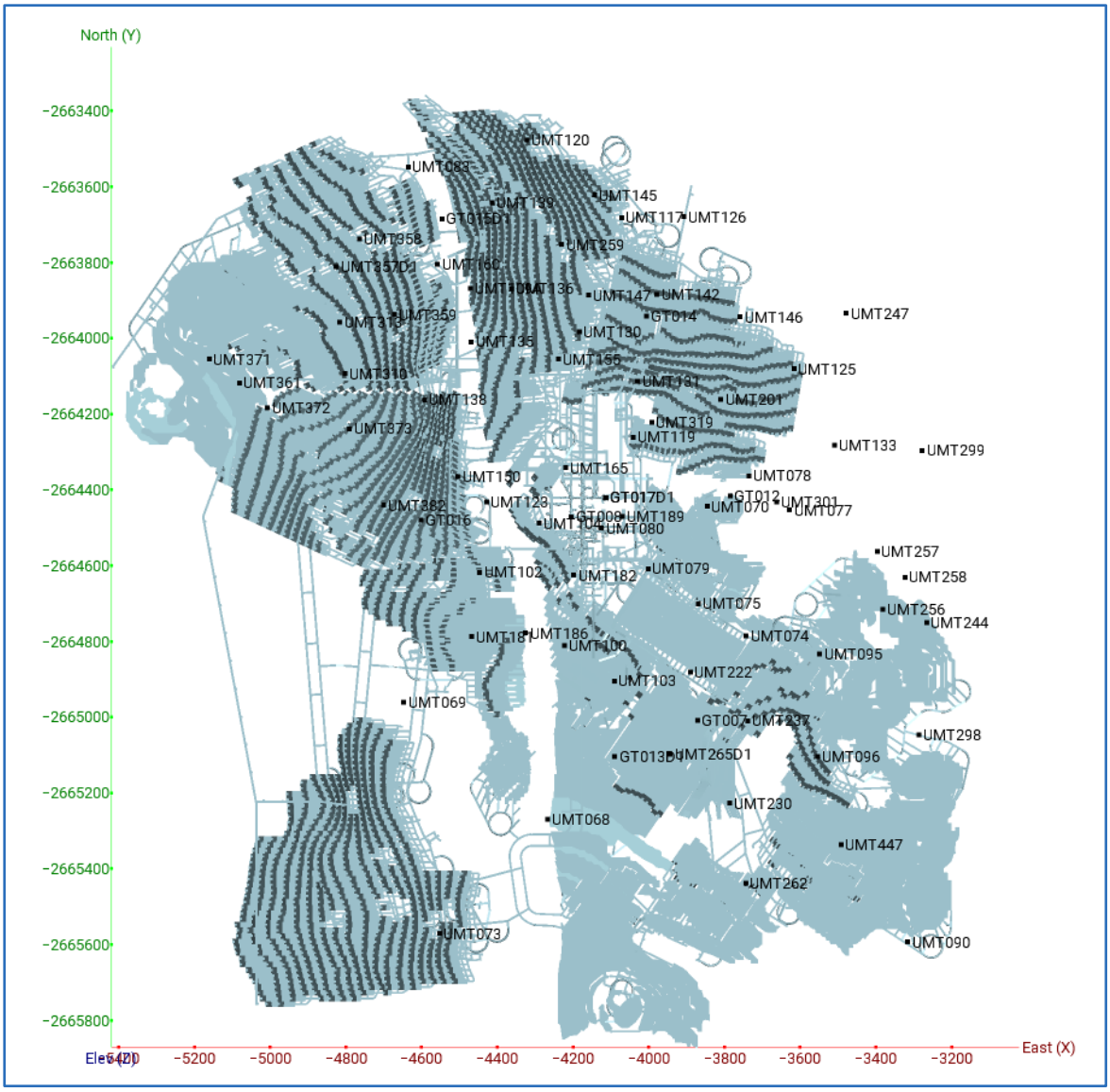
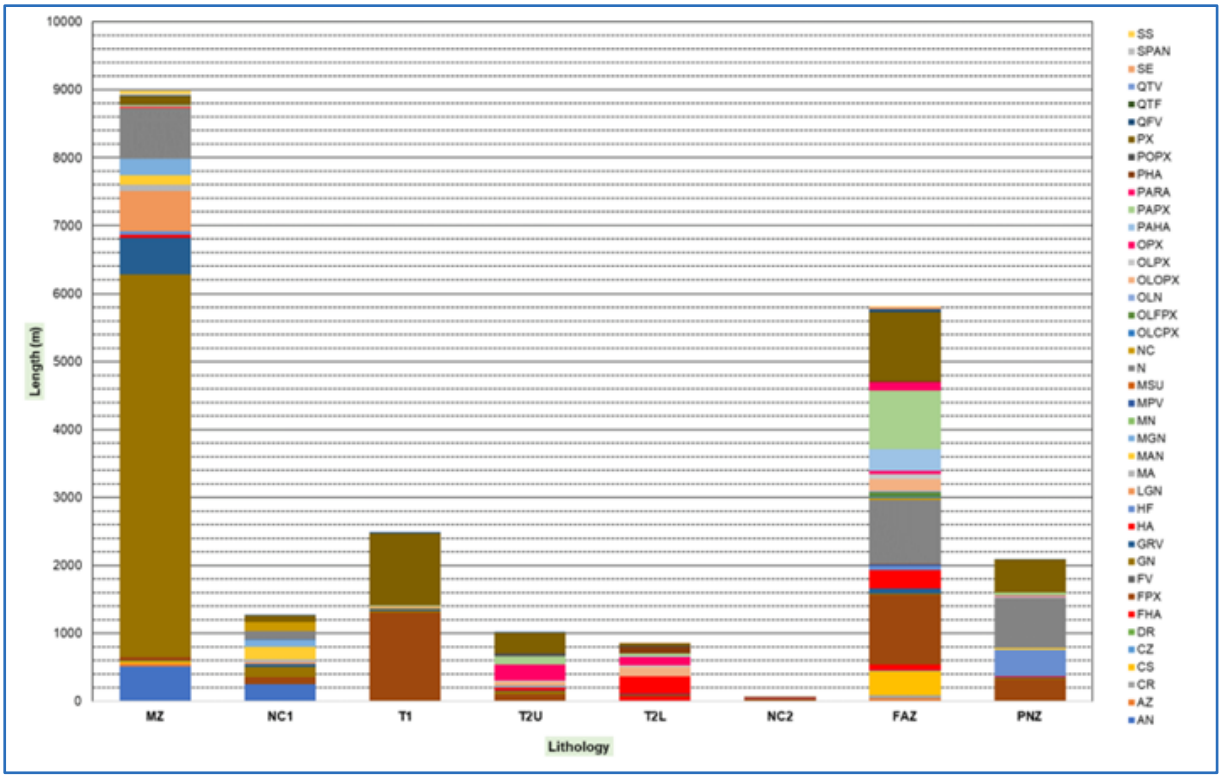


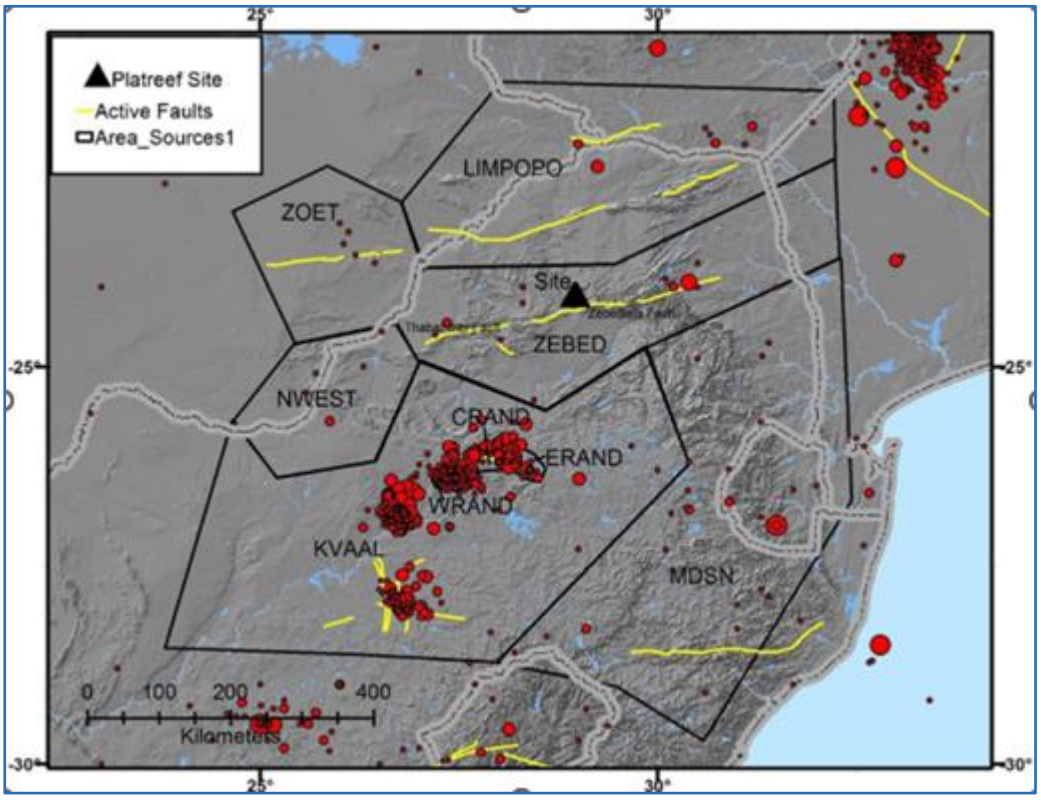
Figure 16.2 Approximate Length of Core Logged Per Rock Type



16.1.1.2 Regional Seismicity

Local seismicity in Limpopo is generally classified as low, though earthquakes have been recorded in the region. Most natural seismicity near the mine site is along the Murchison Greenstone Belt, Zebediela Fault, and especially the Thabazimbi Fault. The Platreef Project is located in the ZEBED zone, and only seismic events with magnitudes of $M_W \geq 4$ are plotted.

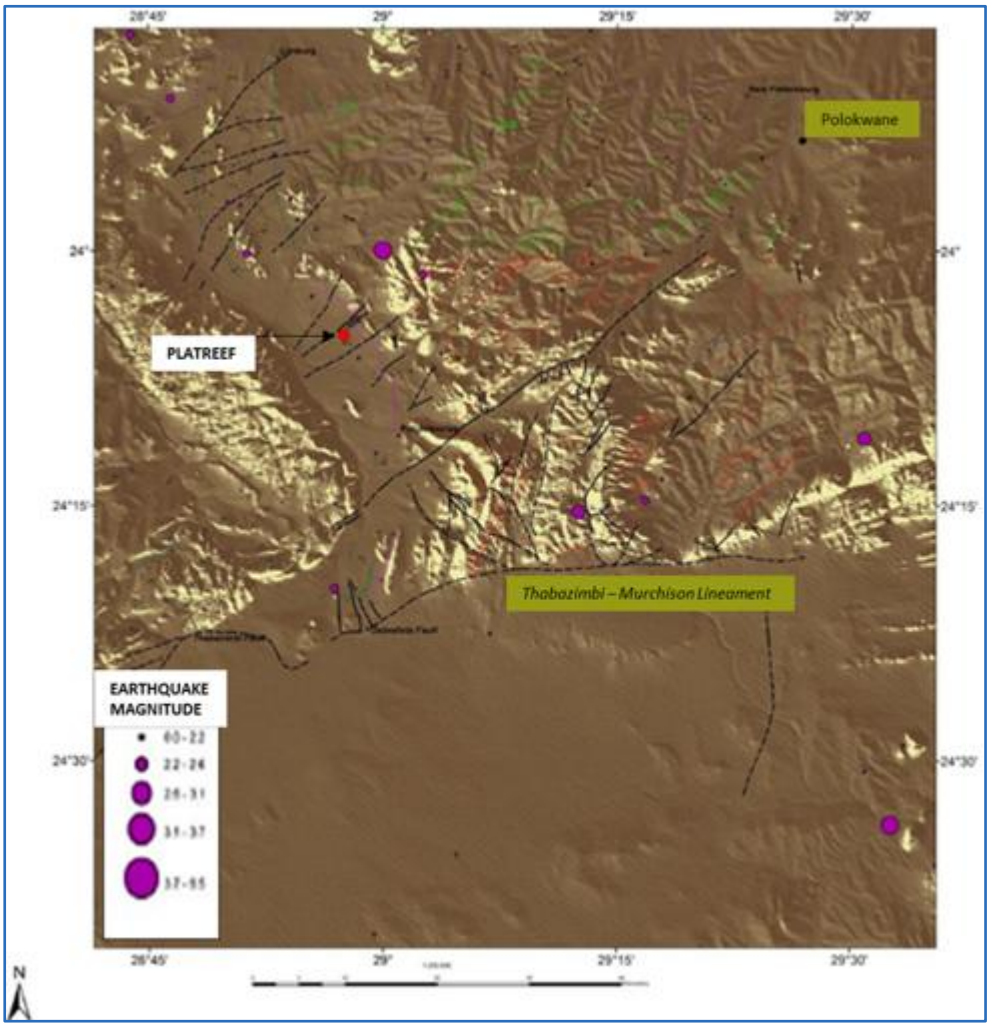
Figure 16.3 Spatial Distribution of Earthquakes ($M_W \geq 4.0$) and Seismic Source Zones used in the Seismic Hazard Calculation



Midzi et al., 2015

Most of the natural seismicity near the mine site appears to be located along the lineament defined by the Murchison Greenstone Belt, Zebediela fault and especially the Thabazimbi Fault (Figure 16.3). A structural map showing earthquake epicentres located 50 km and less from site is shown in Figure 16.4.

Figure 16.4 Structural Map of the Area Located Within 50 km of the Mine Site Indicated by the Red Dot



Midzi et al., 2015

Surface structures are more vulnerable than underground infrastructure due to large surface waves associated with natural earthquakes. Major geological structures on the Platreef property haven't been linked to large-magnitude tectonic earthquakes.

The planned mining will cause significant stress changes and induce seismicity on major geological structures. Support strategies which include measures for highly stressed and dynamic conditions have been catered for, and seismic monitoring, to manage the risk of rock bursts underground and regional surface vibrations.

16.1.1.3 Stress Regime

A stress measurement programme was conducted in 2015 to better understand the stress regime in the Platreef area. Samples were selected from depths between 750 m and 1,110 m below ground level from available core in seven boreholes (GTO12, GT013, GT014, GT015, GT016, UMT361 and UMT371). They were sent to the Australian Centre for Geomechanics at the University of Western Australia, where in-situ stress directions were determined using Deformation Rate Analysis and Acoustic Emission methods.

The test results indicated that the stress ratio was moderate to low at all locations, with consistent magnitudes but variable orientations. No correlation between stress measurements and domains could be established. Due to this variation, the stress field was analysed based on the structural geological information provided in the structural report (The structural interpretation of the farms Turfspruit and Macalacaskop,1051-RP-80-001, 2015) and the FS report (DRA-J0283-STU-REP-909 S5 20160831_REVA2, 2016).

The regional structural events highlighted two-fold orientations that have been observed and concur with the previous Northern Limb studies. Combining the two events creates a composite sigma 1 (σ1) axis that would be trending approximately W-E to WNW-ESE (Figure 16.5). Most of the σ1 directions from the stress measurements fall within these two trends or are very close to it. The stress data correlates with the tectonic history. The stress state tabulated in Table 16.2 was applied to the elastic modelling of the full mine design.

Figure 16.5 Composite Stress Envelopes

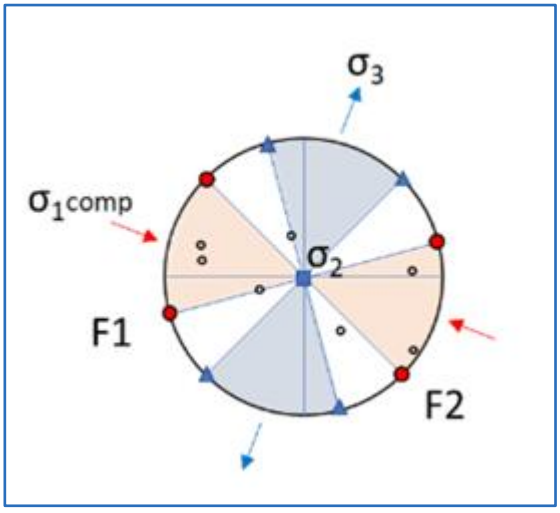


Table 16.2 σ1 Resultant Stress State

k1 Trend	Horizontal Stress 1 (k1)	Horizontal Stress 2 (k2)	Vertical Gradient (MPa/m)
114	1.1	0.9	-0.032

16.1.1.4 Rock Mass Classification

The Q System (Barton et al, 1974) and modified Q' System (Barton and Grimstad, 1993 and Barton, 2002) were used for delineating ground conditions, support recommendations and stope design aspects.

Rock mass conditions range from very poor to extremely good, but most of the rock mass is generally fair to good (interquartile range between 5 and 60). Lower rock mass quality should be expected, and excavations in these areas may require more stringent support or experience increased overbreak. Limited core was available for geotechnical analysis in the NC2 unit, resulting in low confidence in their geotechnical characterisation. More information is needed for assessments of development infrastructure in deeper mine areas. Boreholes will need to intersect and surpass the LZ, as current boreholes were terminated there.

Figure 16.6 and Figure 16.7 present the Q' ranges of the stratigraphic units and each of the lithological units, respectively.

Figure 16.6 Box and Whisker Plot Showing Extreme Range, Interquartile Range and Median Values of Q' Per Stratigraphic Unit

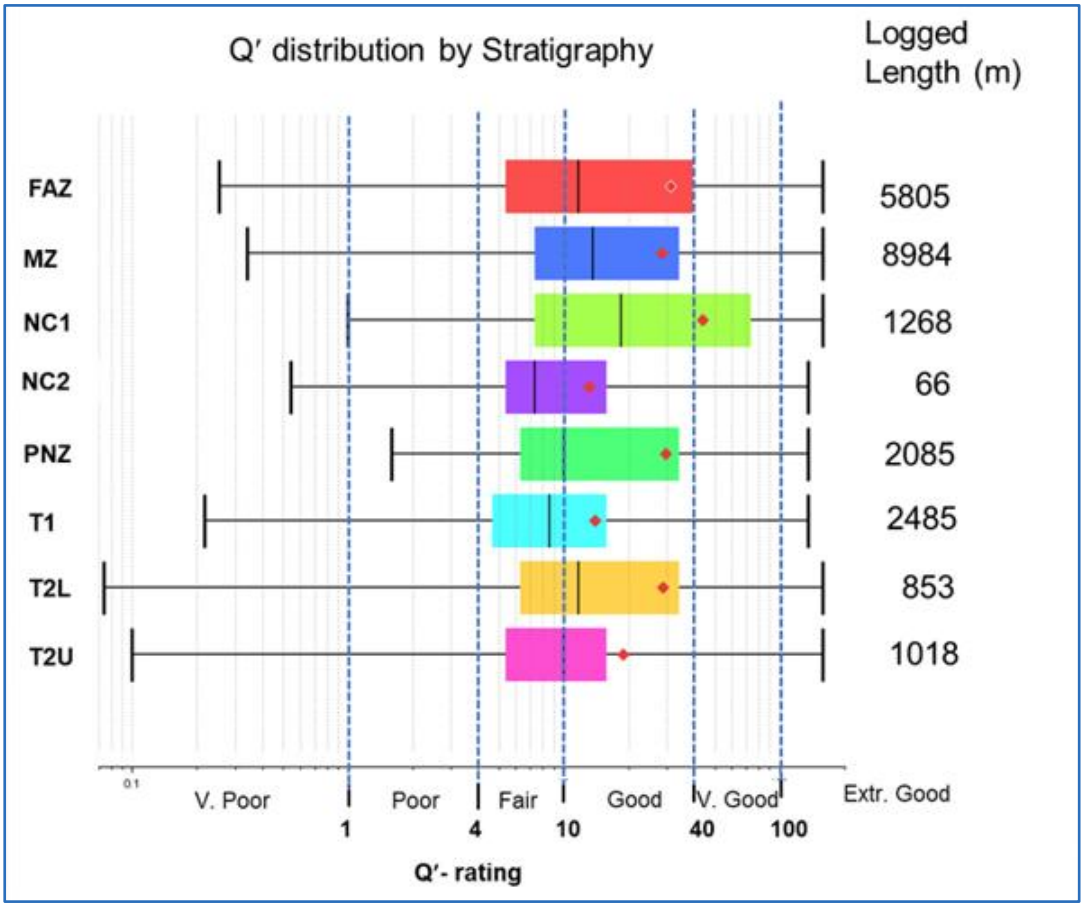
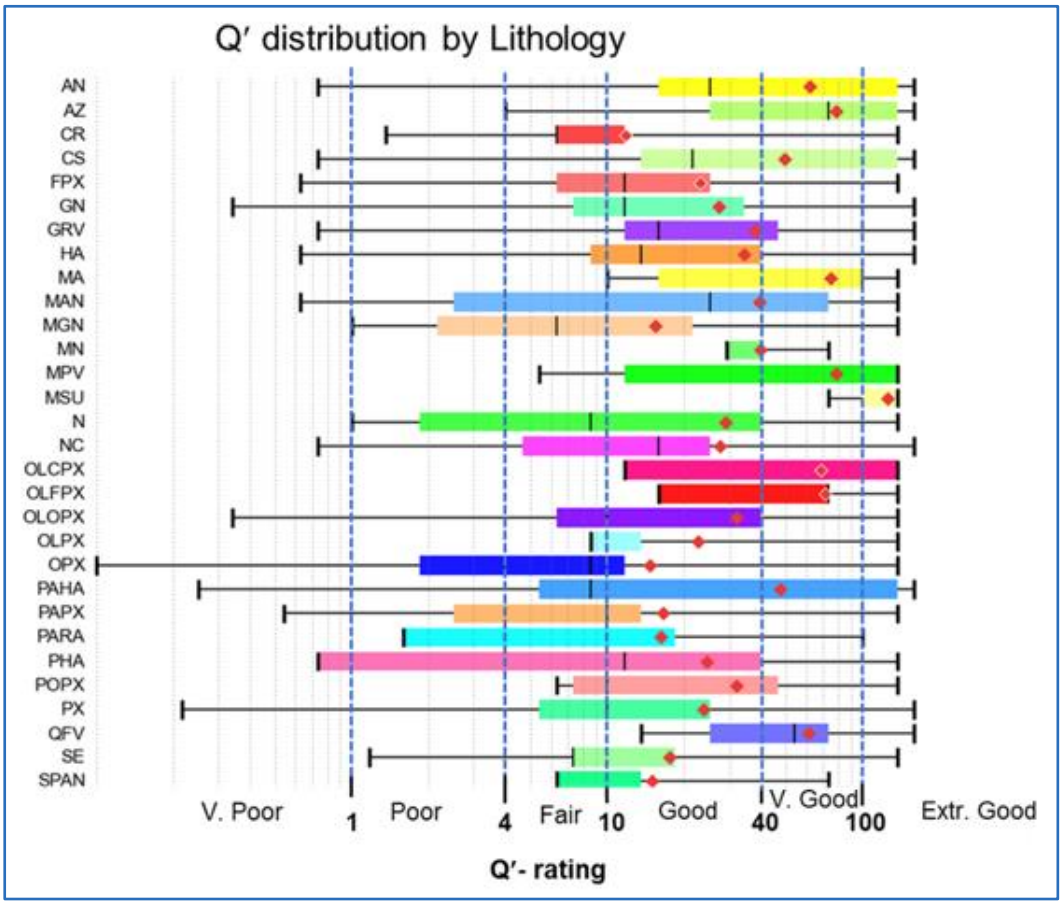


Figure 16.7 Box and Whisker Plot Showing Extreme Range, Interquartile Range and Median Values of Q' Per Lithology



16.1.1.5 Intact Rock and Rock Mass Properties

Summaries of the rock mass and intact material properties are tabulated in Table 16.3 and Table 16.4. These were extracted from the laboratory testing programmes, sample selection, processing and analysis of the results in the 2016 FS report and were utilised in the assessments of the support strategy, elastic modelling of the mine layout, inelastic simulation of sill pillar extraction and the empirical stope design.

Table 16.3 Rock Strength Properties

Stratigraphy	MZ	NC1	T1	T2U	T2L	NC2	FAZ	PNZ
No. of Samples	102	49	68	58	47	14	87	43
UCS Mean (MPa)	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
m_i	20	21	10	10	10	14	11	17

Table 16.4 Intact Elastic Material Properties

Stratigraphy	No. of Tests	Results	Minimum	Mean	Maximum	Standard Deviation
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

The material properties (Table 16.5) used in the numerical analysis were determined from the average strength parameters of the reef and footwall rock, where the mine design access development is mainly sited.

Table 16.5 Input Parameters for the Models

Material	Density (MN/m^3)	σ_{ci} (MPa)	Ei (MPa)	Poisson's ratio
Rock	0.032	187	89000	0.3

16.1.1.6 Geotechnical Characteristics of Major Structures

The prominent geological features identified on Platreef mining area and considered in the mine design are illustrated in Figure 16.8 and a summary of their respective structural trends provided in Table 16.6.

Figure 16.8 Plan View of Major Features Traversing the Project Area

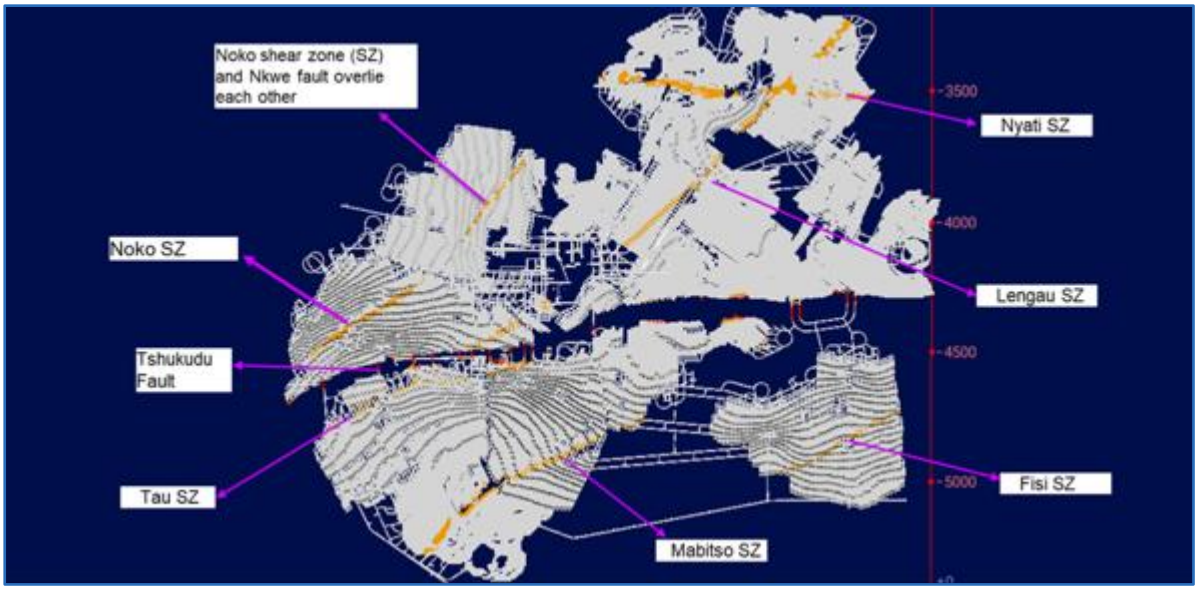


Table 16.6 Orientation of Major Structures

Structure	Orientation	Dip Direction
Tshukudu Fault and Nyati Shear Zone	50° to 70°	East
Nkwe Fault and Related Structures	55°	North-East
LAF's (040, 045, 050) and Associated Granitic Veins (LGVs)	40° to 50°	North-East
T2U Chromite Seams	40° to 45°	Variable

The Tshukudu fault is the largest structure observed in the Project area. This fault 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 the ductile shear zones.

Poor rock mass conditions are associated with the Tshukudu and Nkwe Faults, such as dense zones of sympathetic jointing and friable infill, and water ingress are associated with these features which may cause major geotechnical challenges warranting additional support requirements.

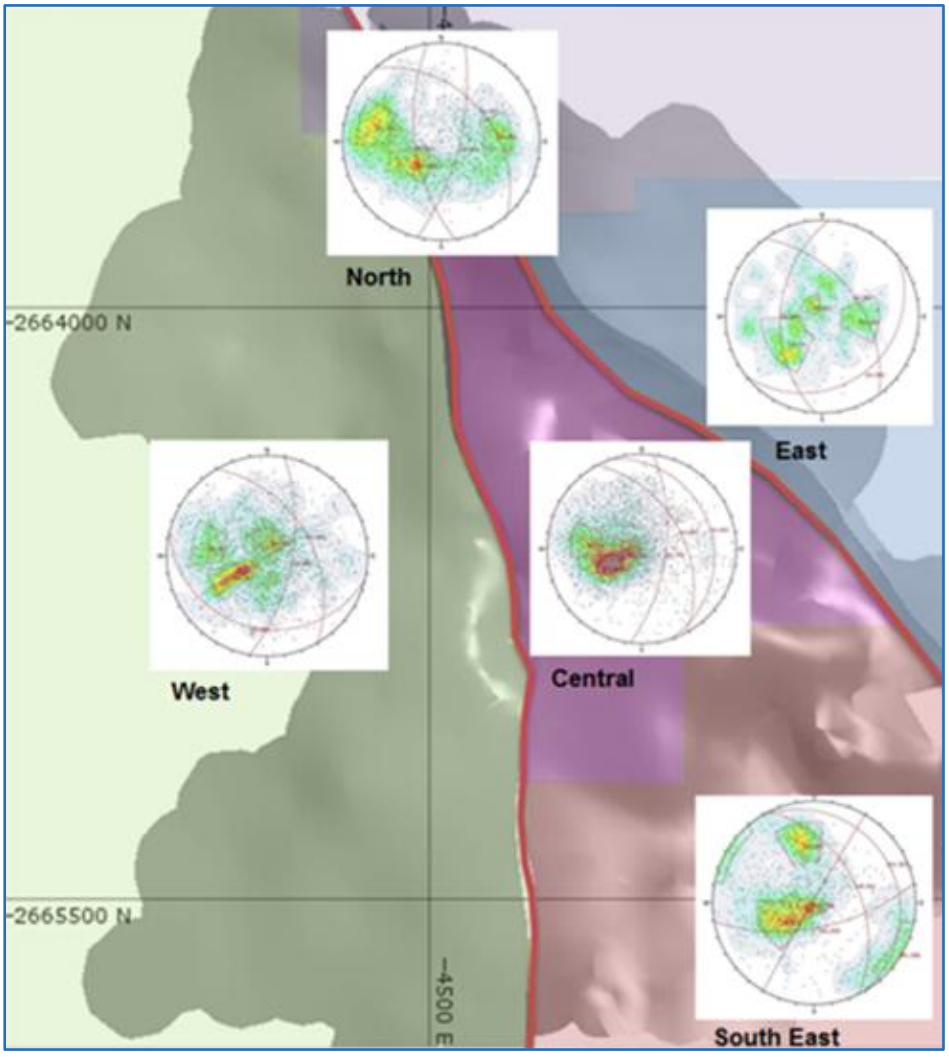
The granitic veins (LGVs) are mostly competent hard rock commonly bound by sharp contacts, except where the contact is associated with low-angle features (LAFs) with altered clay material gouge, which may pose a fall-of-ground risk.

The chromite seams associated with the T2U reef horizon can occur as single or multiple (splaying) stringers typically known to act as release surfaces, creating a potentially unstable beam resulting in large falls of ground. The positioning of excavations in relation to the T2U Cr will have an influence on hanging wall stability, support requirements, cost, and productivity.

16.1.1.7 Geotechnical Characteristics of Joints

Five structural domains were identified in the 2017 FS (SRK Report 482446) as illustrated in Figure 16.9, with the mining footprint intersecting the Tshukudu Fault along the western boundary and the Nkwe Fault along the NE section of the Project area.

Figure 16.9 Identified Joint Sets for each Structural Domain



A summary of the dip angle and dip direction for the six major joint sets per structural domain is provided in Table 16.7, with the updated orientation data shown in brackets. No additional joint sets were identified.

Table 16.7 Summary of Joint Sets for each Structural Domain

Domain		North	West	Central	South-East	East
Joint Set 1	Min Dip	20	19	31	27	26
	Mean Dip Angle (°)	**36**	**38**	**42 (39)**	**42**	**47**
	Max Dip	50	63	56	59	65
	Min Dip Direction	022	042	042	034	024
	Mean Dip Direction (°)	**048**	**060**	**062**	**064**	**054**
	Max Dip Direction	075	075	078	091	087
Joint Set 2	Min Dip	57	49	49	-	-
	Mean Dip Angle (°)	**69**	**62**	**61 (57)**	**-**	**-**
	Max Dip	81	74	74	-	-
	Min Dip Direction	88	83	083	-	-
	Mean Dip Direction (°)	**104**	**100**	**097 (098)**	**-**	**-**
	Max Dip Direction	120	118	111	-	-
Joint Set 3	Min Dip	-	08	06	04	08
	Mean Dip Angle (°)	**-**	**18**	**16 (14)**	**11**	**18**
	Max Dip	-	30	26	18	27
	Min Dip Direction	-	143	028	16	105
	Mean Dip Direction (°)	**-**	**194**	**072 (074)**	**169**	**142**
	Max Dip Direction	-	240	112	105	172
Joint Set 4	Min Dip	48	-	-	-	23
	Mean Dip Angle (°)	**60**	**-**	**-**	**-**	**43**
	Max Dip	71	-	-	-	62
	Min Dip Direction	243	-	-	-	249
	Mean Dip Direction (°)	**262**	**-**	**-**	**-**	**274**
	Max Dip Direction	277	-	-	-	298
Joint Set 5	Min Dip	-	-	-	46	-
	Mean Dip Angle (°)	**-**	**-**	**-**	**62**	**-**
	Max Dip	-	-	-	77	-
	Min Dip Direction	-	-	-	151	-
	Mean Dip Direction (°)	**-**	**-**	**-**	**169**	**-**
	Max Dip Direction	-	-	-	186	-

Domain		North	West	Central	South-East	East
Joint Set 6	Min Dip	-	-	-	83	-
	Mean Dip Angle (°)	**-**	**-**	**-**	**88**	**-**
	Max Dip	-	-	-	90	-
	Min Dip Direction	-	-	-	281	-
	Mean Dip Direction (°)	**-**	**-**	**-**	**300**	**-**
	Max Dip Direction	-	-	-	318	-

16.1.1.8 Site Visit

In June 2023, the SRK geotechnical team visited the Platreef Mine to familiarise themselves with the underground rock mass conditions. The purpose of the visit was to conduct a geotechnical assessment of critical areas selected by the Rock Engineer. This assessment enabled the team to characterise the rock mass conditions, map some mined out excavations and evaluate the effectiveness of the support systems currently in place.

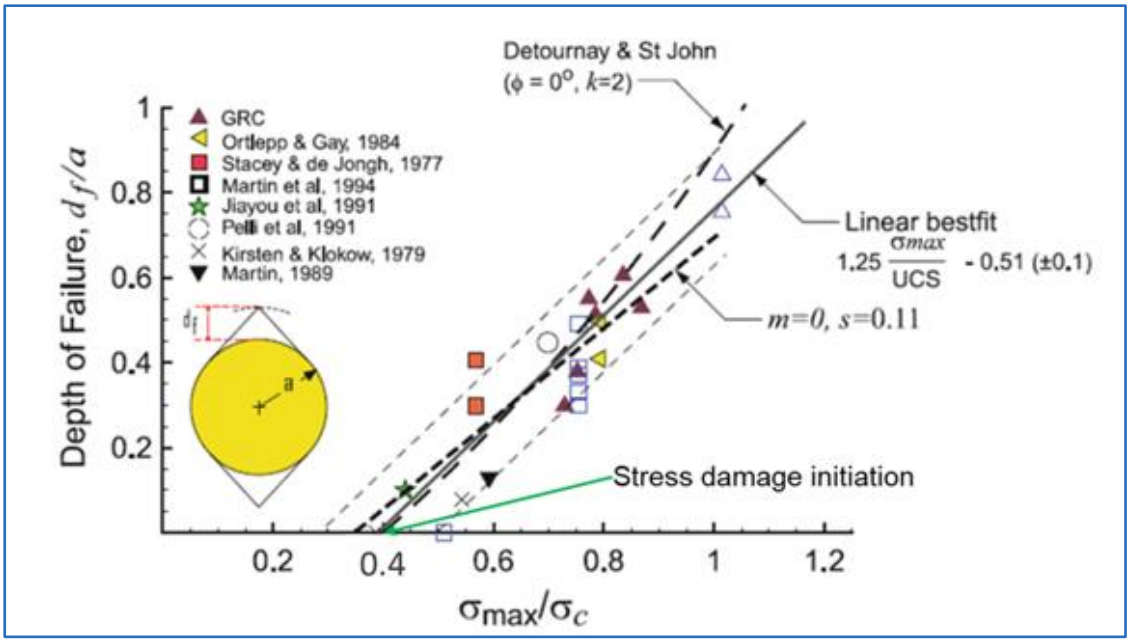
16.1.2 Geotechnical Design

The bulk of the underground infrastructure is sited within a shaft pillar surrounding Shafts 1 and 2. This includes the rock handling system, workshops, explosive systems and dams.

The main mining method is longhole open stoping and drift and fill is carried out where the orebody is relatively thin. The production stope drifts will be initially developed at 7 m x 6 m width and height during ramp up and will be reduced to 6 m x 5 m width and height in full production phase. These drives will be developed at the top and bottom of the stopes. The sub-level stope accesses are horizontally off-set by 15 m from the longhole stopes to mitigate stress damage. Longhole stopes are planned to be 15 m wide x 20 m high, with maximum length prior to backfilling of 60 m.

Elastic modelling of the full mine in yearly steps was carried out using the Rocscience finite element modelling program, RS3 to estimate the potential for stress damage in excavations and seismicity on the major geological structures. This approach is necessary, due to the size of the model, and associated computational time. The depth of failure from the stress levels, for different critical long-term access drives and drifts, was analysed using the empirical relationship between the depth of failure in an excavation and the stress ratio suggested by Martin et. al (1999) (Figure 16.10). The depth of failure (DF) was then used to determine the support requirements based on the stress damage levels the walls of excavations (Table 16.8). This method is appropriate as long as the stress damage is limited to the walls of the excavation. When the sill pillars are formed, the stress damage becomes excessive, and the elastic modelling is no longer representative. Therefore, a conceptual elasto-plastic RS3 model was constructed to represent formation and extraction of sill pillars. Precedent practice with sill pillar extraction in hard rock mines in Canada was also used to evaluate the optimal mining sequence, which includes just in time development in fractured rock masses.

Figure 16.10 Stress Damage Criterion



Martin et. al, 1999

Table 16.8 Stress Levels for Different Support Categories

DF value	DF/a	σmax/σc	σmax (MPa)
0.3	0.09	0.3	60
0.5	0.14	0.5	95
1.5	0.43	0.7	130

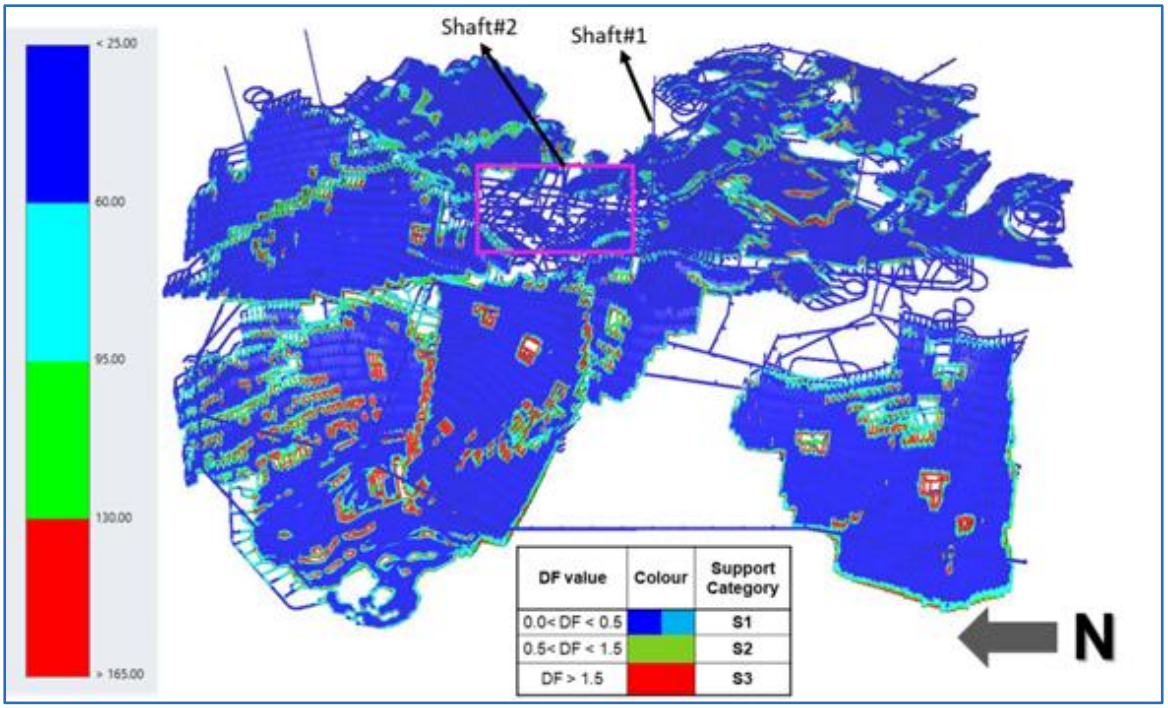
In addition to the stress modelling, block stability analyses were carried out using the Rocscience software program, Unwedge to determine the cable support requirements.

Stope stability and potential overbreak was estimated using empirical methods. Backfill strength requirements were estimated using analytical methods.

16.1.2.1 Critical Infrastructure

The shaft pillar area (Figure 16.11) will experience minimal stress damage during the life-of-mine (LOM) and S1 support is considered sufficient. Additional support will be required when these excavations intersect major structures.

Figure 16.11 Stresses Around Shaft Pillar Area



Critical infrastructure is located away from major geological structures and supported appropriately for the anticipated conditions. Large excavations should not intersect the Tshukudu fault, because the alteration could result in severe unravelling, and it would be impractical to excavate and support. Figure 16.12 shows the excavations that are intersected by other geological structures on 750 m Level, 850 m Level and 950 m Level. Additional support in the form of cable anchors is required, where these structures intersect excavations.

Figure 16.12 Large Excavations Intersected by Geological Structures on Main Levels

Wedge stability analyses were carried out for the large excavations on 750 m Level, 850 m Level and 950 m Level in the mine design to determine the cable anchor support requirements. The analysis was carried out for different large excavation sizes that were categorised based on size and orientation. These large excavations are located in the central geotechnical domain where joint sets J1, J2 and J3 have been identified to be persistent during the 2017 joint characterisation exercise.

Table 16.9 lists all the large excavations analysed.

Table 16.9 List of Designed Large Excavations

Excavation	Width (m)	Height (m)	Length (m)	Orientation (°)	Excavation Category
750 m Level Truck tip #1 (East)	11	8	11.0	0	A1
750 m Level Truck tip #2 (West)	11	8	11.0	0	A1
850 m Level Truck Tip #1 (East)	11	8	11.0	0	A1
850 m Level Truck Tip #2 (West)	11	8	11.0	0	A1
950 m Level Crusher, 950 Level Truck Tip #1	11	8	11.0	0	A1
950 m Level Crusher, 950 Level Truck Tip #2	11	8	11.0	0	A1
750 m Level Truck tip (Shaft #1)	11	8	18.7	90	A2
910 m Level Crusher, 750 Level Truck Tip	11	8	11.0	90	A2
850 m Level Truck tip (Shaft #1)	11	8	18.7	90	A2
910 m Level Crusher, 850 Level Truck Tip	11	8	11.0	90	A2

Excavation	Width (m)	Height (m)	Length (m)	Orientation (°)	Excavation Category
Shaft #3 Conveyor #2 (Bottom of Crusher)	10	12	68.0	90	B
750 m Level Material Transfer	9	6	57.0	0	C
850 m Level Material Transfer	9	6	57.0	0	C
950 m Level Material Transfer	9	6	57.0	0	C
750 m Level - Shaft #3 Fan	9	6	19.0	0	C
950 m Level Crusher, 950 Level Coarse Bin Bottom #1	9	8	30.0	0	D
Shaft #3 Conveyor #6 (Bin Feed Conveyor)	9	10	22.0	90	E
Ore Bunker x 2	8	6	16.0	0	F1
950 m Level Shaft #2 Loading Level	8	6	16.0	0	F1
750 m Level Emulsion Bay	8	9	19.5	0	G1
850 m Level Emulsion Bay	8	9	19.5	0	G1
Large Bay x 5	8	9	30.0	110	G2
Medium Bay x 5	8	7	30.0	110	H1
Medium Bay (Breakdowns) x 4	8	7	30.0	110	H1
950 m Level Crusher Fines Bin Bottom (c/w Feeder and Bulkhead)	8	7	11.0	0	H2
Shaft #3 Conveyor #2 (Bottom of Crusher)	7	6	7.0	0	I1
Shaft #3 Conveyor #6 (Bin Feed Conveyor)	7	6	7.0	0	I1
Conveyor Decline	7	6	50.0	0	I2
950 m Level Crusher (c/w Crusher, Feeder and Bulkhead)	7	15	23.0	90	J
750 m Level Receiving Station (Paste)	6	6	12.0	0	K
750 m Level Explosives System x 2	6	6	12.0	0	K
850 m Level Receiving Station (Paste)	6	6	12.0	0	K
850 m Level Explosives System x 2	6	6	12.0	0	K
950 m Level Receiving Station (Paste)	6	6	12.0	0	K
950 m Level Explosives System	6	6	12.0	0	K
950 m Level Dirty Water Dam	6	6	12.0	0	K
950 m Level LHD Tip	6	11	9.0	90	L
950 m Level Shaft #2 Silo Bottom	8	7	14.0	90	L
Shaft #3 Conveyor #3 (N-S Lift)	7	6	7.0	0	I1

The analysis was carried out for roof wedges since they were considered most critical. The orientations of the excavations were measured from the full mine design model and used in the analysis. Table 16.10 summarises the support recommendations for the large excavations.

Table 16.10 Results Summary for Large Excavation Support

Excavation Category	Width (m)	Height (m)	Length (m)	Orientation (°)	Joint Combination	Wedge Weight (kN)	FoS	Support Pattern
A1	11	8	19	0	245	4,738	2.0	Primary Support (S1) + 6m cable anchor in a 2m x 2m pattern
A2	11	8	19	90	245	919	2.5	Primary Support (S1) + 6m cable anchor in a 3m x 3m pattern
B	10	12	68	90	245	690	2.4	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
C	9	6	57	0	245	503	2.3	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
D	9	8	30	0	245	2,846	1.9	Primary Support (S1) + 4.5m cable anchor in a 2m x 2m pattern
E	9	10	22	90	245	503	3.1	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
F1	8	6	16	0	245	1,999	1.9	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
F2	8	6	33	90	245	354	3.5	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
G1	8	9	20	0	245	1 999	1.9	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern

Excavation Category	Width (m)	Height (m)	Length (m)	Orientation (°)	Joint Combination	Wedge Weight (kN)	FoS	Support Pattern
G2	8	9	30	110	245	374	4.6	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
H1	8	7	30	110	245	374	4.6	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
H2	8	7	11	0	245	921	2.4	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
I1	11	8	19	90	245	919	2.5	Primary Support (S1) + 6m cable anchor in a 3m x 3m pattern
I2	10	12	68	90	245	690	2.4	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
J	9	6	57	0	245	503	2.3	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern
K	9	8	30	0	245	2 846	1.9	Primary Support (S1) + 4.5m cable anchor in a 2m x 2m pattern
L	9	10	22	90	245	503	3.1	Primary Support (S1) + 4.5m cable anchor in a 3m x 3m pattern

There are seven vertical shafts within the shaft pillar area. These are critical excavations, which should not be adversely affected by stress changes over the life-of-mine. They should as far as possible be sited in good quality rock masses, but where unfavourable ground conditions are encountered, adequate support should be installed to ensure their long-term stability, for the required purpose. All the shafts traverse weathered rock near surface, which varies in thickness. The bulk of the length of the shafts will be in competent gabbronorite, and depending on their location, will traverse through several deeper lithologies.

The shafts were in various stages of development at the time of completing this report. The status of geotechnical investigations and development for each shaft is listed in Table 16.11.

Table 16.11 Summary of Platreef Vertical Shafts

Shaft Name	Sinking Methodology	Shaft Diameter (m)	Stage of development	Support	Comments
Shaft#1	Blind sinking	7.25	Completed	1.8 m split sets Mesh 300mm concrete lining	Geotechnical investigations were completed by SRK. The planned depth of the shaft was reduced to avoid intersecting the Tshukudu fault, which was highly altered in the core. No major issues encountered during sinking.
Shaft#2	Raise bore & sliping	10	In progress	1.8 m split sets Mesh 300mm concrete	Geotechnical investigations completed by SRK. Raisebore at 3.1 m diameter and slipe to full diameter. There are very weak zones at 38 m–44 m to 50 m, at 57 m–62 m and at 65 m–69 m. Several long, weak intervals to a depth of 70 m below surface, indicating potential unravelling. Low advance rate when reaming through weak zone.
Vent Shaft#3	Raise bore	5.1	In progress	Unlined	Geotechnical investigations completed by Platreef. 50 mm shotcrete support recommended where poor ground conditions are encountered. Shaft stopped at 750 m depth before intersecting Tshukudu fault. Weak zones at 358 m–363 m and at 740 m–750 m depths. Fair conditions from approximately 548 m depth and below. Shotcrete weak zone to mitigate risk. Low advance rate when reaming through weak zone.

Shaft Name	Sinking Methodology	Shaft Diameter (m)	Stage of development	Support	Comments
Vent Shaft#4	Raise bore	5	In progress (pilot hole completed)	Unlined	Geotechnical investigations completed by SRK. 5 m diameter to be implemented as recommended (*599326-Raisebore ventilation shaft stability analysis_Memo)* by SRK. Weak granitic zone at 62 m–69 m and pyroxenite shear zone at 215 m–221 m depths. Shotcrete weak zone to mitigate risk. Low advance rate when reaming through weak zone.
Vent Shaft#5	Raise bore	4.5 or 5	Geotechnical drilling to confirm ground conditions underway	Unlined	Initial geotechnical investigation of nearby drillhole completed by SRK. Shear/fault zone with three joint sets at 16 m–31 m and at 221 m–228 m depths (*599326-Raisebore ventilation shaft #5 stability analysis_Memo*). Additional geotechnical drillhole at shaft location to assess shear/fault zone to finalise decision.
Vent Shaft#6	Raise bore	5.1 (proposed)	Not commenced	Unlined	Geotechnical investigation is planned. No major faults anticipated. The reef horizon is compressed and unusual at 415 m depth, which requires investigation.
Vent Shaft#7	Raise bore	5.1 (proposed)	Not commenced	Unlined	Geotechnical investigations are planned. The base of the shaft is near the Tshukudu fault which will need to be investigated.

The silos/ore passes are mainly placed in the Lower Zones (LZ1 and LZ2). Ore pass longevity is influenced by ground conditions, ore pass configuration, operational considerations, and ground support. Unexpected failures could have a significant effect on production, if redundancy is not built into the system. The ore pass longevity was estimated for each ore pass, using a method developed by Hadjigeorgiou and Mercier-Langevin (2008). The method includes a Longevity Reduction Factor (LRF) and a Longevity Extension Factor (LEF).

The analysis indicated that the estimated longevity for the analysed silos and ore passes are lower than their required throughput during their operational life. All the designed silos/ore passes will need to be rehabilitated or replaced. Shaft #3 ore passes and 950 m Level Silo #2 will need to be rehabilitated halfway into their life to achieve the required throughput. 750 m Level and 850 m Level Silo #2 and 1050 m Level Silo #2 will require to be rehabilitated several times during their life due to high throughput tonnages expected and this will affect production. Table 16.12 shows the comparison between the estimated silo/ore pass longevity with different support, and the required ore pass life.

Table 16.12 Summary of Rock Handling System Wear Rate

Silo/Ore pass	Silo/Ore pass Anticipated Life (Mt)				Required Life (Mt)
	No Support	Abrasion Resistant Lining (ARL)		30mm ARL plus Cable Bolting in Fault Zone	
		150 mm	300 mm		
Shaft #3 Ore pass 1	3.0	4.2	5.1	7.1	14.4
Shaft #3 Ore pass 2	5.4	7.6	9.2	12.9	20.6
Shaft #2 Silo 1	5.4	7.6	9.2	12.9	117.0
750 m & 850 m Level Silo 2	6.0	8.4	10.2	14.3	52.5
950 m Level Silo 2	4.2	5.9	7.1	10.0	26.0
1,050 m Level Silo 2	6.0	8.4	10.2	14.3	38.0

16.1.2.2 Longhole Stope Design

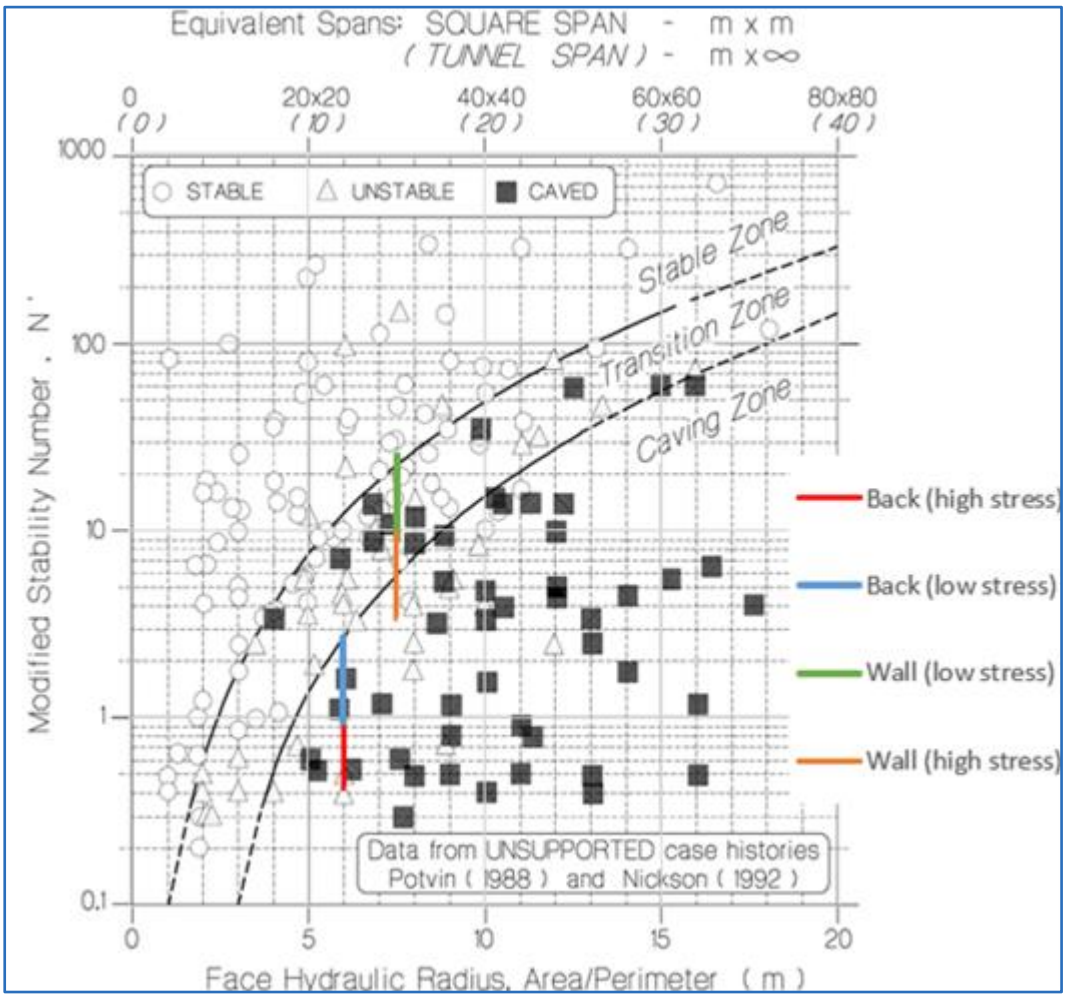
Longhole stopes are planned to be 15 m wide x 20 m high, with maximum length prior to backfilling of 60 m. Most longhole stopes are designed with top and bottom access for drilling and blasting from the supported top drive and ore removal from the bottom. Less than 5% of stopes will be blasted bottom-up. The top access and downhole drilling method helps manage overbreak due to the supported top drive. Additional support in the form of cable anchors will be installed if there are poor ground conditions or major geological structures. Tight backfill from the top drive limits overbreak during secondary stope mining. The vertical stopes walls cannot be supported but are inherently more stable due to the geometry.

The stope stability assessment was carried out using the empirical method developed by Mathews et al (1981) and modified by Potvin (1988), Potvin and Milne (1992) and further by Nickson (1992).

Stope backs are not stable without support, indicating major overbreak, though only a small proportion will be unsupported. With support installed in the top drive, stope backs will remain stable even under high stress. The stope wall plots in the transition zone, indicating significant overbreak can be anticipated.

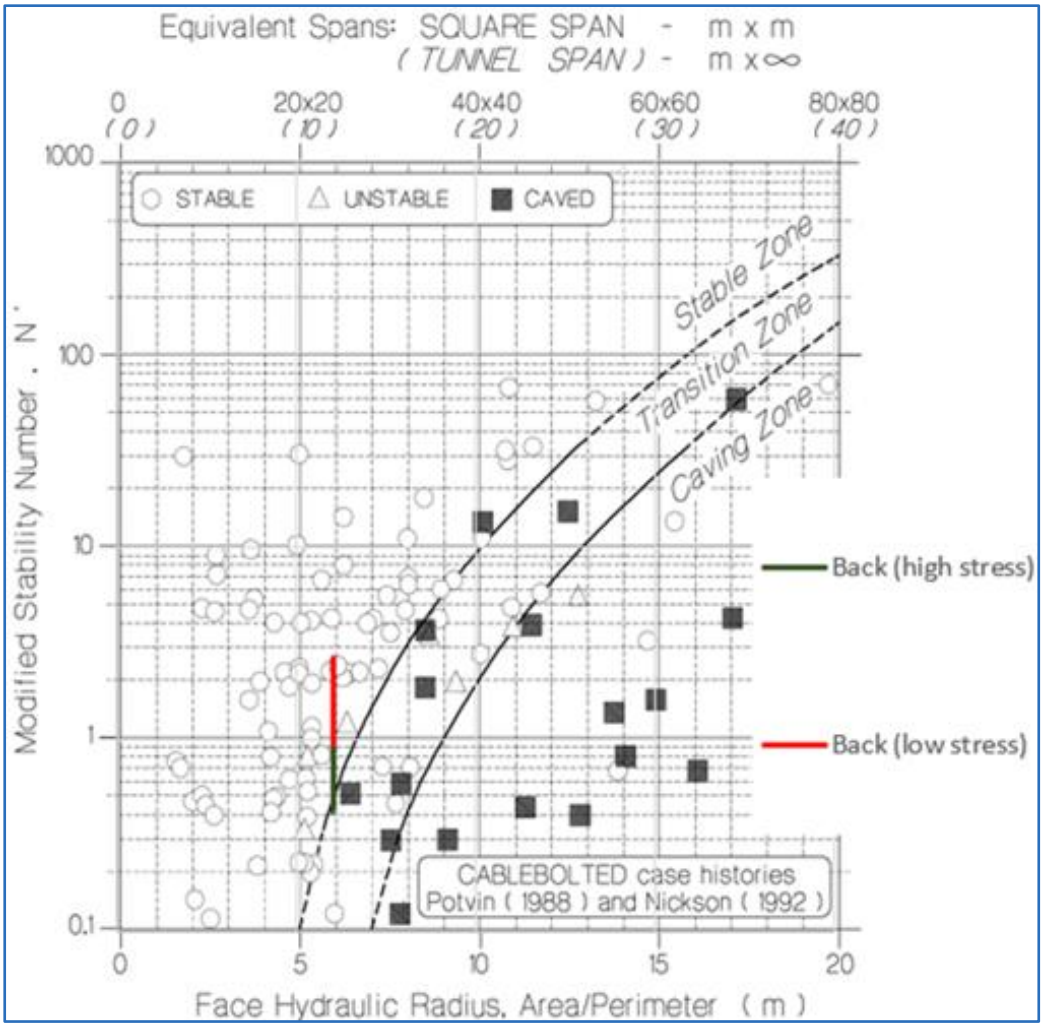
Figure 16.13 and Figure 16.14 present the design cases for the stope back and stope walls on the stope stability charts for unsupported and supported stopes, respectively.

Figure 16.13 Stability Chart for Unsupported Cases



Hutchinson & Diederichs, 1996

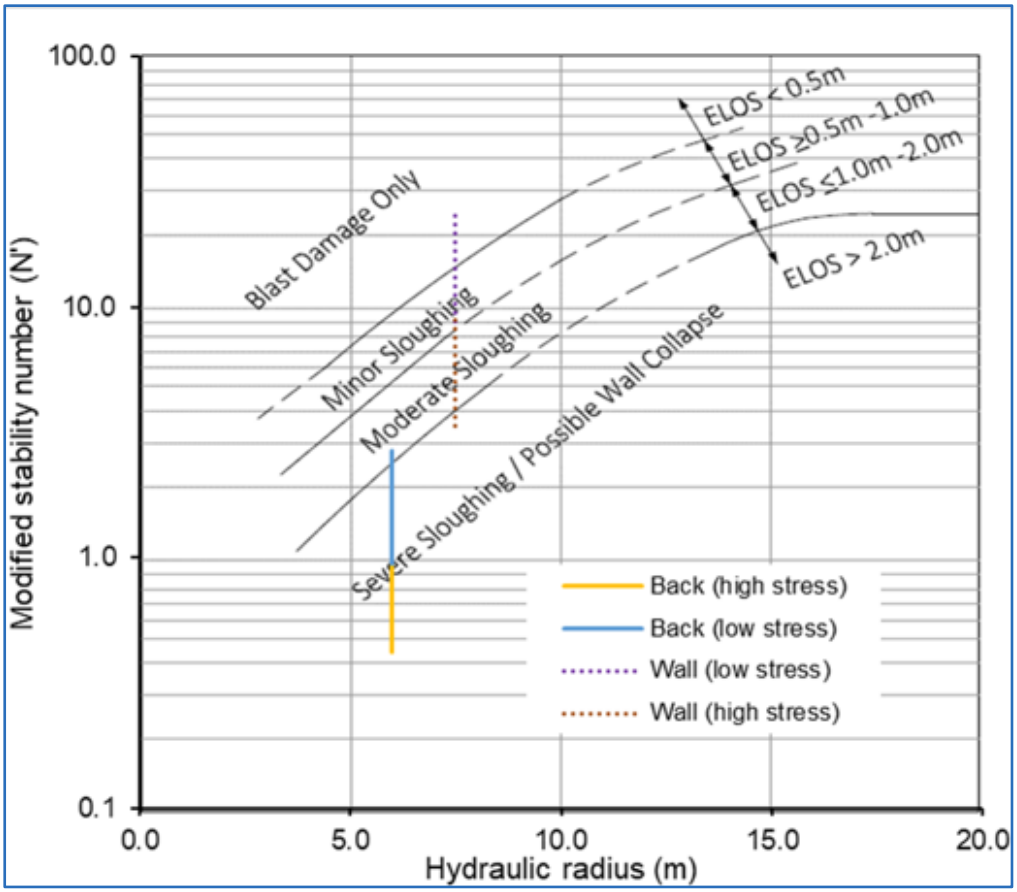
Figure 16.14 Stability Chart for Supported Cases



Hutchinson & Diederichs, 1996

The potential for equivalent linear overbreak and sloughing (ELOS) was estimated by applying the work of Clark and Pakalnis (1997) and Clark (1998). Figure 16.15 indicates little to no sloughing is expected from the stope walls during the earlier stages of mining whilst moderate sloughing is anticipated for all rock mass conditions during the later stages of mining. Severe sloughing can be expected from unsupported stope backs in both low and high stress conditions.

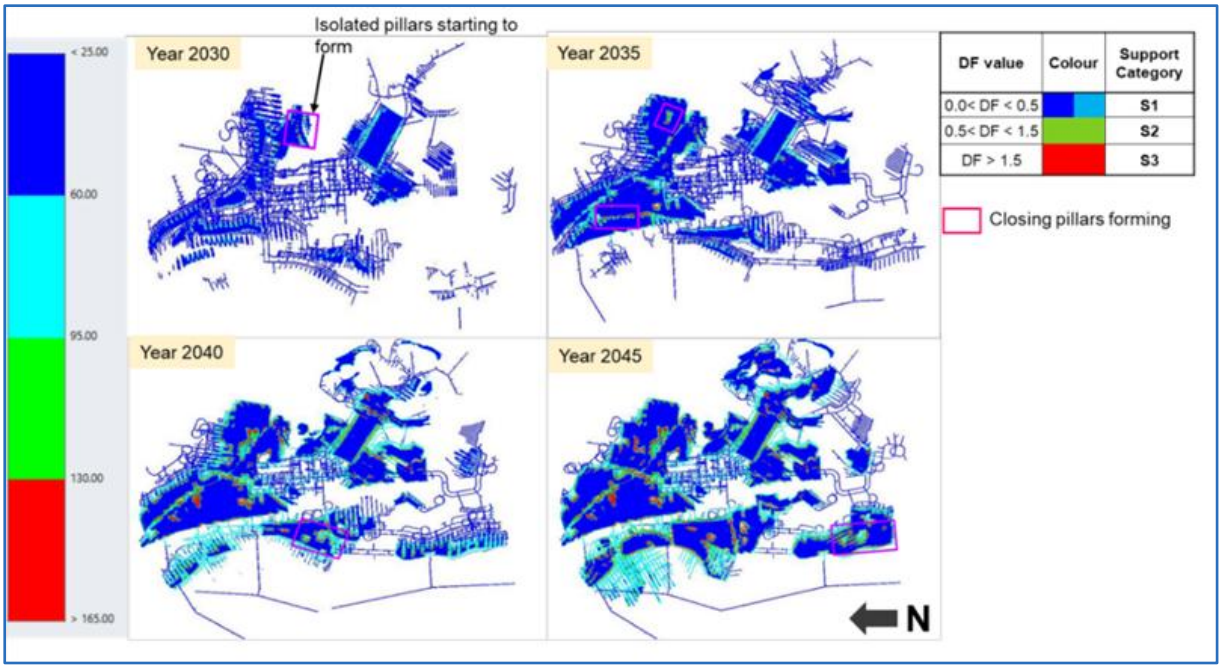
Figure 16.15 Empirical Estimation of Overbreak and Sloughing



The numerical analysis results of the revised mine design indicate that the ramps, main drives and crosscut developments were sited away from the influence of the stoping related stress changes. Most of the access drives, and stope drives do not experience high stress and S1 support is adequate.

High stress concentrations start to occur in the stope access drives and drifts when sill pillars start to form. Stope drives should be mined "just in time", to avoid early deterioration from stress changes. Figure 16.16 shows the stress build-up for the some of the mining periods.

Figure 16.16 Stress Changes on Access Development



The results of the elastic analysis indicated that significant stress concentrations develop during the formation of sill pillars. However, these stress concentrations were not accurately simulated using elastic modelling. To better evaluate rock mass behaviour during the extraction of sill pillars, a conceptual elastoplastic finite element model was constructed. This analysis methodology effectively captures the progression of rock mass damage, which could potentially lead to face bursts and increase personnel exposure during mining operations.

Figure 16.17 shows the stress redistribution during the controlled extraction sequence of the primary stopes on the level below the sill level. When the primary stopes are blasted, the rock mass in between the primary stopes yields, resulting in a stress reduction in the lower portion, but a high stress concentration in the unyielded portion at the top. This high stress concentration is likely to result in rockbursts during developing of the top stope drives and S3 support is therefore required.

Figure 16.17 Level Below Sill Pillar – Stress Redistribution During Mining of Primary Stopes

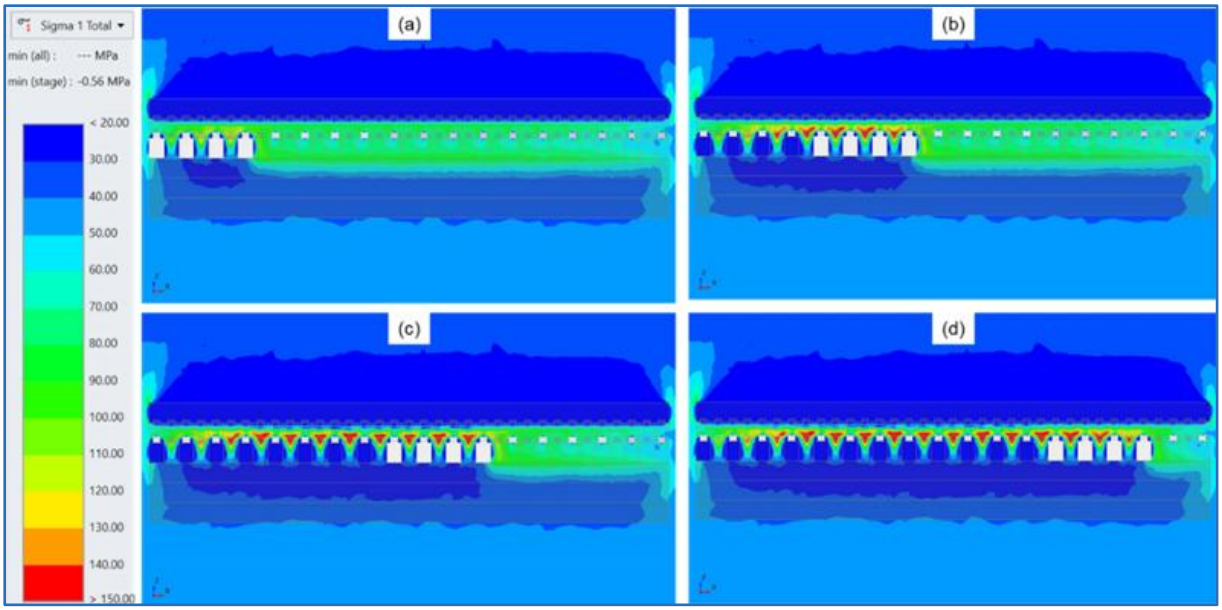
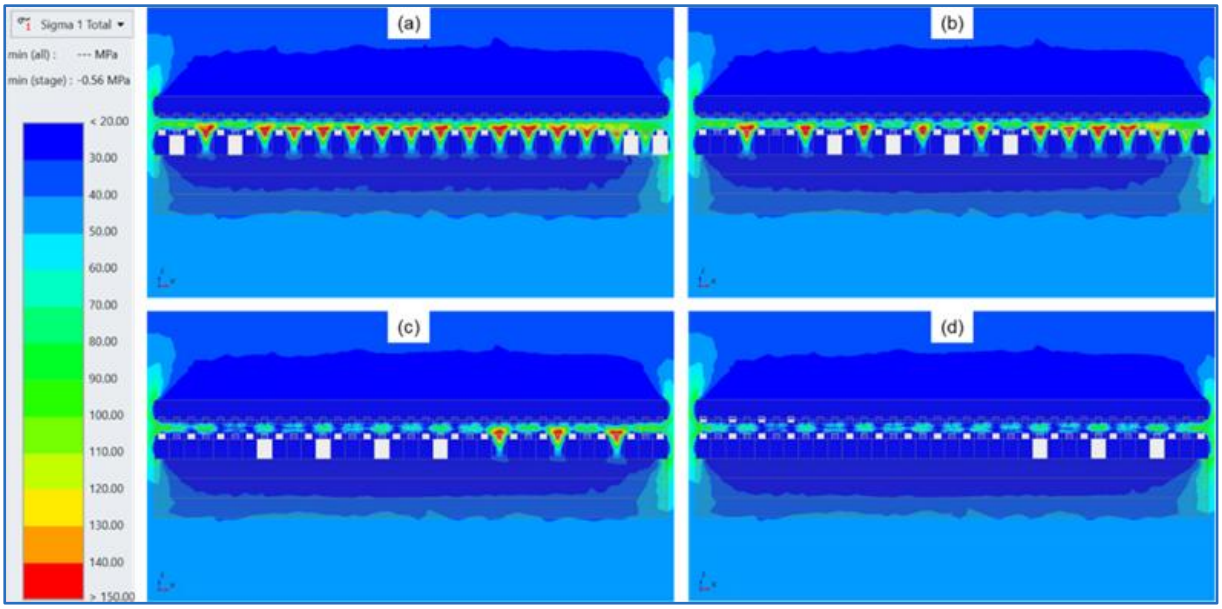


Figure 16.18 shows the stress redistribution during the mining of secondary stopes. It is apparent that the stress in the sill pillar reduces due to yielding of the rock mass. This suggests that strainbursts will be less likely during sill pillar extraction.

Figure 16.18 Level Below Sill Pillar – Stress Redistribution During Mining of Secondary Stopes



The elastoplastic analysis indicates a significant risk of strainbursts and facebursts during the extraction of secondary stope top drifts below the sill pillar. Therefore, installing a seismic system before mining this sub-level is essential. A dense seismic network, capable of recording small seismic events in 3D, is crucial to verify rock mass yielding. Seismic re-entry protocols will be vital for the safe extraction of the sill pillar.

Mining the primary stopes on the sill sub-level before the secondary stopes below should eliminate high stress concentrations and ensure rock mass yielding before developing the secondary stopes' top drive. Changes to the sill pillar extraction sequence are likely once the rock mass response is measured by the seismic network.

For the FS 2024, it is recommended to extract the sill pillars at a reduced rate to allow for sequence adaptation, support installation, and seismic re-entry protocols.

16.1.2.3 Drift and Fill Design

The production stope drifts will be initially developed at 7 m x 6 m width and height during ramp up and will be reduced to 6 m x 5 m width and height in full production phase. The development of primary and secondary drifts will be relatively straightforward, requiring only S1 type support, provided that tight filling of the primary drifts is achieved.

In a large drift and fill mining block, major relaxation of the hanging wall is expected when tertiary drifts are extracted due to the softness of backfill compared to rock. This may cause the jointed rock mass to unravel, requiring more support during tertiary drift mining. Tight filling of primary and secondary drifts is crucial to prevent enlarging the effective span between backfilled drifts during tertiary drift development.

The depth of failure is very low in the primary and secondary drifts, and increase when the tertiary drifts are extracted, due to the increase in span between the stiffer rock. This could potentially lead to block failures and longer support will be required in the tertiary drifts.

Numerical modelling results indicate that the further from the nearest rock pillar, the greater the depth of failure in tertiary drifts due to significant hanging wall relaxation, requiring additional cable bolt support. Mining more than one tertiary drift at a time before backfilling increases failure risk and tensile zones, potentially leading to overall failure. Final drifts should be mined one at a time in a controlled sequence. Notable hanging wall failure is expected when backfill is on both sides of the final drifts.

The primary, secondary, and tertiary sequence is recommended, allowing controlled retreat mining with additional cable anchor support. Final drifts should be mined safely one at a time, with immediate backfill placement after each drift to confine temporary pillars, which are unstable without confinement. Any backlog in backfill placement should be addressed through a risk assessment. Table 16.13 summarises the recommended cable bolt support required for the 6.5 m and 7.5 m wide tertiary drifts.

Table 16.13 Cable Bolt Recommendations

Drift Span (m)	Arch Height (m)	Bolt Length (m)	Bolt Weight (tonnes/m)	Bolts/m	Row spacing	Bolts/row	Bolt spacing (m)
6.5	3.3	3.8	54	1.5	2.0	3.0	1.6
7.5	3.8	4.3	72	1.9	2.0	4.0	1.5

16.1.3 Management of Geological Structures

The Tshukudu and Nkwe faults are major steep east and north-east dipping geological structures identified in the structural model. Additionally, several steep and flat dipping structures were identified. Five geotechnical structural domains were identified, and the joint sets were characterised.

Table 16.14 describes the geological structures, potential hazards, and the appropriate management strategy, which need to be taken into consideration during mine design. However, each stope should be geotechnically assessed and these strategies may be refined further during further studies and/or when additional information becomes available as mining progresses.

Table 16.14 Summary of Potential Risks and Potential Strategies associated with the Geological Structures

Structure	Characteristics	Hazard	Strategy
Tshukudu fault	Dips at 50° to 70° E. Displacement 60 m to 80 m. Affected area 5 m to 30 m with occasional talc joints and an up to 8 m wide zone of intense jointing with friable carbonate with chlorite/talc infill.	Hazardous ground conditions – FOGs and poor development rates. Increased water inflow.	Plan geotechnical loss (15 m on each side of the Tshukudu wireframe). Reduce number of excavations traversing through the fault. Significantly reduce the advance rate when developing through the features. Excavation oriented perpendicular to the fault. Ground consolidation and water sealing ahead of the tunnel. Use of resin injection and self-drilling anchors, wet Crete steel arches with void filling to be considered.
Nkwe fault	Dips at 55°NE. Displacement 40 m. Affected zone can be up to 24 m, with occasional infilled joints. Many structures have been healed.	Potential seismicity. Hazardous ground conditions – FOGs and poor development rates	Plan geotechnical loss (10 m on each side of the Nkwe wireframe). Use of cover drilling to characterise the feature prior to development. Seismic monitoring network – geophones to be installed near the feature. It must be considered during seismic network design. Cover drilling to characterise the feature, prior to development. Reduce the advance rate when developing through the features. Geotechnical inspection for input into the design.

Structure	Characteristics	Hazard	Strategy
Steep dipping structures	Nyati (E dipping) not well characterised. Noko, Lengau, Tau, Mabitso, Fisi faults. Dip at 60° to 75° NE.	Wall instability. Overbreak. Production delays. Equipment damage.	Reduce unfilled stope length in the vicinity of modelled faults and poor ground in the block model. Stope geotechnical assessments and reconciliation during extraction. Avoid placing large excavations in the vicinity of these structures.
J1	Steep NE dipping joint, 58% clay infill.		Reduce unfilled stope length in the vicinity of modelled faults and poor ground in the block model. Stope geotechnical assessment and reconciliation during extraction.
J2	Steep E dipping joint, 59% clay infill. North, central and west domains.		
J4	Steep W dipping joint, 51% clay infill. North and east domains.		
J5	Steep SE dipping joint, 68% clay infill. In south-east domain.		
J6	Vertical joint in south-east domain only, wide spacing, insufficient information characterised.	Limited influence.	Reduce unfilled stope length in the vicinity of modelled faults and poor ground in the block model. Stope geotechnical assessment and reconciliation during extraction.
LAFs with associated granitic veins and shear zones	LGV 10 to 70. Variable characteristics.	Large FOGs due to the formation of wedges. Back instability associated with wedges, overbreak, production delays. Equipment damage.	Avoid placing large excavations in these areas. Avoid placing 3-way and 4-way intersections in these structures. Where it is not possible to avoid, additional support (cable anchors) will be required, and the design will be site specific. Where it is not possible to avoid placement of ore passes, abrasion resistant support should be considered when intersecting these structures. Surface support such as wetcrete and/or, mesh will be required. Stope and drift back support may be required. Stope geotechnical assessments during mining. Mine the T2U chrome if located in close proximity to the planned stope. Avoid leaving brows near the T2U in the hanging wall
Chrome Stringers (T2U)	Sub-horizontal, low tensile strength. Top of T2 and random single or multiple planes.		
J3	Flat NE dipping joint set, 63% clay infill. Central and south-east domains.		

16.1.4 Support Requirements

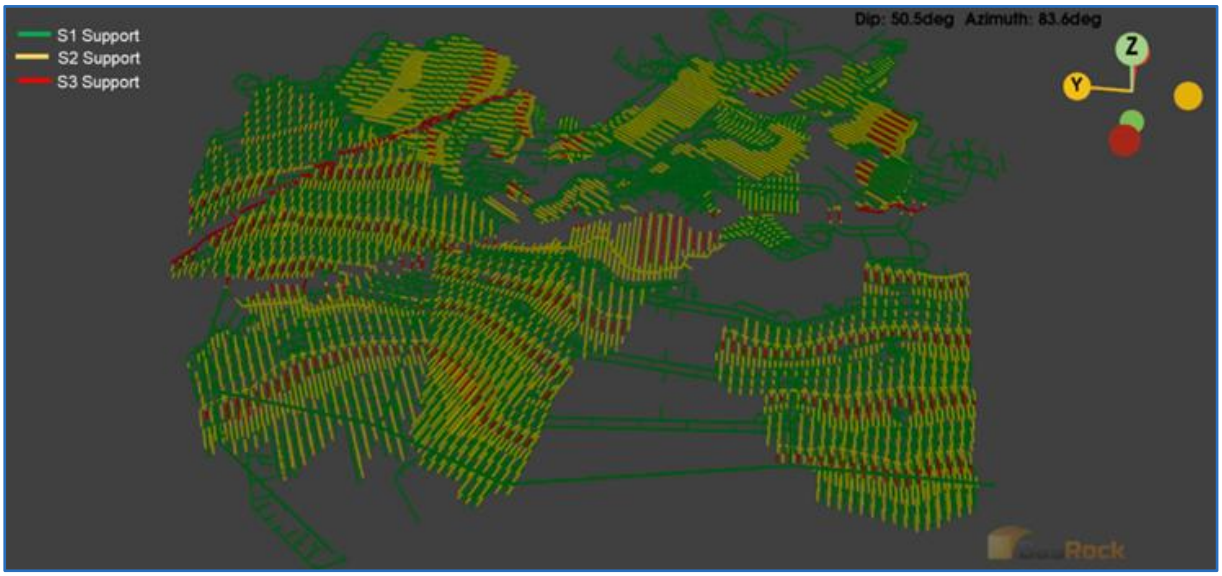
Support recommendations were based on a combination of rock mass ratings, stress damage potential and major geological structures. The potential stress induced damage on the main access drives and stope drives for different critical long-term access drives and drifts was used to determine support requirements, taking into consideration the potential of instability when identified major structures are intersected.

During the early phases of mining, stress damage will be very limited, and the standard support will suffice. As the overall extraction and mining depth increases, stress concentrations will increase, and significant stress damage and seismicity can be anticipated, which will need to be managed.

Figure 16.19 shows the recommended support classes for the different lateral development in the design to ensure stability during their operational life. On levels away from the sill level, the stope drifts for primary stopes will require S1 support whilst the secondary stope drifts should be supported with S2. On the level below the sill level all stopes drifts (for both primary and secondary stopes) will require S2 support. The stope access drives for these levels will be supported with S1 support.

On the sill level, the stope access drives and the stope drifts for primary stopes will require S2 support and the stope drifts for secondary stopes will have S3 support.

Figure 16.19 FS 2024 Base Design Support Recommendations



The support recommended for stope drives excavated in backfill comprises of 4.5 m cable anchors in the back to tie the block of fill into solid rock in the drive and 2.4 m split set on 1.0 m x 1.0 m spacing in the back and sidewalls up to the grade line. Additionally, 1.8 m x 3.0 m #4 galvanized welded wire mesh, blast resistant should be installed and 50 mm thick in-cycle fibre-reinforced shotcrete sprayed floor to floor.

A mine-wide seismic system will be installed within the first five years of reef mining to monitor the seismic response to mining. The support requirements cater for different stress regimes.

The proposed support standards and specifications for Platreef are listed in Table 16.15 and Table 16.16.

Table 16.15 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 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. Required for main level haulages and ramps (long-term).
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. Not in main level haulages and ramps (long-term).
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).

Excavation Type	DF (From Numerical Modelling)	Q Rating (from Rock Mass Classification)	Support Standard
Tunnel support for high stress, dynamic conditions (S2B)	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 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	Primary support installed during development. Secondary support: 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 (S1A)	DF <0.5 m	Q>10	Minimum of 2.4 m long resin bolts, installed on 1.8 m x 1.8 m grid, down to 1.5 m from the footwall + backfill support.
Production drifts (S1B)	DF <0.5 m	4<Q<10	Minimum of 2.4 m long resin bolts, installed on 1.8 m x 1.8 m grid with mesh or 50 mm shotcrete, down to 1.5 m from the footwall + backfill support.
Production drifts (S2)	0.5<DF<1.5m	1<Q<4	Minimum of 3.0 m long resin bolts, installed on 1.2 m x 1.2 m grid with mesh or 50 mm shotcrete, down to 1.5 m from the footwall + backfill support.
Production drifts (S3)	DF >1.5 m	Q<1	Minimum of 3.0 m long resin bolts, installed on 1.0 m x 1.0 m grid with mesh or 50 mm shotcrete, down to 1.5 m from the footwall + backfill support.
Support in backfill walls		-	Minimum of 3.6 m split sets (SS 46) in the back to tie block of fill into solid rock and 2.4 m long split sets (SS 46) in the walls to the grade line on 1.0 m x 1.0 m spacing, 1.8 m x 3.0 m galvanized welded wire mesh, + 50 mm in-cycle fibre-reinforced shotcrete sprayed floor to floor.

Excavation Type	DF (From Numerical Modelling)	Q Rating (from Rock Mass Classification)	Support Standard
Support for hazardous geological structures (shear zone/extensive area)		-	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.
Support for LAF and chrome stringers (stope drives & development)			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 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 SDAs every 1.5 m (spacing 0.5–1.0 m). Resin injection through the SDAs for consolidation. Steel arches spaced 0.5–1.8 m apart depending on rock quality (expected 1.0 m), void filling between arches and tunnel walls. Resin consolidation and reinforcement may also be required in the face.

Excavation Type	DF (From Numerical Modelling)	Q Rating (from Rock Mass Classification)	Support Standard
Large excavation support	-	-	Primary support + pre-tensioned, grouted, cable bolts (minimum length = half excavation span), maximum spacing = 0.5 x length.
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. Installed within 1.0 m of the brow.
Critical ore passes		-	3.0 m long pre-tensioned, grouted, cable anchors installed in a 2.0 m x 2.0 m pattern.
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 16.16 Support Type Minimum 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 46)	Minimum outer diameter 45.5–47 mm; minimum 420 MPa (yield stress) steel, minimum steel thickness 3.2 mm, hole size 43–45 mm.
Split set (SS-39)	Minimum outer diameter 35 mm to 38 mm, minimum 420 MPa (yield stress) black Supraform steel, minimum steel thickness 2.3 mm, hole size 35 mm to 38 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	Galvanised/weld mesh, minimum 5 mm gauge, maximum 100 mm aperture, blast resistant, compatible with mobile bolter.
Shotcrete	Minimum 25 MPa (28 day strength) fibre reinforced shotcrete.
Osro straps	300 mm wide straps with five 10mm rods, minimum 500 MPa black steel.
Capsule resin	Two component urethane silicate resin capsules. Fast (<60s) 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 to 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 Monitoring and Instrumentation

Monitoring is an essential part of the geotechnical mine design and can be used to evaluate the predicted behaviour against actual performance. The monitoring data can be used to optimise the designs. The following sections briefly describes the monitoring.

16.1.5.1 Excavation Monitoring

The following monitoring should be considered:

- Ongoing geotechnical mapping should take place at regular intervals in the development to verify the rock mass conditions determined in the FS. This will also allow for the identification of localized weak zones such as LAF and potentially unstable wedges, which should be supported. The joint data should be used to optimise empirical stope designs.
- Elastic modelling of mined and planned stopes as well as development tunnels should be carried out on a regular basis to determine where high stress concentrations are located. This will allow for updates to the planned design or for increased support to be applied in the affected areas.
- Stope drives must be mapped to determine the rock mass conditions and geological structures that may influence stability of the stopes.
- Stope reconciliation using lidar cavity monitoring must be carried out routinely to evaluate overbreak and underbreak. This information should be used together with the geotechnical mapping and stope drilling information top optimise the performance of stopes.
- Deformation monitoring of stress damaged stope drives and vulnerable access drives located in closing pillars and on the sill level should be carried out using lidar scanning.

16.1.5.2 Backfill Monitoring

During the implementation stage of the Project monitoring of the backfill performance is required. Monitoring will be required during the first phase of the Project and the data obtained will be used to improve the designs. The following will need to be monitored:

- Backfill earth pressure on the barricade using Total Earth Pressure Cells (TEPC) to measure load generated on the barricade during filling.
- Pore water pressure using vibrating wire piezometers to measure the water pressure at the barricade during filling.
- Samples must be collected during batching of backfill. Early strength tests will be required, if early extraction (<28 days) of secondary stopes becomes necessary for any reason. In-situ coring of backfill in stopes will be required on occasion to confirm that the required backfill strengths are being achieved

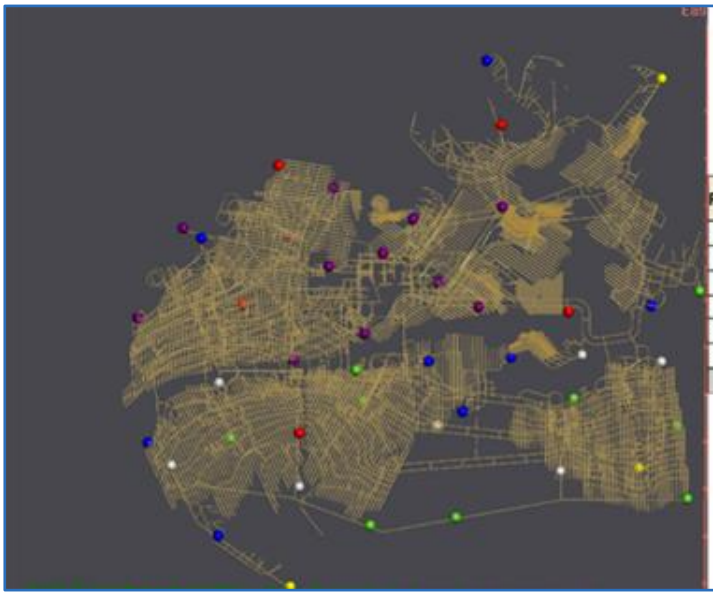
16.1.5.3 Seismic Monitoring

During the early stages of mining very little stress damage and seismicity is anticipated. Temporary sill pillars will be formed between main levels, which result in high stress concentrations and seismicity. A seismic monitoring system will be required to assess seismic hazard for the purpose of managing seismic risk.

The seismic system should be sensitive and accurate enough to monitor the progression of rock mass damage, which will show that the highly stressed temporary pillars (secondary stopes) are completely fractured, which means that strainbursting will not occur during the development of secondary stope drives. The location accuracy should be reliable enough to implement reliable re-entry protocols after stope blasts and large seismic events.

A seismic system with a total of 45 sensors have been designed for Platreef (Figure 16.20). It comprises a combination of uniaxial and triaxial sensors sited in the foot wall and hanging wall. The seismic system should enable good sensitivity and location accuracy in 3D throughout the mining footprint. The design ensures that six sensors will always be within 500 m of a seismic event, which is benchmarked on international mine seismic networks. The expansion of the network is scheduled to ensure that mining areas are well covered at least two years before the stress concentrations associated with extraction of sill pillars and closing pillars manifest.

Figure 16.20 Proposed Seismic Monitoring System



Period	No. of sensors required	H/W Sensors	F/W Sensors	Colour
Aug-30	13	6	7	Purple
Jan-33	5	3	2	Red
Jul-37	7	3	3	White
Dec-40	8	3	6	Green
Jun-45	9	5	3	Blue
Jan-48	3	1	2	Yellow
Total	45	21	23	

16.1.6 Geotechnical Risks

The purpose of the risk register is to record identified risks, their severity, and the actions taken to mitigate the identified risks. Potential hazards and risks are described and strategies to manage these risks during the implementation phase are recommended in Table 16.17.

Table 16.17 Geotechnical Risks

No.	Geotechnical Hazard	Description	Consequence(s)	Control(s)
1	Seismic potential associated with Nkwe fault and other major structures.	Underground mining operations at depth around the Nkwe fault zone may induce seismic activity leading to falls of ground, damage to equipment and injuries or fatalities.	Seismic related falls of ground, injuries or fatalities, damage to equipment and infrastructure.	Adhere to appropriate mining sequence. Backfill placement with required strength implemented.
2	Areas of high stress concentrations in development and stoping.	The mining depth, access ways, LHOS and drift profiles outlined in the FS mine design could induce high stress zones in the vicinity of these excavations, resulting in potential damage to excavations and ground support.	Local high stress damage of the ground support and excavations leading to poor ground conditions, equipment damage, production losses, lost time injuries.	Numerical modelling assessment of stresses with has been conducted. Use of the appropriate support (S3) dedicated for high stress underground environment. Ground monitoring such as underground inspections, or primary stress measurements at deeper production levels.
3	Uncontrolled unravelling (running) due to a combination of weak infill, a highly fractured, wide fault influence zone in Tshukudu fault.	Fault is characterised by sections of incompetent/friable carbonate and chlorite/talc infill of variable degree. Underground mining operations at will likely encounter potential water inflow issues.	Uncontrolled unravelling (running) leading to poor ground conditions, damage to equipment, production losses.	Significantly reduce the advance rate when developing through. Geotechnical drilling ahead of the tunnel face to confirm the rock mass conditions. Ground consolidation and water sealing ahead of the tunnel. Use of resin injection and self-drilling anchors, shotcrete arches.

No.	Geotechnical Hazard	Description	Consequence(s)	Control(s)
4	Undercutting backfilled stopes.	The current FS layout has planned extraction in various periods that will undermine the backfilled drift and fill and longhole stopes.	Instability of backfill sills causing equipment damage, production losses, lost time injuries or fatalities.	Tight filling will be required in the primary stopes to maximise recovery of the secondary stopes. The extraction sequence should be to minimise cold joints during backfilling. For LHOS, fill the entire stope with addition binder and 2.5MPa sill strength will be required
5	Adverse ground conditions associated with prominent geological features (faults, LAFs).	The Project area is traversed by faults, LAF's and weaker chromitite partings which have the potential to create adverse ground conditions.	Key block creation, falls of ground, injuries or fatalities, damage to equipment and infrastructure.	Support design caters for the features. Strategies have been recommended to minimise effects of these features.
6	Tight filling in the drift and fill sections has not been achieved.	Instances where imperfect tight filling has occurred of if the fill is completely ineffective.	Onset of major failure in the stope hanging wall after mining of the second lift. Problems can be anticipated in the drift back. It will not be possible to extract tertiary drifts.	Intense support comprising 6 m long anchors may need to be installed. Optimise the backfill placement method in subsequent drifts.

16.2 Underground Mining

16.2.1 Introduction

The Platreef 2025 FS includes two main phases. Phase 1 is defined as establishing Shaft 1 remaining infrastructure, development and other infrastructure required to sustain 770 ktpa. Phase 2 commences just after equipping Shaft 3 when production begins to ramp up to 4.07 Mtpa and Plant 2 is commissioned.

Key steps involved in preparing the Platreef 2025 FS are as follows:

- Shaft 1 changeover completed for permanent hoisting and start development on 750 m, 850 m and 950 m Levels. The hoisting capacity is 60–65ktpm.
- Initial development focus on establishing Shaft 3 underground infrastructure that includes tips on every level, ore passes, crusher excavation, conveyor to Shaft 3 loading arrangement and access to Shaft 4 raiseboring position. Other priorities include service excavations like emulsion bays, dams, batching facilities, workshops and laydown areas.
- Secondary development priority is focussing on getting access to the nearest, highest grade stopes on 750 m and 850 m Levels.
- Shaft 3 is planned to be commissioned in February 2026 with a hoisting capacity of 4 Mtpa. Development focus after this is ramping up development to open up sufficient stopes to maintain steady state to fill both concentrators. Shaft 1 is utilised for logistics only during this time from February 2026 to July 2028. 770 ktpa concentrator on site to be commissioned Q4'25 and the second concentrator with a capacity of 3.3 Mtpa to be commissioned Q4'27 (or combined capacity of 4.07 Mtpa).
- Shaft 2 personnel and material shaft is required to do logistics and dedicate Shaft 1 for hoisting by July 2028.
- Assumes dry stacked tailings dam (for on-site concentrator).

The Platreef 2025 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 Review of the Mine Design

For the Platreef 2025 FS, changes were made to the mine design to meet the requirements for the Development Plan including development to establish Shaft 3 as a hoisting shaft and not a ventilation shaft as per 2022 FS. Also, design changes were made to streamline rock handling from lower levels to Shaft 3 via a crushing and conveyor system and dedicated tramming truck haulages on each level to separate logistics from ground handling. The major changes from 2022 FS to 2025 FS are shown below in Table 16.18.

Table 16.18 2022 FS vs 2025 FS

Item	2022 FS	2025 FS
Ore Mining Rate Maximum Hoisting and ROM	5.17 Mtpa 6.0 Mtpa	4.07 Mtpa 4.7 (hoisting) & 5.25 Mtpa (ROM)
Shaft 3	Ventilation return shaft	Equipped to hoist 4 Mtpa
Shaft 2	Major shaft – 6 Mtpa	Commissioned as logistics shaft in H2 2028.
Ore Handling	Lowest tip 950 m Level	Lowest tip 1,050 m Level (conveyor system from 1,100 m Level to 950 m Level).
Crushers	2 x crushing stations	5 x crushing stations
Development Sizes	5 x 5 m	6 x 5.5 m
Development Jumbos	Function: Face drilling Bolter utilised for bolts Rates: Dev.: 225 m/mth/jumbo Drift and fill: 25 000 t/mth	Function: Barring Support: Bolts & mesh (bolters removed) Face drilling Rates: Dev.: 60-180m/mth ramp up Drift and fill: 20 000 t/mth
Loading and Hauling	18 t LHD and 54 t Truck Trucks used on every level	14 t LHD and 42 t Truck (<2029) 18 t LHD and 54 t Truck (>2029) (Shaft 1 size limitation – 65 t planned for Phase 3) Ore passes on inter-levels to main haulage levels. Limited trucks on inter-levels for development only. Trucks used to haul tonnes from ore passes to main level tips with minimal hauling in ramps.
Hauling	All tonnes hauled back to main shaft tips from all levels.	Trucks hauling limited to main levels and development on inter-levels. Use ore passes for ore tonnes on each level that reports on main levels.
LH Stopes	Bottom-up mining No easer for slot drilling All down-holes	Levels top-down Between levels bottom-up Easier for slot blasting incl. Initial stopes using up-holes but LOM stopes using down-holes.
Target Initial Stopes	High grade drift and fill stopes far away from shaft complex.	Medium grade LH stopes close to shaft.
UG Equipment Quantities	Central workshops on main levels.	De-centralised approach with drill rig being maintained in satellite workshops close to working areas.

The Platreef 2025 FS includes two main phases. Phase 1 is defined as establishing Shaft 1 remaining infrastructure, development and other infrastructure required to sustain 770 ktpa. Phase 2 commences just after equipping Shaft 3 when production begins to ramp up to 4.07 Mtpa and Plant 2 is commissioned. The Platreef Project is designed based on longhole stoping and drift and fill mining methods.

The Platreef 2025 FS evaluates a development of Platreef, with an initial 770 ktpa underground mine and a 770 ktpa capacity concentrator, targeting mining areas close to Shaft 1and Shaft 3. First concentrate production for this option is targeted in 2025, with the hoisting Shaft 3 commissioned in Q1'26 and personnel and material Shaft 2 commissioned in Q3'28. The second concentrator with a 3.3 Mtpa capacity enabling a combined process capacity of 4.07 Mtpa will be commissioned in Q4'27.

During Phase 1 of the Platreef 2025 FS, the main access to the mine will be via a 986.3 m deep, 7.25 m diameter ventilation shaft (Shaft 1). Development has commenced around Shaft 1 via the principal access/haulage levels (the 750 m, 850 m, and 950 m). The initial focus will be to establish Shaft 3 underground infrastructure that includes tips, ore passes, crusher and conveyance system from different levels to Shaft 3. The secondary priority will be to access the initial stopes on 750 m and 850 m Levels. Shaft 3 will enable the hoisting capacity to increase from 770 ktpa to 4.7 Mtpa. Shaft 1 will also be a ventilation return shaft with Shaft 1 serving as an intake shaft. The secondary focus will be to access the first stopes on 750 m and 850 m Levels respectively.

Figure 16.21, Figure 16.22 and Figure 16.23 show the plan view of the initial development on three main levels of 750 m, 850 m, and 950 m.

Figure 16.21 Platreef 2025 FS – Initial (Pre-Production) Development on 750 m Level

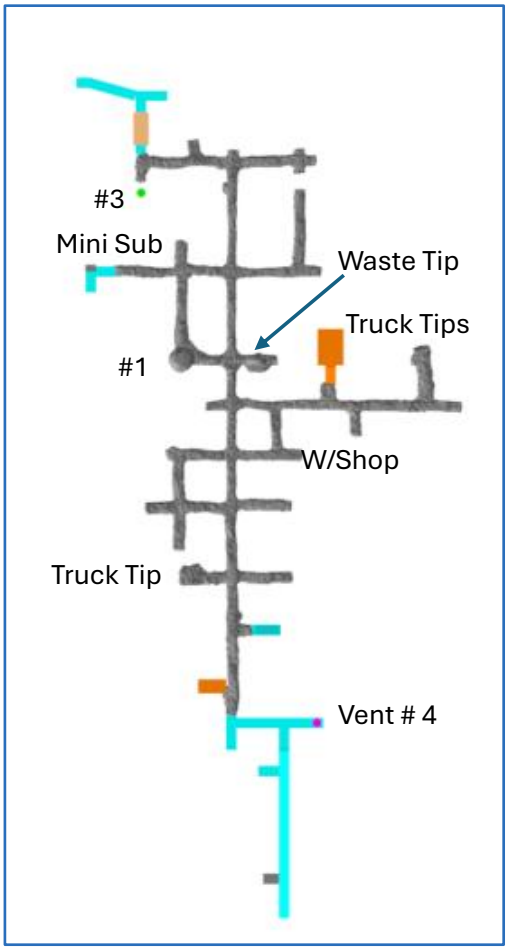


Figure 16.22 Platreef 2025 FS – Initial (Pre-Production) Development on 850 m Level

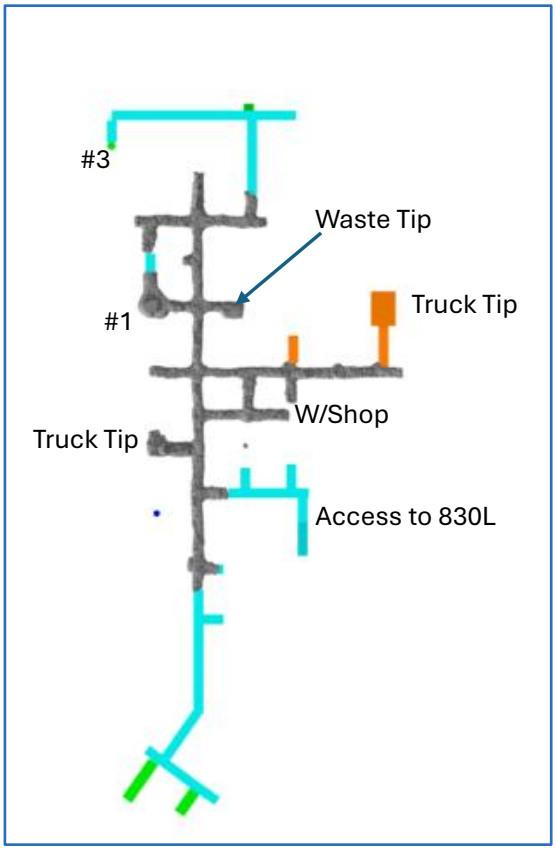
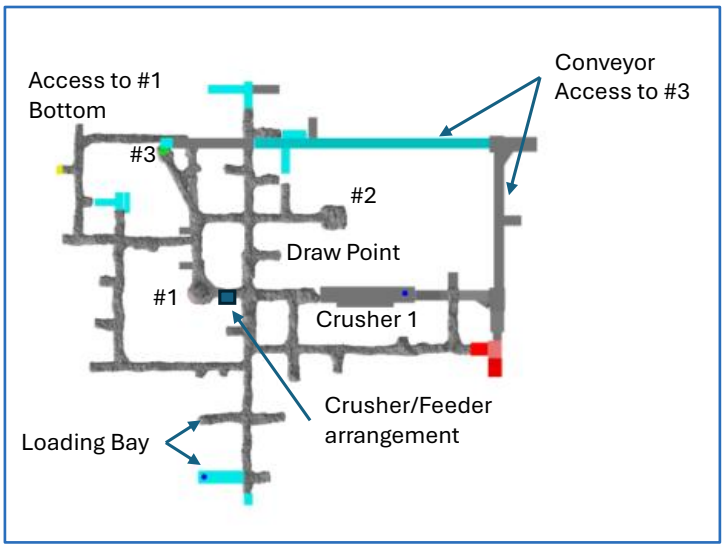


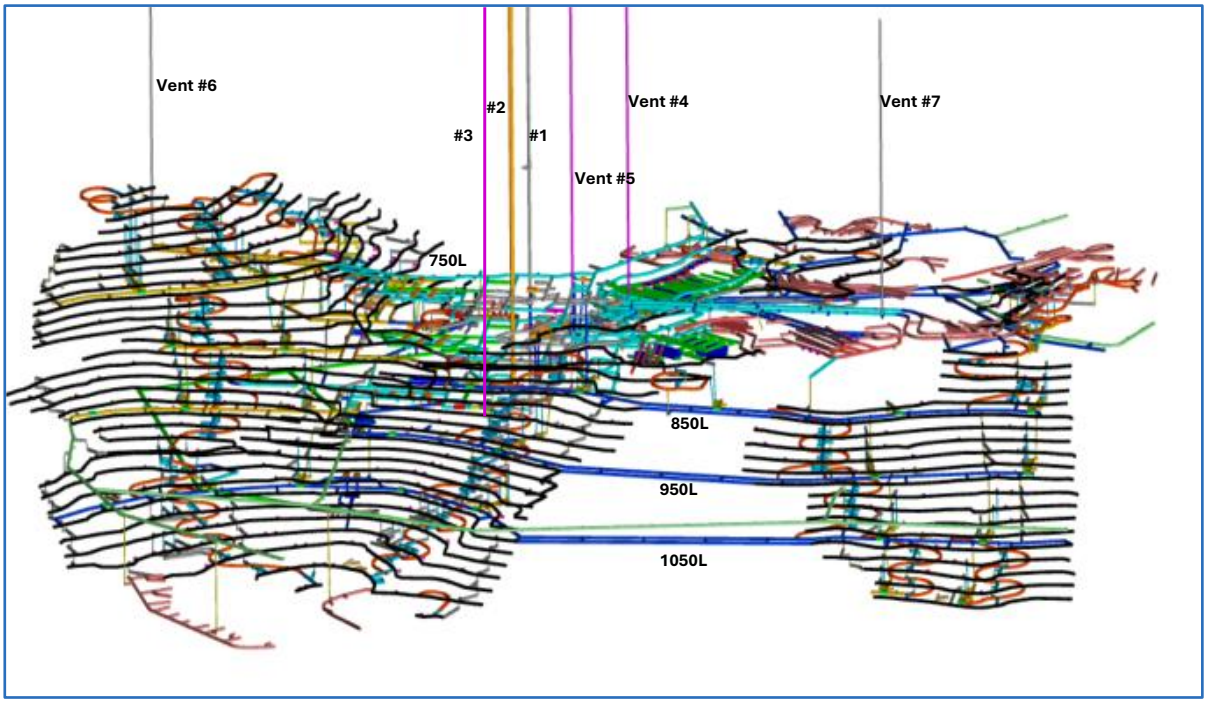
Figure 16.23 Platreef 2025 FS – Initial (Pre-Production) Development on 950 m Level



In Phase 2, primary access to the mine will be by a 996 m deep, 10 m diameter personnel and material (logistics) shaft (Shaft 2) and Shaft 1 and Shaft 3 will remain as secondary and tertiary accesses to the mine. During mine production in Phase 1, Shaft 1 will serve as a ventilation intake too. In Phase 2, Shafts 1, 2 and 4 will also serve as ventilation intakes while Shafts 3, 5, 6 and 7 will be ventilation exhaust raises. Shafts 3-7 will be raisebored shafts and reamed to 5.1mØ.

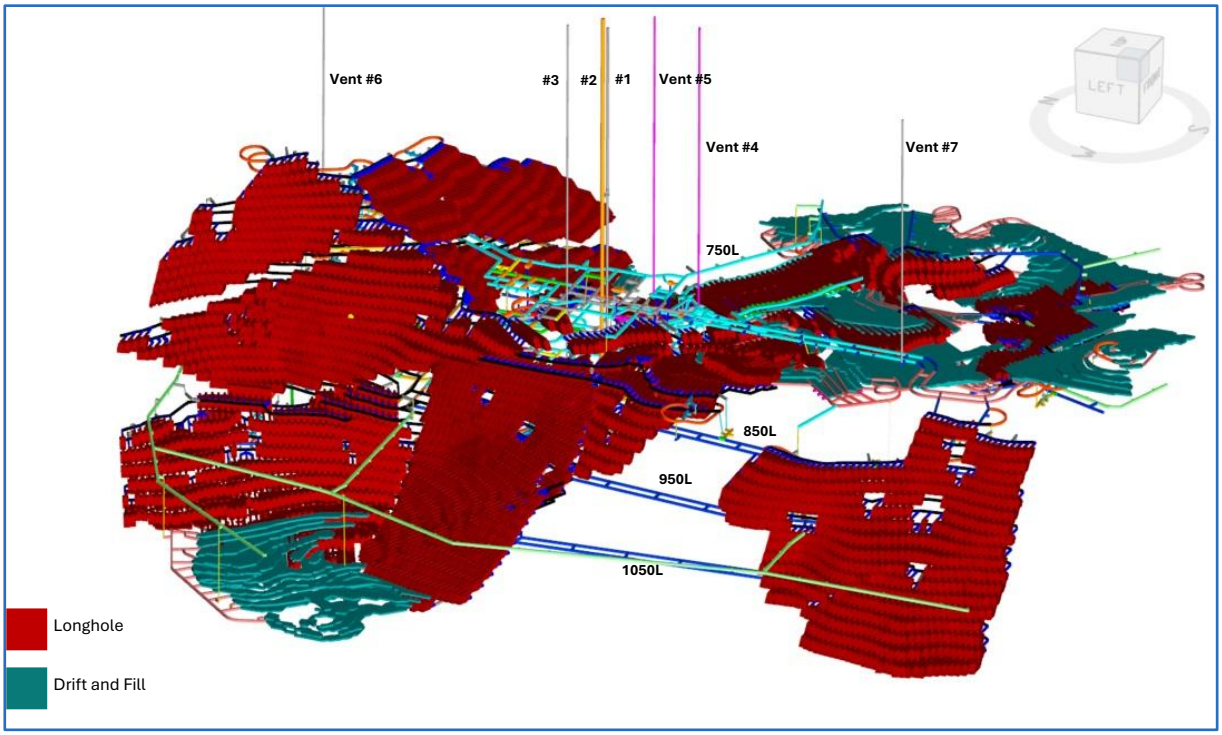
The shafts complex will connect to 750 m, 850 m and 950 m Levels while four main access levels will be established as primary haulage levels. These are the 750 m, 850 m, 950 m and m haulage levels. Figure 16.24 shows the proposed shaft and raise locations and the main access levels in an elevated view (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 16.24 Mine Access Layout



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 16.25 is an isometric view of the mining areas by method (looking north-east).

Figure 16.25 Elevated View of Mining Areas by Method



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 longhole stopes will cause dilution of up to 25% for a single cut and 50% for a double cut.
- 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.

16.2.3 Mine Design Parameters

The following mine design criteria were used for the Platreef 2025 FS.

16.2.3.1 Mine Planning

- Lateral development has a maximum grade will be 9°. Ramps will level off at sublevels to reduce risk of rollovers. 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 will have a maximum raisebore diameter is 5.1 m. Ore passes will be spaced 250 m apart on the levels. Ventilation passes will be spaced 400–450 m apart on levels.
  - 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. A primary-secondary stoping sequence, without adjacent secondary stopes being mined simultaneously. 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.
  - 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 6 m to reduce the length of ground support in the back and walls to 2.4 m. 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.

16.2.4 Mining Method Selection

Sublevel blasthole (longhole) stoping with cemented paste fill, supplemented with development waste rock in secondary stopes, 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 have been considered for the Platreef 2025 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. 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.

16.2.5 Mining Shapes Design

The following mining methods were used in the Platreef 2025 FS. All methods use cemented paste fill supplemented with a development waste rock to fill secondary open stopes.

- Longhole stoping – For ore zones with vertical thicknesses exceeding 15 m, longhole stoping methods include the following:
- Transverse longhole stoping,
- Longitudinal longhole stoping.
- Drift and fill mining (mechanised cut and fill) – For thinner portions of the ore zones (less than 10 m thick).

16.2.5.1 Longhole Stoping

The primary mining method selected for the Platreef 2025 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 2025 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

The primary mining method selected for Platreef is transverse longhole stoping with backfill. Transverse longhole stoping extracts ore in blocks oriented perpendicular to the strike of the orebody.

The longhole stope designs assume a stope design width (level tunnel spacing) of 15 m for all transverse longhole stopes. The stope lengths vary depending on the thickness of the ore envelope from a minimum of 15 m to a maximum of 60 m, as governed by the geotechnical hydraulic radius requirement. The drilling level at the top of the stope and the mucking level is developed at 6 m W x 5 m H. The drives are however developed to 7 m W x 6 m H in the initial 3 years of the ramp up for additional ore tonnes from development.

The stopes have been designed with a minimum hanging and footwall angle of 55° for practical drilling and acceptable dilution. The down-hole sequence will enhance blasting control and hanging wall stability (top drive's tunnel support will assist in keeping the stope hanging wall stable and limit overbreak). There will be limited up-hole rings during the initial phases of the ramp up to assist with longhole stope availability.

Transverse longhole stoping is a bottom-up stoping method with the lowest stopes on the main levels being mined first. In the primary stopes, the upper drilling drift and lower mucking drift will be developed perpendicular to the strike of the ore body.

After the stope has been retreated, a bulkhead will be constructed at the lower mucking drift, and the stope backfilled with paste fill from the top drilling drive. Once the paste fill is poured, it will require the fill to cure prior to blasting against the backfill. The primary-secondary sequence entails mining two primary stopes on top of each other and paste filled before the first secondary stope is mined.

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 drift and a mucking drift. These development 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.

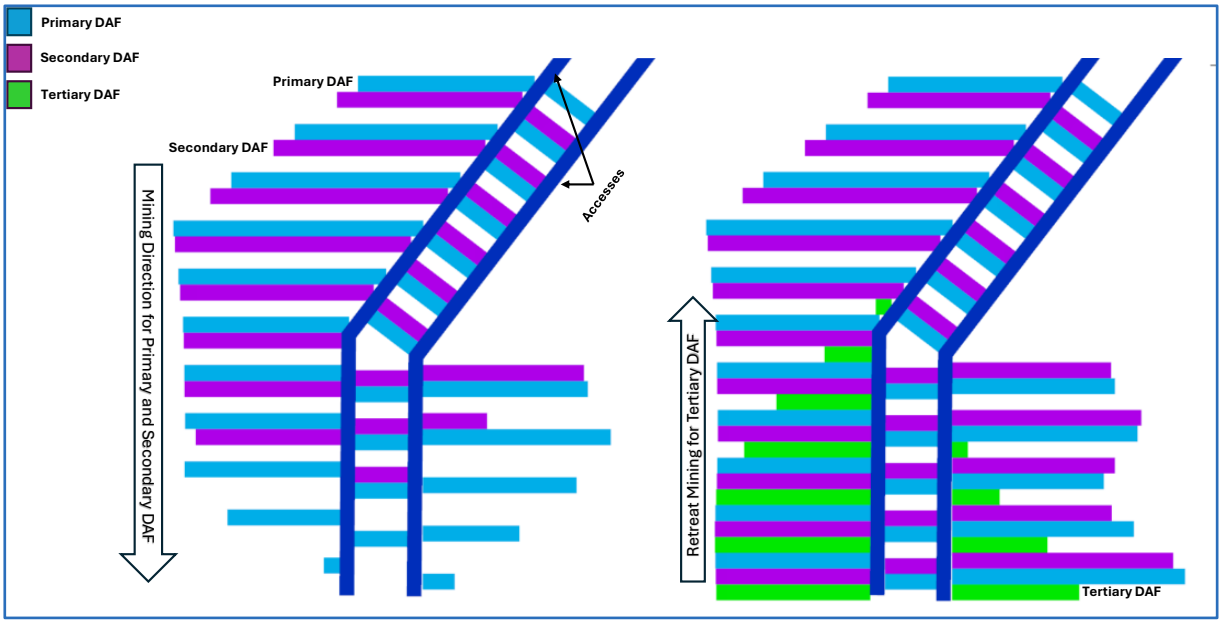
This method requires the drifts be driven to a defined boundary; this boundary is normally either a change in mining method or the end of the mineralised deposit. Then, the stopes are mined out in sequence from the end of the drift working back to the access. Similar in process to transverse longhole 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.

16.2.5.2 Drift and Fill Mining

In drift and fill mining areas the ore zone will be divided into 5 m high horizontal slices (or lifts), and 6 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.

Figure 16.26 Typical Drift and Fill Mining Layout



Ore zones are divided into 5.0 m H horizontal slices (or lifts), and 6 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 6.0 m W flat back. Drifts will be driven with flat backs for recovery purposes.

Mining widths are limited to 6 m to reduce the length of ground support in the back and walls to 2.4 m. If mining widths exceed 6 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.

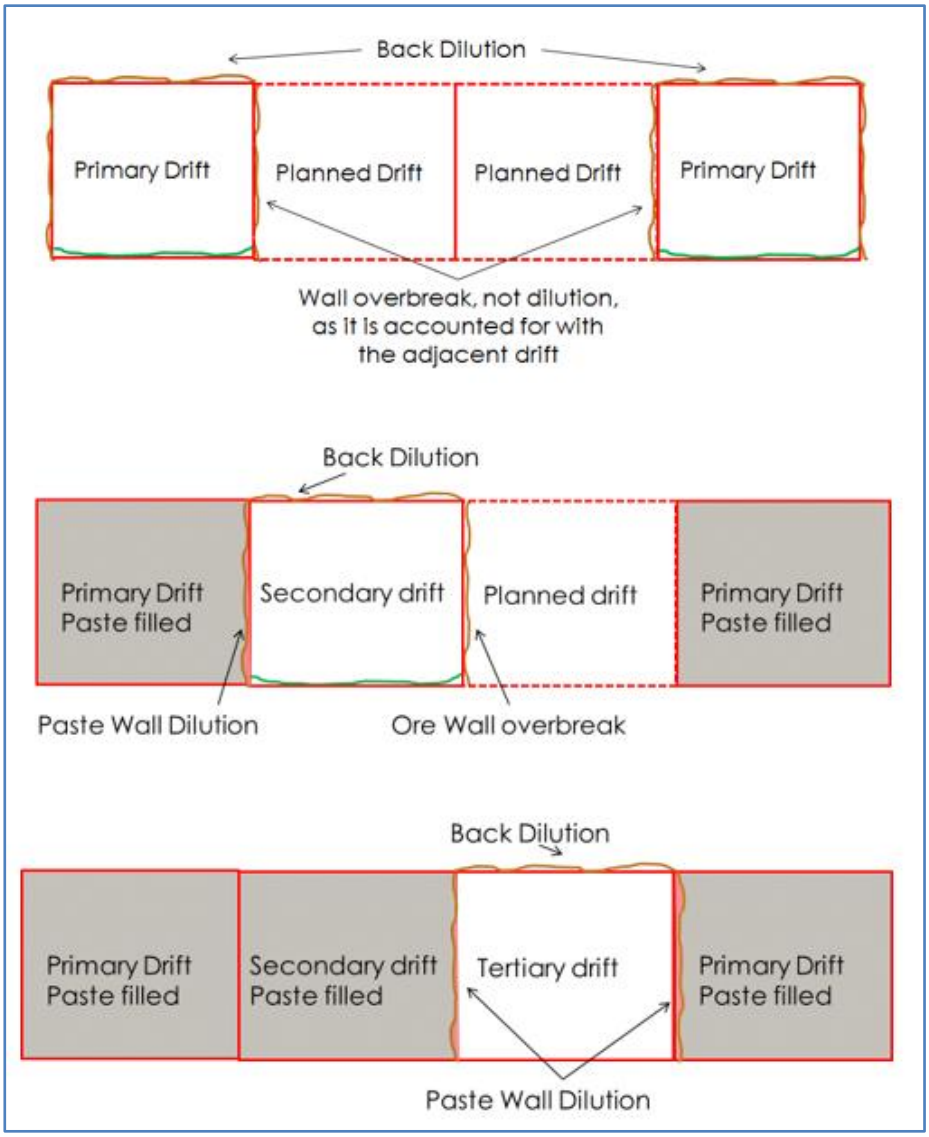
The drifts will be filled with paste fill on completion. After the paste fill has cured to the required strength, another drift will be driven next to the fill.

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 16.27). 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 6 m (flat back) in cross section and 15 m long, so using the density of 3.14 t/m3 the total tonnage per stope will be 1,413 t.
- Using a single heading production rate of 376 t/d (4.0 m/d advance rate), one stope will be mined in approximately 3.75 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/3.75 days/stope + the initial panel ~9 panels).
- Each panel (defined as the combination of primary, secondary and tertiary) is 18 m wide (6 m/stope x 3 stopes), nine available panels translate to 162 m required for the total length of one active area, including the 6 m wide access drift.
- 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 300 t/d at 20% loss blast.
- Second lifts in the drift and fill stopes will be accessed by sliping of the backs in the access ramp and installing new ground support.

Figure 16.27 Primary, Secondary and Tertiary Stoping



16.2.6 Dilution and Recovery Factors

16.2.6.1 Waste Drifts

The waste drives in 3 year plan is 6 m W x 6 m H. Waste drift recovery and dilution are 100% and 10%.

All ore development will be done using a 4.4 m round length. Drilling will be done with an automated drilling system and drilling accuracy will be good to achieve a 4.0 m advance. The same blast pattern will be applicable to the 4.5 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.

16.2.6.2 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. Dilution percentages range between 2.3% and 8.2% based on stope length for primary stopes, and between 8.1% and 14% based on stope length for secondary stopes.

The Platreef 2025 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. The thickness of overbreak for paste walls is 0.5 m.

16.2.6.3 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 16.19.

Table 16.19 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

16.2.7 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
- Oversized muck left in stope (Including mine design recovery and oversize rock/unblasted wall ore)
- The mining recovery was lowered to 90% for stopes in sill pillars being extracted last
- A mining recovery factor of 98% was applied to drift and fill stoping areas and to stope end slashes

16.2.8 Mine Access Designs

Mining zones included in the mine plans for the Platreef 2025 FS, occur at depths ranging from approximately 570 m to 1,170 m below surface. Shaft 1, Shaft 2 and Shaft 3 are the primary access points to the mine, along with four ventilation raises.

Access from the shaft complex 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. Another main haulage level.

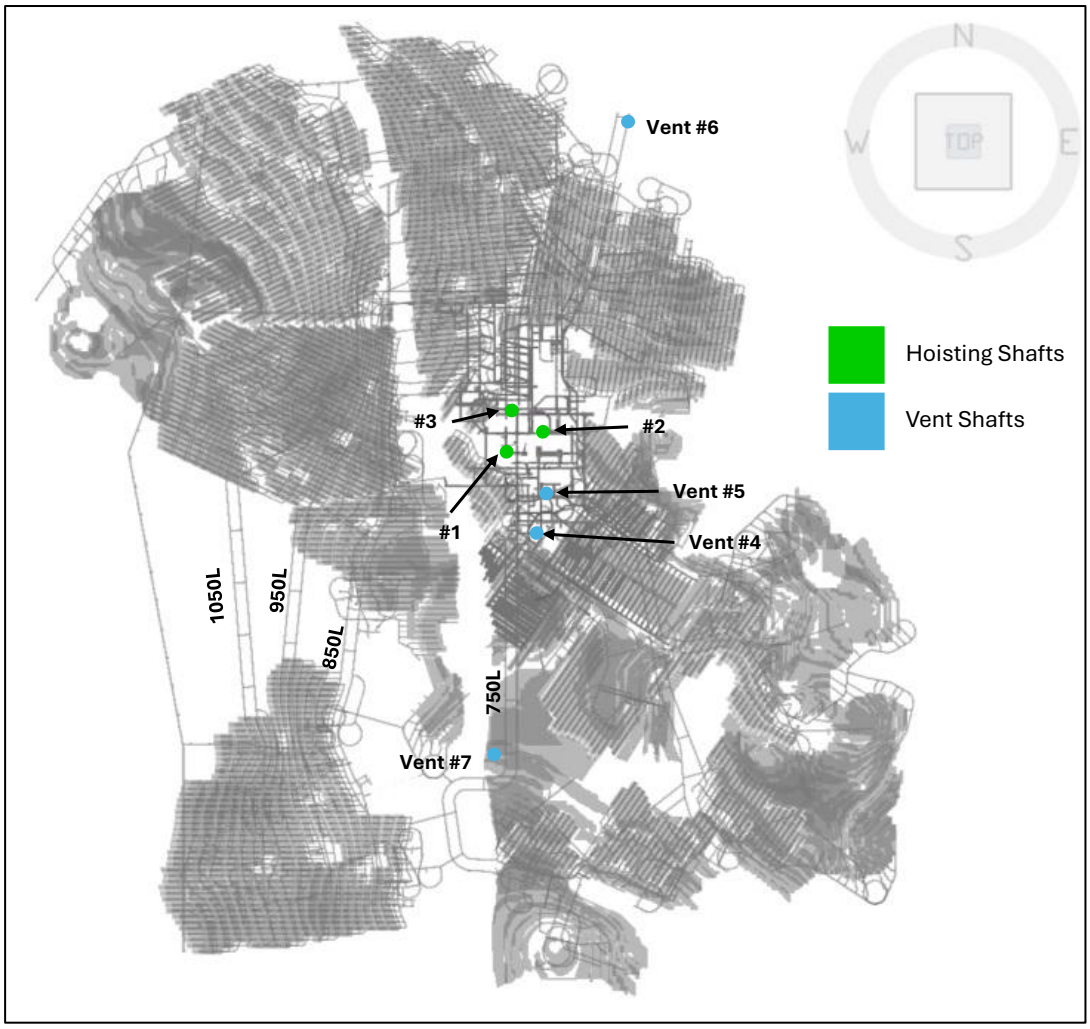
16.2.8.1 Shafts

Shaft functions and design parameters are summarised in Table 16.20. Shaft locations are shown in Figure 16.28.

Table 16.20 Shaft Functions and Design Parameters

Shaft	Function	Diameter (m)	Depth (m)
Shaft 1	Early mine development at 60–65ktpm/escape/ventilation intake	7.25	981.6
Shaft 2	Service shaft for logistics and services	10.0	1,100
Shaft 3	Production – 4.0 Mtpa (labour and material/mine services/ventilation exhaust)	5.1	950
Shaft 4 Ventilation	Ventilation intake	5.1	750
Shaft 5 Ventilation	Ventilation exhaust	5.1	750
Shaft 6 Ventilation	Ventilation exhaust	5.1	750
Shaft 7 Ventilation	Ventilation exhaust	5.1	750

Figure 16.28 Shaft Locations



Shaft 1

Shaft 1 has a 7.25 m diameter and sinking commenced in October 2015 and reached a planned depth of 981.6 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.

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.

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 m³/s) to limit losses.

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 m³/s).

Shaft 2 main features are summarised below:

- Shaft functions – production/service/ventilation intake.
- Total depth – 1,100 m.
- Shaft diameter – 10 m.

16.2.8.2 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 and 750 m Level to 950 m Level.
- Raise diameter – 6.0 m finished (6.1 m reamer head).
- 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.
- This will be equipped for second egress access.

16.2.8.3 Ventilation Raise 2

- Raise functions – ventilation exhaust.
- Total depth – 963 m.
- Raise diameter – 6.0 m finished (6.1 m reamer head).
- Operating level stations – 750 m Level, 850 m Level, and 950 m Level.

16.2.8.4 Ventilation Raise 3

- Raise functions – ventilation exhaust
- Total depth – 743 m.
- Raise diameter – 6.0 m finished (6.1 m reamer head).
- Up cast at maximum – 20 m/s.

16.2.9 Ventilation

Fresh, refrigerated air will be supplied via Shafts 1, 2 and 5 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. The ventilation system is based on a "once-through" system, eliminating the re-use of ventilation between drifts, mining levels and mining zones. Heat, dust and blasting fumes will be extracted directly to the main return system utilising a forced-exhaust secondary ventilation system. This ventilation strategy has major benefits on accumulated heat loads, airborne pollutants (such as diesel particulate matter (DPM)), re-entry times, mining flexibility and workplace conditions in general.

Refrigerated air will be supplied from surface bulk air coolers (BAC) situated in the sublevel drifts at Shaft 1 and Shaft 2. The refrigeration and cooling strategy is based on bulk air cooling and will not make use of underground chilled water. A detailed refrigeration study was conducted by others and is not included in this scope of work. The outcome of this study was used in the refrigeration design.

16.2.9.1 Primary Ventilation

Ventilation System Design Considerations

The ventilation design criteria itemise several applicable air condition parameters that are used to determine air quantity requirements for the life-of-mine. The air quantities required 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 required in production and development drives. Ventilation modelling is used to verify the expected air quality and air temperatures (wet-bulb and dry-bulb) in production and other areas.

Primary Ventilation Requirements

Underground mining operations at Platreef Mine will take place in separate zones that share a common vertical shaft system for ore and personnel transport. Capital development from the Services Shaft (Shaft 1) started in 2022 on 750 m Level, 850 m Level and 950 m Level. The production schedule aims to achieve the first production in 2025. Production level development will continue until steady state production is reached in 2029, when 4.1 Mtpa ore will be mined.

Primary ventilation requirements are a function of mining activities in conjunction with mobile equipment. Therefore, depending on the mining activity in the production areas, air quantities ranging between 0 m^3/s (no activities) and up to 18 m^3/s are required. For non-production levels, a base ventilation requirement of 30 m^3/s is allowed for. The primary ventilation requirements are shown in Table 16.21.

Table 16.21 Steady State Primary Ventilation Requirements

Description	Air Quantity (m^3/s)	Quantity	Total Air Quantity (m^3/s)
Drifts/LHS Developing	18	6	108
DF/LHS Stopes	18	32	576
Non-Production Levels	30	12	360
Leakage		10%	104
Workshops	25	3	75
Crushers	50	1	50
Conveyor Belt	50	1	50
Total Ventilation			**1,323**

Primary Ventilation Infrastructure

The dimensions and airflow capacities of the intake and exhaust shafts are shown in Table 16.22.

Table 16.22 Intake and Exhaust Shaft Capacities

Description	Depth (m)	Diameter (m)	Area (m^2)	Velocity (m/s)	Airflow (m^3/s)
Intake Shafts					
Shaft 1	980	7.25	41.3	12	495
Shaft 2	1000	10	78.5	12	940
Shaft 5	750	5.1	20.4	26	530
Total Intake					**1,965**
Return Shafts					
Shaft 3	950	5.1	20.4	10	204
Shaft 4	750	5.1	20.4	26	530
Shaft 6	850	5.1	20.4	26	530
Shaft 7	850	5.1	20.4	26	530
Total Return					**1,794**

Intake System

During steady state, fresh air will be supplied from surface via Shaft 1, Shaft 2, and Shaft 5. 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.

To reduce the impact of a shaft pillar, Shaft 5 will be raised bored to 750 m Level and extended to 850 m Level with two Ø4.1 m fresh air raises.

Return Air System

During steady state, return air will exhaust to surface through three exhaust shafts (Shafts 3, 4 and 6) using a combination of surface and underground fan stations.

Shaft 3 will be equipped for ore hoisting, broken ore will be transported with a conveyor belt and a loading bin arrangement on 920 m Level. During the equipping of Shaft 3, the Shaft 2 raise borehole will be utilised as the main exhaust shaft in conjunction with the two Ø1500 mm ventilation columns installed inside Shaft 1.

Once Shaft 4 is operational and the main fan station is commissioned in September 2025, most of the return air will be exhausted via Shaft 4. However, the development on 950 m North will be ventilated with an exhaust duct that discharges air into Shaft 2 until such time that sliping of Shaft 2 commenced.

When the equipping of Shaft 3 is completed, ore hoisting will commence via a loading bin arrangement from 920 m Level. To comply with legislation, the conveyor belt feeding the loading bin must be ventilated directly to a return system. For this reason, the loading bin was specifically designed to accommodate ventilation doors and a set of exhaust fans to ensure compliance. During steady state, Shaft 3 will be used to ventilate the workshops, conveyor belts and crushers.

The increase in ventilation demand will be satisfied with the introduction of an additional exhaust shafts. Shaft 6, complete with main fan station, are required by 2032, ultimately increasing the return air system to 1300 m^3/s.

Each production level will be ventilated as separate ventilation districts. Inter-level return air raises will be used to establish a return airway system between mining levels and, ultimately, to the main return airway on 750 m Level. To prevent increased main fan pressures, the diameter of inter-level return raises must be increased from Ø3.0 m to Ø4.1 m.

16.2.9.2 Secondary Ventilation

This section provides strategies to ventilate development and stoping production areas as well as services such as workshops, pump station, crushers, etc.

Ventilation on Demand (VoD)

A ventilation-on-demand philosophy was adopted during the FS 2022 study and was based on ventilating mining activities rather than available drifts and stope access crosscuts. The philosophy adopted three scenarios:

- Loading with LHD - 18 m3/s (Exhaust duct)
- Drilling/support - 18 m3/s (Exhaust duct)
- No activity - 0 m3/s

Initial Development Phase

The initial development commenced from Shaft 1 and includes establishing mining levels, ramps, and access to the ore body. During this period, the ventilation strategy is to exhaust ~130 m^3/s using two sets of three fans on surface, arranged in series and connected to two (2) Ø1500 mm diameter, Corten ventilation ducts installed in Shaft 1. The Ø1500 mm diameter ventilation ducts extend to the three main levels (750 m Level, 850 m Level and 950 m Level), where they are reduced to Ø1200 mm, delivering 40 m^3/s to each level. On each level, there are two 110 kW fans that assist the pressure of the exhausting ducts. The Ø1200 mm on-level exhaust ducts are moved and extended to create a through ventilation scenario as development advanced on each level.

Ore Development Ventilation Strategy

A fixed exhaust with a force overlap ventilation system will be used to ventilate the drifts and stope access crosscuts. The main difference compared to a traditional exhaust-force overlap system is that the exhaust duct will not be extended with the advancing face. A single Ø1400 mm diameter exhaust duct will be carried over the length of the perimeter drive. From the perimeter drive, the Ø1400 mm exhaust duct will be reduced to Ø1,016 mm and extended 20 m into each stope access drive. A Ø760 mm diameter force duct will be carried from the Ø1016 mm exhaust duct and will extend with the advancing face and should not be carried more than 20 m from the face. The overlap distance between the exhaust duct intake and the force duct intake point should at least be 10 m. The Ø1400 mm parameter drive exhaust duct will be connected to a 420 kW centrifugal fan that is installed inside the RAR cubby. Each stope access development end will be equipped with a Ø1016 mm actuated damper inside the exhaust duct to control the ventilation flow based on the activity. A single-stage 45 kW, Ø760 mm force fan will be installed on each force duct. The ventilation system can support the mining of four drifts or access crosscuts, each supplied with 18 m3/s.

The aim of this system is to benefit from the advantages associated with an exhaust-force system without the complexity of extending two ventilation ducts, moving the force fan and force duct to maintain the required overlap distance.

Stoping Ventilation Strategy

The ventilation requirements for the longhole stoping phase are similar to the development phase. For this reason, the ventilation system utilised for the development phase will be re-used during stoping. Stoping will be done on a retreat basis, and likewise, the force duct will be retreated with stoping. The ventilation system can support two stopes (primary or secondary) in the pre-developed block in conjunction with the developing block.

Once the stope is completed, the fixed exhaust duct, exhaust fan and force fan can be removed, and the t-piece blanked off. A retaining wall will be constructed inside the stope access, and the stope will be backfilled. The process will be repeated with the follow-on primary stopes and subsequently with the secondary and tertiary stopes (if any).

16.2.9.3 Mine Heat Load and Cooling

Mine heat load is largely dictated by production rate, mining equipment, mobile equipment, mining depth and production area distance from shaft infrastructure. Mine mobile equipment, mining depth and distance from shaft infrastructure are the most significant factors in the total heat generated at steady state production.

General descriptions of the main heat components are given below:

- Surrounding Rock - Transfers heat from the hot (virgin) rock in the interior. This heat transfer increases with depth due to higher virgin rock temperatures and a greater exposed rock surface area. The rate of heat transfer is influenced by the age of the excavation; newly exposed rock surfaces transfer heat rapidly, but this rate decreases over time. In general, heat transfer from the surrounding rock ceases approximately two years after excavation.
- Broken Rock - Creates a heat load because it enters the underground environment at a warm temperature and ultimately is cooled to the ambient underground temperature. The component increases with depth with higher virgin rock temperature.
- Auto-Compression - As a result of the conversion of potential energy, heat from auto-compression will depend on the depth to which the air travels and the airflow volume circulating through the mine.
- Auxiliary Fans - Do not perform useful thermodynamic work, and all electrical energy entering a fan motor is converted into heat energy. The magnitude of this component depends on the method used for secondary ventilation control and the number of operational fans. In general, the heat load is not depth dependent.
- Mobile Diesel Equipment - Generates a heat load of nearly three times that of the rated engine power, assuming no energy is converted to useful work due to the inherent inefficiency of the internal combustion engine. Typically, 35% of the energy generated from diesel fuel is converted to potential energy. The remaining 65% is converted to heat.

Overall Heat Load Components and Heat Modelling Results

The main feature of the ventilation and cooling design process is a fully interactive simulation of the energy balances to determine air temperatures, airflow rates, heat loads and cooling requirements. Ventilation simulation models provide this, and modelling results have determined that the total mine heat load peak at ~60.0 MW for the year 2036 snapshot.

16.2.9.4 Refrigeration and Cooling

Construction of Phase 1 (7 MW) commenced and is scheduled to be completed by January 2025. The detailed design for Phases 2 and 3 is currently conducted by Howden, Mine Cooling Division.

Cooling Strategy

Surface bulk air coolers (BAC) are required to supplement the cooling requirements underground. Ventilation network simulation and calculations indicate that 28.0 MW of bulk air cooling will be required at steady state. Ambient air drawn into Shaft 1 and Shaft 2 will be cooled using a closed-loop cooling coil-type heat exchanger supplied with chilled water from the refrigeration plant. Phase 1 will consist of 7 MW air-cooled chillers and a 7 MW surface BAC at Shaft 1. Phase 2 will consist of additional 7 MW water-cooled chillers, complete with Condenser Cooling Towers (CCT) and an additional 7 MW of BAC capacity at Shaft 1. Phase 3 will consist of an additional 14 MW water-cooled chillers and associated CCTs with a 14 MW BAC at Shaft 2.

Bulk Air Coolers

The Shaft 1 bulk air cooler will be constructed in the proximity of the shaft. Refrigerated air will enter a connecting sub-level drift, adjacent to the shaft and mix with air from surface at the sub-bank. Shaft 1 is an equipped shaft with an airflow capacity of ~500 m^3/s. The temperature in Shaft 1 will be limited to 12.0° C_{WB}/12.0° C_{DB}.

The Shaft 5 is an unequipped shaft with an airflow capacity of ~600 m^3/s. The temperature in Shaft 5 will be limited to 8.0° C_{WB}/8.0° C_{DB}. The Shaft 5 bulk air cooler will be constructed adjacent to the shaft and connected directly with a shaft top bend. The shaft top bend will allow ambient air to enter and mix with air from the bulk air cooler.

The bulk air coolers will have the following process specifications (to be confirmed from the study being conducted):

Table 16.23 Surface Bulk Air Cooler Duty Requirement

Description	Shaft 1	Shaft 5
Air Cooling Duty (MW)	14.0	14.6
Airflow into shaft (m^3/s)	500	620
Airflow onto BAC (m^3/s)	500	400
Air temperature onto BAC (WB/DB)	20.5/30.0	20.5/30.0
Air Temperature off BAC (WB/DB)	11.6/11.6	8.0/8.0
Mixing Air Temperature in shaft (WB/DB)	11.6/11.6	14.4/18.0

16.2.9.5 Surface Main Fan Stations

Two surface exhaust main fan stations are planned, one on each exhaust shaft (4 and 6). Each fan station will be identical and will comprise a bifurcated centrifugal fan arrangement.

Fan Station Description

Bifurcated main fan stations will be installed on surface at Shafts 4 and 7. At each station, two fans will be operational to achieve the full duty point at steady state (no standby fans at steady state). The peak operating duty per fan station will be 600 m^3/s (at a shaft collar) at a pressure of 5.3 kPa and density of 1.02 kg/m^3.

16.2.9.6 Underground Fan Stations

Four booster fan stations are planned for on 750 m Level, 850 m Level, 920 m Level, and 950 m Level. These fan stations will ventilate the workshops and conveyor belt system via Shaft 3.

Fan Station Description

Axial flow fan stations will be installed underground at Shaft 3. The fan stations for Shaft 3 will be a permanent installation and will consist of two axial flow fans installed in a wall at 750 m Level, 850 m Level, 920 m Level and 950 m Level stations. The primary purpose of the fan station at 920 m Level is to ventilate the conveyor belt directly to return (Shaft 3).

16.2.10 Mine Backfill

The mining methods chosen for Platreef require backfill as a support medium. The backfill system's size and the timing of module implementation have been re-evaluated, starting at an initial capacity of 0.77 Mtpa and ramping up to 4.07 Mtpa (0.77 Mtpa + 3.3 Mtpa).

The backfill system will utilise a cemented paste fill (CPF) system. Initially, it will include a Phase 1 paste plant, supplied by the 770 ktpa Phase 1 concentrator, along with a backfill reticulation system. This will later be replaced by a Phase 2 paste plant and reticulation system to support the increased production capacity of 4.07 Mtpa of ore.

16.2.10.1 Backfill Method

Cemented Paste Fill will be used for all stopes, regardless of the mining method. In certain secondary stopes, the CPF will be supplemented with waste rock. The waste rock will be trucked from underground development ends and dumped into the stopes from the top access drive. A summary of the backfill requirements and CPF system design criteria are shown in Table 16.24.

Table 16.24 LHOS Backfill Requirements and CPF Design Criteria

Item	Phase 2 Filtration Plant	Phase 2 Paste Plant
Plant Feed Medium	Tailings thickener underflow slurry (55–60%m)	Dry filter cake tailings (89.7%m)
Product Medium	Filter cake (89.7%m)	Cemented paste (76%m)
Throughput Per Module	Design: 370 dry t/h Nominal: 296 dry t/h	Design: 210 m³/h (347 dry t/h) per module Nominal: 180 m³/h (297 dry t/h) per module
Output Density	2.591 t/m³ (89.7%m)	2.03 - 2.08 t/m³ (74 - 76%m)
Annual Operating Hours	6,012 h/y (329 d/y x 18.3 h/d)	6,131 h/y (329 d/y x 18.7 h/d)
Paste Plant Utilisation & Availability	Availability: 329 d/y, 90% Utilisation: 246 d/y, 75%	Availability: 329 d/y, 90% Utilisation: 227 - 268 d/y, 70% - 83%
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: 3,704,000 t/y Maximum: 328,000 t/mth	
Ore Production for DF	Maximum: 1,464,900 t/y Maximum: 140700 t/mth	
Ore Production for Backfilled Voids	DF and LHOS Ore mined: 3,799,000 t/y Ore drive development: 1,333,000 t/y Peak monthly voids generated: 1,028,000 m3	
Filling Allowances	Backlog filling allowance: 10% Overbreak allowance: 5%	
Backfill Voids	100% (LHOS areas to be mined in cut and fill method)	
Backfill Panel Size	LHOS: Nominally: 15 m on strike, 26 m high, length varies 10 m - 60 m. Nominal Stope Size: 19,500 m³	
Required Backfill Strength (IVP 01)	Primary plug pour: 150 kPa at 2 days Primary bulk pour: 350 kPa at 28 days Secondary plug pour 150 kPa at 2 days Secondary bulk pour: 350 kPa at 28 days Capping pour all stopes: 1200 kPa at 28 days	

The cemented paste fill properties are summarised in Table 16.25.

Table 16.25 Cemented Paste Fill Properties

Item	Value/Description
Backfill Mass Concentration Range	74–76 %m (70 Pa–85 Pa Yield Stress)
Rheology	4% Binder Addition Yield Stress Constant A: 3.75 × 10^3 Yield Stress Exponent B: 13.86 Viscosity Constant A: 10 × 10^6 Viscosity Exponent B: 48.04 6% Binder Addition Yield Stress Constant A: 7.20 × 10^3 Yield Stress Exponent B: 16.16 Viscosity Constant A: 10 × 10^6 Viscosity Exponent B: 59.53 8% Binder Addition Yield Stress Constant A: 4.68 × 10^3 Yield Stress Exponent B: 14.56 Viscosity Constant A: 10 × 10^6 Viscosity Exponent B: 49.68
Compressive Strength Correlations	7 day strengths: W:B = 296.02 × UCS $^{-0.625}$ 28 day strengths: W:B = 206.32 × UCS $^{-0.553}$

16.2.10.2 System Description

The Phase 1 plant is a dry feed batch plant. The plant will receive dry filter cake as tailings feed to produce paste backfill. The Phase 1 plant will only target stopes near the shafts according to the mine plan, and hence it is designed as a gravity feed plant.

The Phase 2 Pressure Filter plant will receive tailings in the form of thickener underflow slurry from the process plant at a mass concentration of 55 to 60%m. The slurry will be stored in agitated filter feed tanks next to the filter plant building.

The plant will consist of two parallel modules. Four (1 duty and 1 standby pump per module) filter feed pumps will draw tailings from the filter feed tank and pump it to one of the two vertical plate pressure filters (module 1 or 2) for further dewatering. A single operating module can sustain the instantaneous throughput requirement of a single paste plant module.

Filter cake produced by the pressure filters will drop onto filter cake collector conveyors which will deposit the cake onto a stockpile (covered) at the plant. A front-end loader will collect filter cake from the stockpile and load the material into the filter cake loading system.

The Phase 2 paste plant consists of two modules that can be operated simultaneously. By implementing modularised designs, the plant can be constructed and scaled to the desired capacity and operated simultaneously to accommodate multiple different pours to different stopes as required.

The plant will receive tailings in the form of stockpiled filter cake. The filter cake will be fed into the plant through a feeding system. Filter cake will be loaded via a ramp into surge bins. Live bottom feeders will extract the filter cake onto the feeding conveyor. The feeding conveyor will be equipped with tramp metal belt magnets to extract any foreign material which bypassed the surge bin screens.

The feeding conveyor will discharge the filter cake into the first stage of mixing (conditioning mixer). The mixer aims to repulp the filter cake with the addition of process water into a high solids mass concentration slurry/paste. The conditioning mixer will discharge into the continuous paste mixer.

The continuous paste mixers will receive tailings paste, trim slurry, water and cement from the binder system following the pre-defined recipes based on stope location and strength required. 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.

The paste will be delivered to the designated stope by the UDS and will discharge from the top drive to fill the stope. A barricade will retain the initial plug pour that will cure before the bulk of the stope will be filled. Should subsequent mobile equipment top access be required after pours, a capping pour can be deposited for the final 5 m residual void.

Mix Designs

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 risk of barricade failure.

The strength of the bulk (main) pour for the primary stope is determined by the vertical exposure of the stope. The secondary stopes will require backfilling as well to prevent potential liquefaction, however lower final strengths are achieved with the bulk pours. The final capping pour for both primaries and secondaries requires a higher strength to allow for bogging of the next lift.

Mass Balances

For the Phase 2 Filter Plant, the tailings will be received as thickener underflow slurry from the Phase 2 concentrator. The produced filter cake will be stockpiled to create a buffer for the adjacent Phase 2 CPF Plant. Filter cake will be loaded into the Paste Plant via the loading system.

Summary mass balances of the main streams are presented in Table 16.26 and Table 16.27 for the Phase 2 Filter and Paste plants, respectively.

Table 16.26 Summary Mass Balance for Phase 2 Filter Plant (Per Module)

Stream Description	Dry Solids	Water	Total Throughput	Density	Concentration
Tailings Slurry to Filter	401 t/h	268 m³/h	395 m³/h	1.694 t/m³	60.0%
Filter Cake to Stockpile	370 t/h	43 m³/h	159 m³/h	2.591 t/m³	89.7%
Filtrate	0.3 t/h	283 m³/h	283 m³/h	0.998 t/m³	0.1%

Table 16.27 Summary Mass Balance for Phase 2 Paste Plant (Per Module)

Stream Description	Dry Solids	Water	Total Slurry	Density	Concentration
Filter Cake to Conditioning Mixer	273.3 t/h	31.5 m³/h	117.6 m³/h	2.590 t/m³	89.7%
Trim Water to Conditioning Mixer	-	54.9 m³/h	54.9 m³/h	0.997 t/m³	0.0%
Slurry to Paste Mixer	273.3 t/h	86.6 m³/h	359.6 m³/h	2.082 t/m^3	76.0%
Trim Water to Paste Mixer	-	3.6 m³/h	3.6 m³/h	0.997 t/m³	0%
Cement to Paste Mixer	11.4 t/h	0.0 m³/h	3.7 m³/h	3.100 t/m³	100.0%
Paste Backfill to UDS	284.7 t/h	90.2 m³/h	180.0 m³/h	2.081 t/m³	76.0%

Underground Reticulation System Components (UDS)

The underground reticulation componentry of the Phase 1 backfill plant and Phase 2 backfill plant operations will follow a similar design.

All underground and surface steel backfill piping will utilise, as far as possible, a standard pipe clamp for all piping. These clamp-type pipe couplings (also known as T&E Couplings, Hi-press Couplings, Backfill Couplings, U-coupling, taper pipe coupling etc.) are used extensively throughout the South African UG mining industry for backfill and water and are rated for 25 MPa. They consist of shoulders/hubs/stub-ends that are welded to the pipe and two clamps that are installed using two bolts.

The pipe spools consist of standard bends and exact length spool pieces which make replacement underground easier. In addition to this, all ancillary spool pieces required for the UDS are shown; dump tees, transition pieces, bursting disc assemblies, pressure transmitter, etc.

On surface, high-pressure knife-gate valves and flush-mounted piston type water injection valves will be used to isolate the PD pumps and supply flush water from high-pressure multistage water pumps.

As far as possible, manual switch-over spools will be used underground to distribute backfill along the various routes of the UDS from the borehole. Swing bend arrangements will be used to direct the flow into the required areas. For main trunk routes actuated switch-over valves can be considered.

Actuated paste-specific dump valves will be installed for pressure release to protect the borehole in addition to the blasting tees usually used. For longer routes, dump valves or switch-over spools should be installed at suitable locations to divert flushing water to designated sumps.

Manual low-pressure, chain-operated butterfly valves will be installed just before the stopes to allow for the flush water to be diverted away from the stopes. It is good practice to divert as much flush water away from the stopes as possible.

Pressure transmitters installed in the reticulation system will be critical for monitoring the operational condition of the system and will be tied into the paste plant control system, reporting pressure data back to the plant operators and alerting of any change in the state of the system. Pressure transmitters will be installed at the top and bottom of the borehole, at the PD pump discharge, and at suitable locations underground. Pressure transmitters will be paste service, flush-mounted pressure transmitters with abrasive duty seals, rated for 20 MPa to track the magnitude of potential over-pressurisation.

To protect against over-pressurising the pipeline, top-hat-style rupture discs will be installed at the base of the borehole at the Sch120 to Sch80 specification change as well as at the Sch80 to Sch40 specification change. The top-hat-style discs are designed to be flush-mounted with the pipe obvert to prevent cemented backfill from building up. The rupture disc release is channelled into a pipe that channels the paste to ground level and away from personnel.

16.2.11 Equipment

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.

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.

16.2.11.1 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 16.28.

The jumbos will be two-boom units with a 16 ft feed. 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, Epiroc Easer and an emulsion loading t utility vehicle. 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 utility vehicles 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 (Zones 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 16.28 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	26
Longhole Production Drill Rigs	Ore produced from stopes	2	9
Roof Bolters	All lateral ore (including drift and fill) and waste development	5	5
Shotcreter	Development	3	7
Waste Development LHD (14t)	All lateral waste development	9	9
Production LHD (17t)	All lateral ore development and ore produced from stopes	2	14
Haulage Trucks (50t)	All waste tonnes and ore tonnes	5	15
Explosives Utility Vehicle (Dev and LH Charging)	Mine wide	4	20
Scissor Lifts	Mine wide	4	19
Personnel Carriers - LDV	Mine wide	8	21
Grader	Mine wide	-	6
Utility Equipment - UG	Mine wide	6	19
Multi-Purpose Vehicles (Manito)	Mine wide	4	12
Agicar (Concrete Mixing Truck)	Mine wide	3	12

16.2.11.2 Fixed Equipment

A list of major fixed equipment by category for Platreef is presented in Table 16.29. This list includes fixed equipment, water handling, electrical, material handling, ventilation, and miscellaneous.

Table 16.29 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

16.2.12 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.

16.3 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.

16.3.1 Available Hours

Available time is total shift time minus fixed non-productive time. Table 16.30 summarises the available hours per shift that have been calculated for the Project.

Table 16.30 Available Hours per Shift

Description	Hours
Shift Length	11.50
Subtotal Non-Productive Time	2.50
Available Work Time Per Shift	**9.00**

16.4 Productivity Rates

16.4.1 Waste Development

Waste development will be done through various types of ground conditions with different support requirements. The 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. Note that initially the development is capped by hoisting constraints and skills levels. The crews are planned at 120 m/mth up to December 2026 when the efficiencies are increased to 150 m/mth. The LOM efficiency has been increased to 180 m/mth in 2030.

16.4.2 Ore Development

The first principles rate calculation for ore development applies to 6 m W x 5 m H flat top 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).

16.4.2.1 Longhole Stoping

Longhole stoping is the primary mining method. 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 paste fill will be used as post extraction support.

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. 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.

16.4.2.2 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.

Two methods of drift and fill mining were chosen: 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 applied to determine the production rates.

16.4.3 Mine Development Plan and Scheduling

The mine development has been broken down into four main stages.

- Stage 1 – Lateral Infrastructure development to support Shaft 3
  - This stage includes the holing from different elevations into Shaft 3, tips on all three levels, ore passes, ventilation passes, crusher excavation, conveyor excavation and loading level. Main restrictions are ground handling limitations and ventilation.
- Stage 2 – Lateral development to initial stopes and services infrastructure
  - The second stage is the development on 750 m and 850 m Levels to the initial stopes and access to Shaft 4.
  - Development of initial workshops, emulsions bays, paste fill holes, dams and other services infrastructure before Shaft 3 is commissioned. The main constraints are ground handling, ventilation and logistics.

- Stage 3 – Lateral development after Shaft 3 commissioning
  - The third stage is the ramp up and acceleration of level development off Shaft 1 and Shaft until Shaft 2 is completed and functional. Additional development and longhole stoping crews are added to ramp up to 4 Mtpa. Shaft 1 utilised for logistics only. This stage is limited by ground handling up to 2028 when 4 Mtpa is reached and the major constraint is logistics via Shaft 1.
- Stage 4 – Lateral development after Shaft 2 commissioned
  - The fourth stage occurs after Q3'28 when Shaft 2 is commissioned for personnel and material. The remaining constraint now is ground handling with the combined hoisting capacity of Shaft 1 and 3 being 4.7 Mtpa.

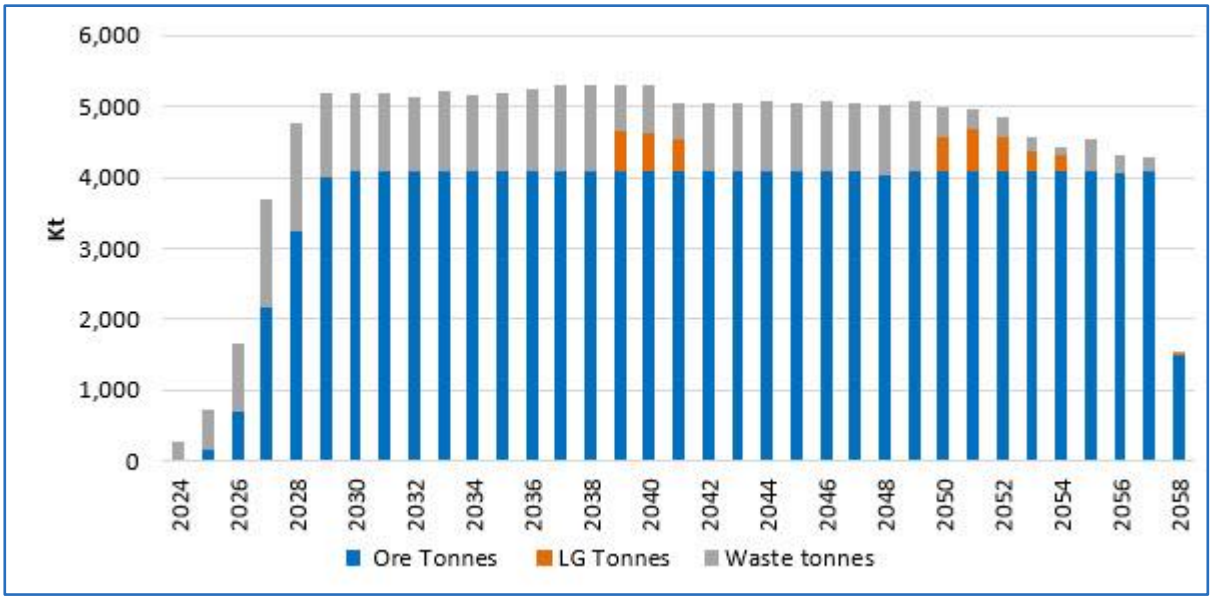
16.4.3.1 Pre-production Development Schedule

The initial refinement of the mining schedule focused on the period up to the end of the pre-production period. This period is restricted in advance rates due to the following:

- Hoisting constraints (60–65ktpm up to February 2026).
- 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.
- Learning curves have been incorporated with (2026 equivalent of 120 m/tm per jumbo, 2027 to 2029 equates to 150 m/mth per jumbo and 180 m/mth per jumbo after 2029).

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 16.29 shows the annual rock production for the LOM.

Figure 16.29 Annual Rock Production



16.4.3.2 Life-of-Mine Development Schedule

Once the pre-production development was defined, the overall LOM development was scheduled, (see Table 16.31 and Table 16.32). 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 240 km of lateral waste development, or 27.2 Mt of waste material, is mined. The overall waste development is summarised in Table 16.33.

Table 16.31 Life-of-Mine Development Schedule to 2038

Description	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Shafts																
Shaft Development	m	3,329	239	837	108	1,192	294	-	-	-	494	166	-	-	-	239
	kt	382	16	215	7	80	20	-	-	-	33	11	-	-	-	-
Vertical Development																
Vertical Development	m	15,738	371	331	346	1,458	1,059	790	364	742	271	660	736	1 083	986	401
	kt	375	13	10	11	33	28	22	8	16	6	14	21	24	25	9
Lateral Waste Development																
Lateral Development	m	238,483	4,191	5,652	12,707	12,503	10.114	9,679	9,732	9,318	9,815	9,577	9,749	10,323	10,563	10,709
	kt	26,980	532	731	1,509	1,427	1,139	1,079	1,079	1,042	1,083	1,044	1,080	1,139	1,199	1,201
Waste Development																
Waste Development	m	257,658	4,802	6,820	13,162	15,152	11,467	10,469	10,095	10,060	10,580	10,403	10,485	11,406	11,549	11,110
	kt	27,751	561	956	1 528	1.54	1.187	1.101	1.087	1.058	1.122	1.069	1.101	1.163	1.224	1.209
Low Grade Development																
Low Grade Development	m	31,864	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	kt	3,661	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste and Low Grade Development																
Waste and Low Grade	m	289,522	4,802	6,820	13,162	15,152	11,467	10,469	10,095	10,060	10,580	10,403	10,485	11,406	11,549	11,110
	kt	31,412	561	956	1,528	1,540	1,187	1,101	1,087	1,058	1,122	1,069	1,101	1,163	1,224	1,209
Ore Development																
Ore Development	m	169,496	1,407	2,165	8,059	7,282	12,820	7,856	9,541	8,435	11,096	10,347	10,552	8,820	7,308	4,484
	kt	18,378	170	254	933	784	1,371	842	1,009	894	1,167	1,085	1,123	942	790	487
Total Development																
Total Development	m	459,018	6,209	8,984	21,221	22,434	24,287	18,325	19,637	18,494	21,675	20,750	21,037	20,226	18,856	15,594
	kt	49,789	730	1,210	2,461	2,324	2,558	1,943	2,096	1,952	2,289	2,154	2,223	2,105	2,015	1,696

Table 16.32 Life-of-Mine Development Schedule to 2058

Description	Units	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	2054	2055	2056	2057	2058
Shafts																					
Shaft Development	m	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	kt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Vertical Development																					
Vertical Development	m	594	768	296	224	360	629	472	369	195	374	431	307	546	294	62	-	218	-	-	-
	kt	13	17	6	5	8	14	10	8	4	8	9	7	12	6	1	-	5	-	-	-
Lateral Waste Development																					
Lateral Development	m	5,859	5,944	4,597	8,565	8,540	8,669	8,640	8,690	8,491	8,473	8,723	3,725	2,437	2,426	1,563	1,080	3,817	2,035	1,578	106
	kt	646	661	510	960	960	960	959	967	960	967	982	426	275	275	174	122	451	249	193	13
Waste Development																					
Waste Development	m	6,453	6,712	4,893	8,788	8,900	9,298	9,112	9,059	8,685	8,847	9,154	4,032	2,983	2,720	1,625	1,080	4,034	2,035	1,578	106
	kt	659	678	517	965	968	974	970	975	964	975	991	433	287	281	176	122	456	249	193	13
Low Grade Development																					
Low Grade Development	m	4,904	4,712	4,080	-	-	-	-	-	-	-	-	4,190	5,119	4,091	2,436	1,915	-	-	-	417
	kt	554	539	449	-	-	-	-	-	-	-	-	478	581	486	294	231	-	-	-	50
Waste and Low Grade Development																					
Waste and Low Grade	m	11,357	11,425	8,973	8,788	8,900	9,298	9,112	9,059	8,685	8,847	9,154	8,222	8,102	6,811	4,061	2,995	4,034	2,035	1,578	524
	kt	1,213	1,216	966	965	968	974	970	975	964	975	991	911	868	767	469	352	456	249	193	63
Ore Development																					
Ore Development	m	4,223	4,317	3,459	3,467	4,371	3,346	2,771	2,307	3,011	4,214	2,317	2,664	2,451	2,902	2,909	3,157	2,406	2,720	1,481	833
	kt	464	471	378	380	483	367	303	250	332	464	253	297	275	325	328	351	262	295	163	88
Total Development																					
Total Development	m	15,580	15,742	12,433	12,255	13,271	12,644	11,883	1, 366	11,696	13,061	11,471	10,886	10,553	9,713	6,970	6,151	6,440	4,755	3,059	1,357
	kt	1,677	1,687	1,344	1,345	1,450	1,341	1,273	1,226	1,295	1,439	1,244	1,207	1,144	1,092	798	703	717	545	356	151

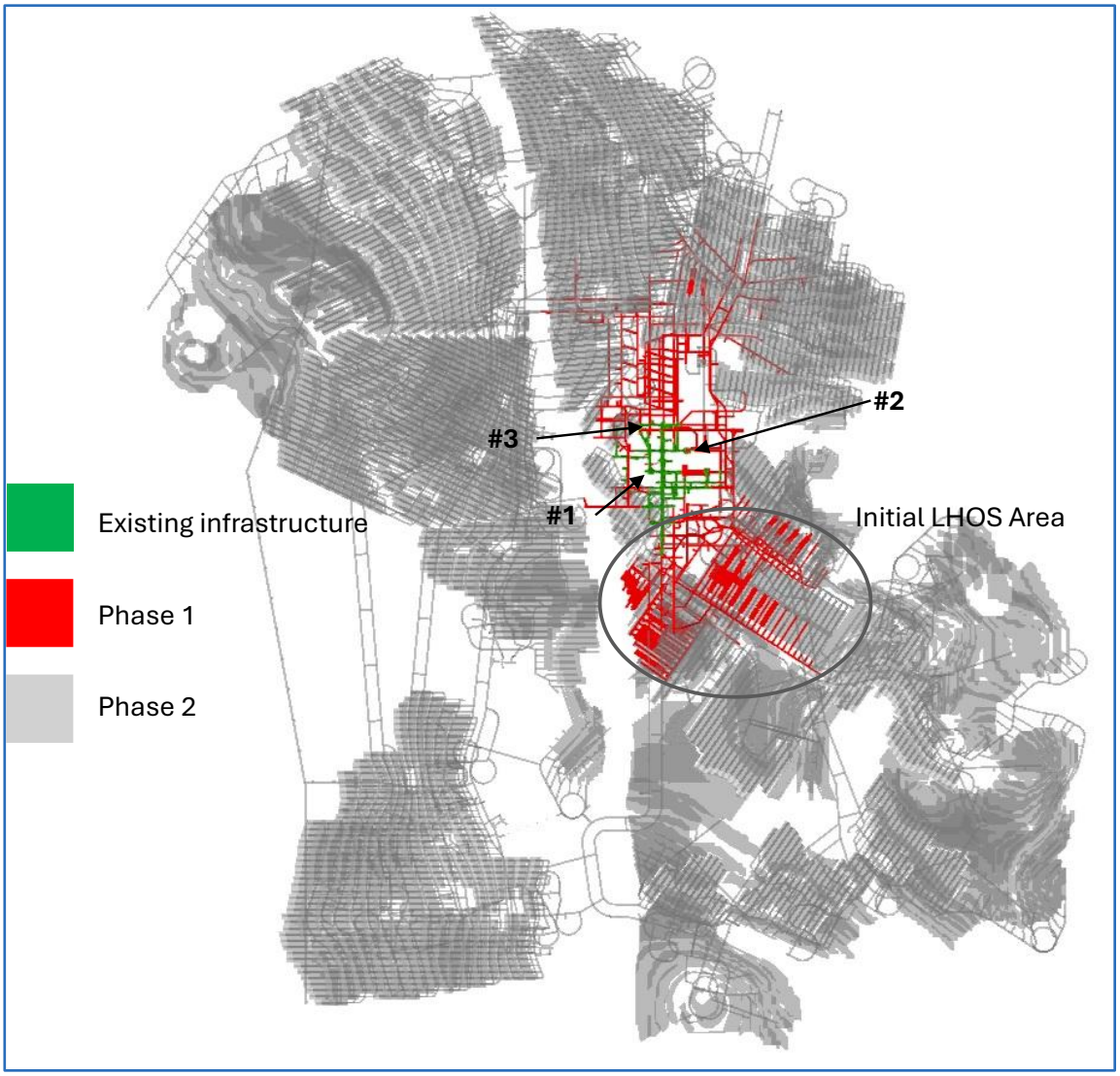
Table 16.33 Life-of-Mine Waste Development Summary

Description	Units	Total
Shafts		
Advance	km	4.6
Waste	kt	444
Lateral Development		
Advance	km	240
Waste	kt	27,197
Vertical Development		
Advance	km	16
Waste	kt	379
Low Grade Development		
Low Grade Advance	km	32
Low Grade Tonnage	kt	3,661
Total Waste Development (Waste+LG)		
Total Advance	km	293
Total Waste	kt	31,695

16.4.4 Production Planning Criteria

Zones close to the south of the shaft complex were identified and prioritised for initial and early development. The development and production timeline schematic of the Platreef 2025 FS is shown in Figure 16.30.

Figure 16.30 Platreef 2025 FS Phased Development Zones



16.4.4.1 Production Ramp Up Schedule

The Platreef 2025 FS Phase 1 has been designed with a concentrator capacity of 0.77 Mtpa. Phase 2 includes the construction of an additional concentrator module with a capacity of 3.3 Mtpa, to be built alongside the existing Phase 1 module. The mine is projected to ramp up to full production at a steady state of 4.1 Mtpa by the year 2030, with mine closure requirements anticipated in 2059.

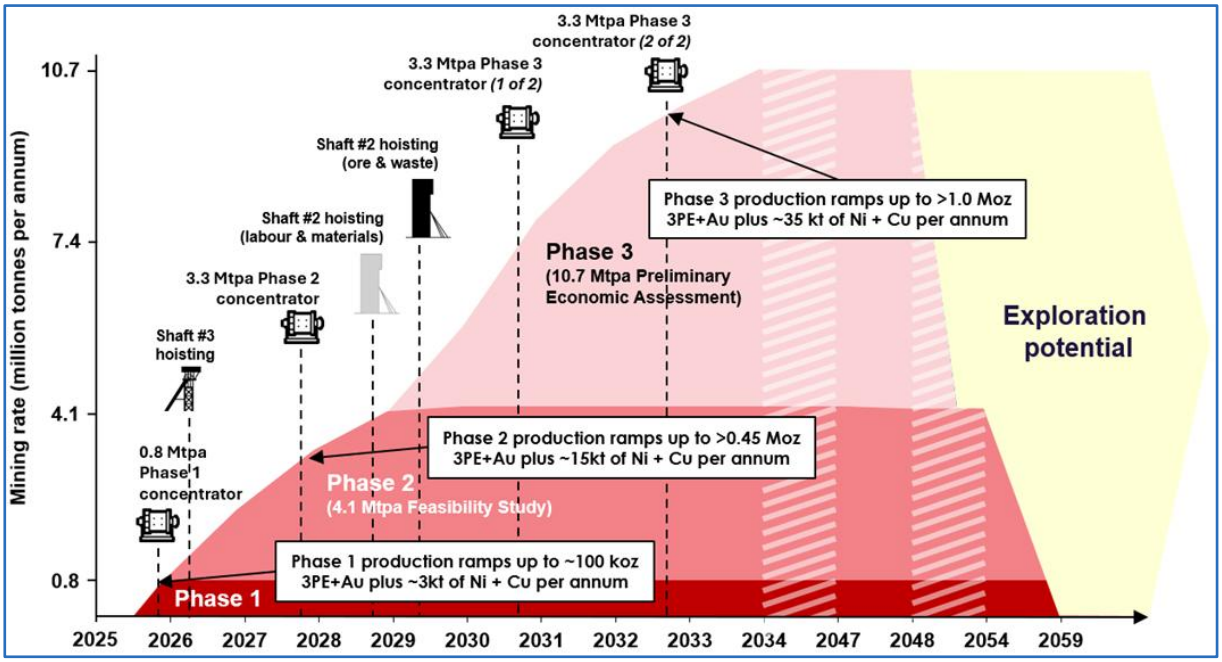
The Platreef 2025 FS will initially use Shaft 1 and Shaft 3 for hoisting ore and waste to feed the Phase 2 concentrator module. Shaft 2 is expected to be initially equipped for hoisting labour and materials from 2029, further increasing total hoisting capacity, providing significant operational flexibility. The key dates for the Platreef 2025 FS are summarised in Table 16.34.

Table 16.34 Platreef 2025 FS Key Dates

Activity Name	Start	Finish
Shaft 1 Commissioning		Q1'22
Restart Development from Shaft 1	Q2'22	
Shaft 3 Reaming Complete	Q1'23	Q4'24
Shaft 3 Equipping	Q4'24	Q1'26
Shaft 2 Sinking to –60 m Level	Q4'25	Q1'26
First Concentrator Start Up		Q4'25
Shaft 2 Sinking to –995 m Level	Q1'26	Q1'28
Shaft 2 Equipping Complete		Q3'28
Plant 2 Start Up		Q4'27
Mine Production Steady State (4.07 Mtpa)	Q4'29	

Figure 16.31 shows the production ramp up and key milestones for the Platreef 2025 FS.

Figure 16.31 Platreef 2025 FS Phased Development



16.4.4.2 Life-of-Mine Production Schedule

The first 10 years of production designed to maximise the production profile. During this period, constraints such as stope availability, ventilation requirements, and hoisting capacity were taken into consideration. Production Summary

Production is summarised in the following table and six figures. Table 16.35 shows the overall LOM production summary. Figure 16.32 and Figure 16.33 show the annual production rate along with the associated NSR based on BDT20 and 3PE+Au, respectively.

Figure 16.34, Figure 16.35, Table 16.36 and Table 16.37 show the annual production rate by mining method and the total production along with the associated NSR based on BDT20 and 3PE+Au, respectively. Figure 16.36 and Figure 16.37 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 16.36 and Figure 16.37.

Table 16.35 Life-of-Mine-Production Summary

Description	Units	Total
Drift and Fill	kt	19,580
Longhole Stopes	kt	88,242
Ore Development	kt	22,042
Total Ore	kt	129,863
Diluted Grades	–	–
NSR	$/t	142
Platinum	%	1.88
Palladium	%	1.93
Gold	g/t	0.29
Rhodium	g/t	0.13
Copper	g/t	0.16
Nickel	g/t	0.33
Sulfur	%	0.80
3PE+Au	g/t	4.22
Max Daily Production Rate	t/d	11,361
Low Grade Stockpile	kt	3,661
Low Grade Stockpile NSR	$/t	41
Waste (excl. Low Grade)	kt	28,034

Figure 16.32 Annual Production with NSR

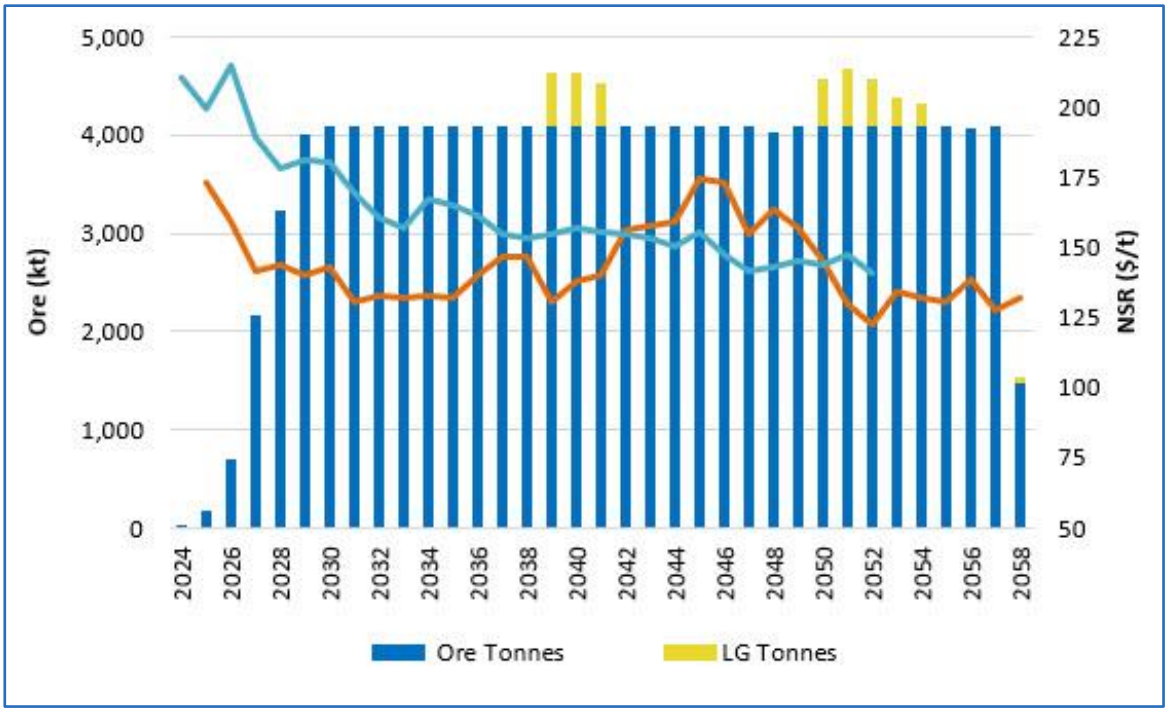


Figure 16.33 Annual Production with 3PE+Au

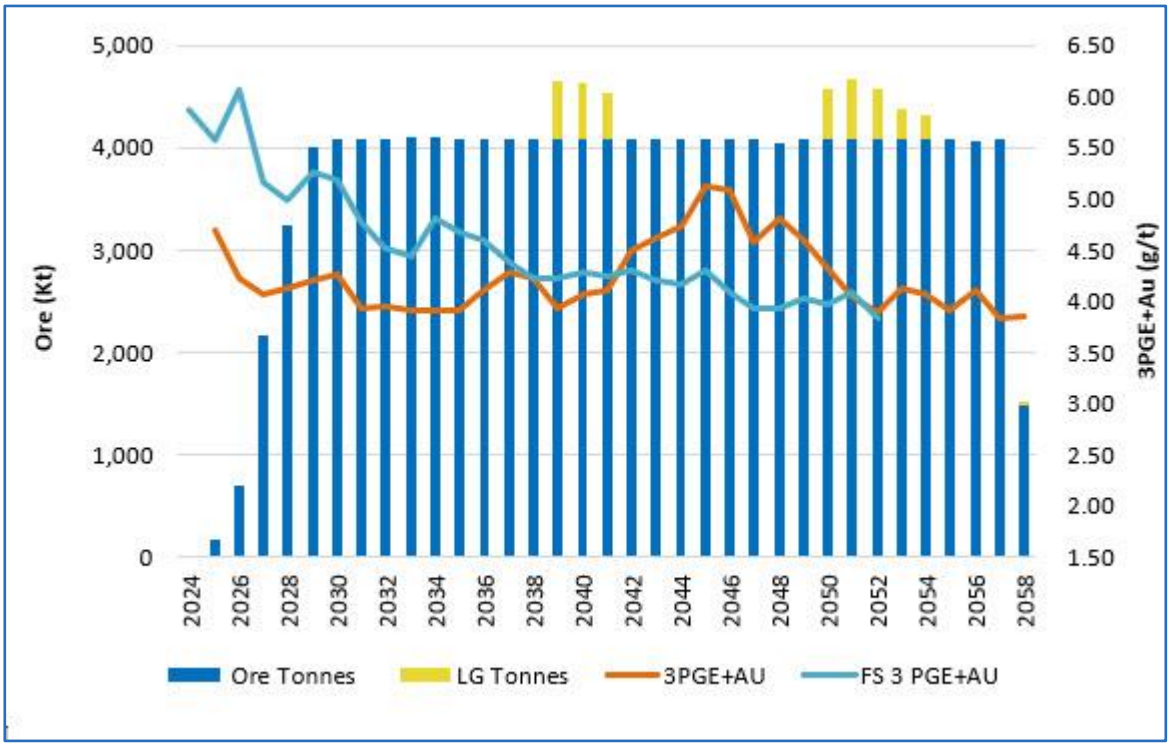


Figure 16.34 Annual Production by Mining Method with NSR

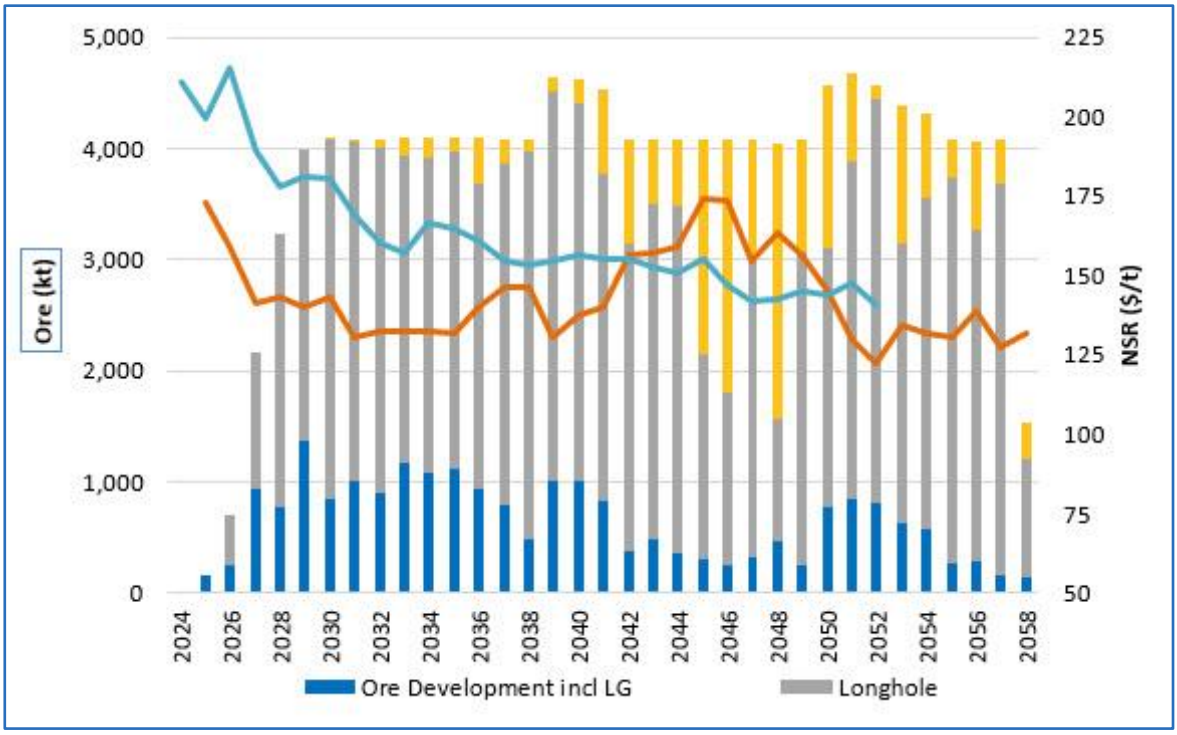


Figure 16.35 Annual Production by Mining Method with 3PE+Au

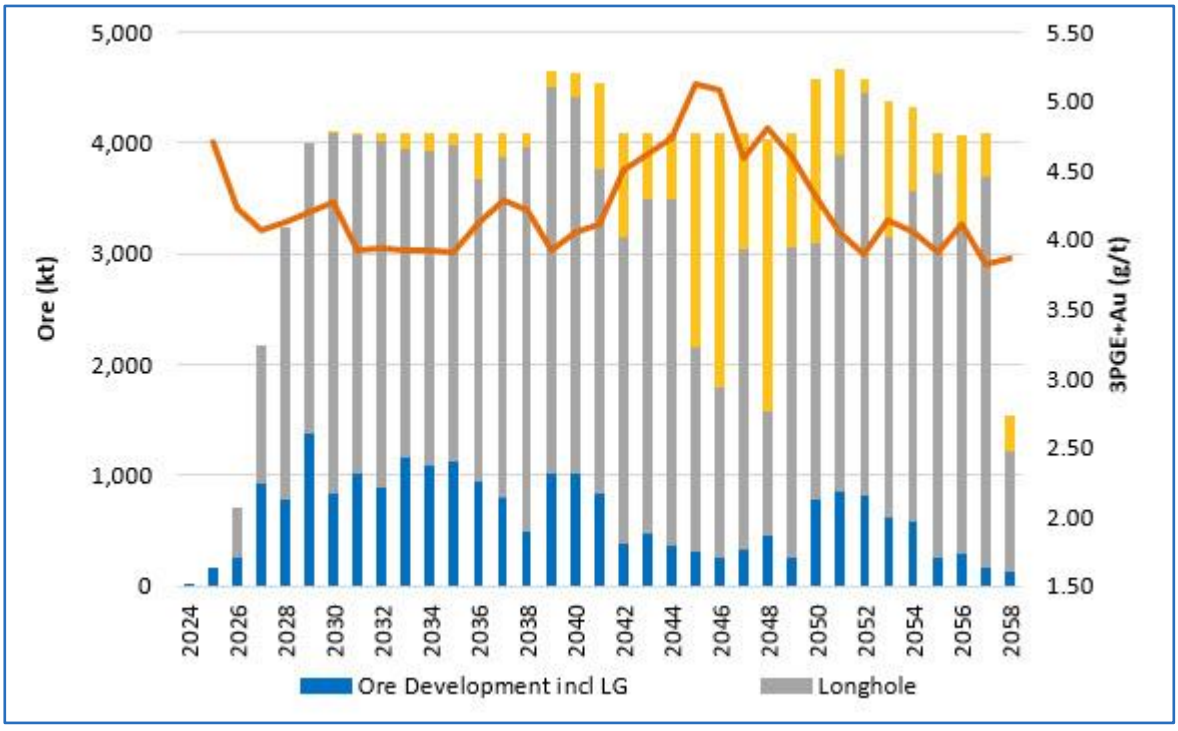


Figure 16.36 Annual Production by Rock Type with NSR

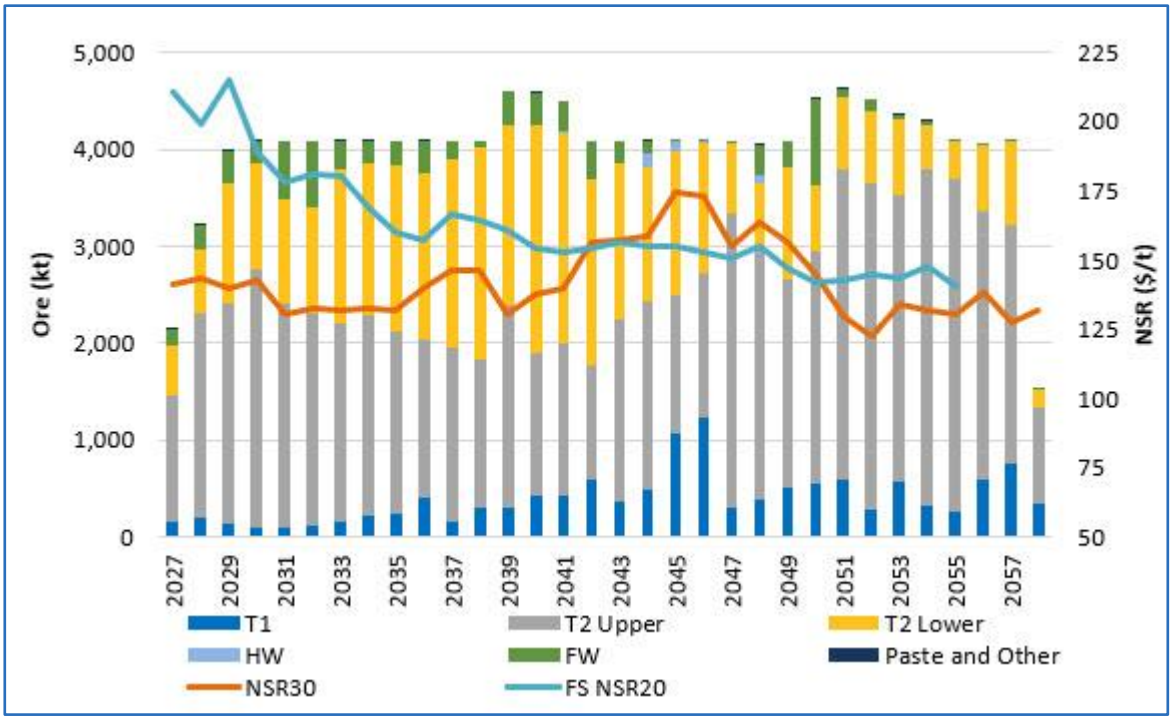


Figure 16.37 Annual Production by Rock Type with 3PE+Au

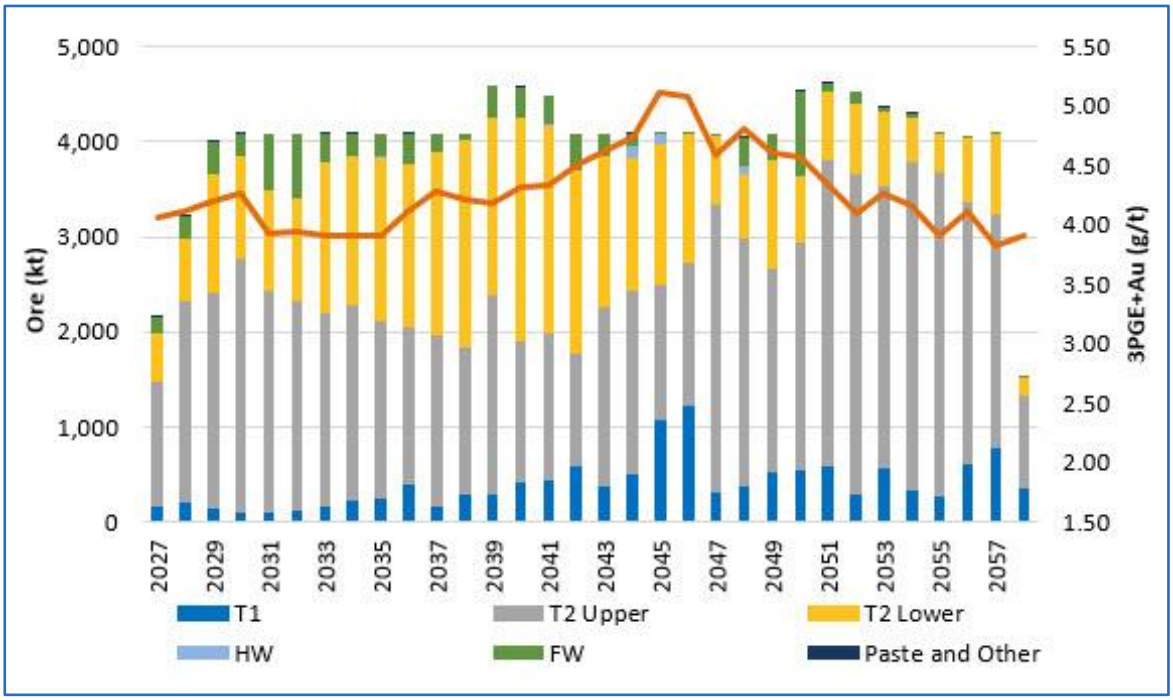


Table 16.36 Mine Production by Method

	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Ore Development	**kt**	**21,921**	**172**	**250**	**866**	**764**	**1,357**	**830**	**1,008**	**894**	**1,167**	**8,143**	**6,471**
NSR	$/t	110	167	140	122	105	111	108	98	106	109	112	108
Platinum	g/t	1.64	2.12	1.81	1.65	1.54	1.60	1.53	1.41	1.49	1.51	1.66	1.72
Palladium	g/t	1.73	2.06	1.78	1.72	1.66	1.72	1.67	1.57	1.63	1.64	1.76	1.75
Gold	g/t	0.26	0.33	0.31	0.27	0.24	0.24	0.25	0.23	0.25	0.24	0.25	0.26
Rhodium	g/t	0.11	0.13	0.11	0.11	0.10	0.11	0.10	0.10	0.10	0.10	0.12	0.12
Copper	% Cu	0.15	0.19	0.19	0.17	0.16	0.15	0.15	0.15	0.16	0.16	0.15	0.15
Nickel	% Ni	0.31	0.37	0.37	0.33	0.31	0.31	0.31	0.30	0.32	0.32	0.32	0.31
Sulfur	% S	0.78	0.95	0.97	0.89	0.79	0.75	0.72	0.78	0.83	0.82	0.78	0.75
3PE+Au	g/t	3.74	4.65	4.01	3.75	3.55	3.66	3.56	3.31	3.48	3.49	3.79	3.85
Longhole	**kt**	**88,242**	**–**	**454**	**1,238**	**2,455**	**2,626**	**3,245**	**3,063**	**3,122**	**2,776**	**30,592**	**38,672**
NSR	$/t	141	–	166	155	156	155	152	141	138	139	142	137
Platinum	g/t	1.85	–	1.98	1.91	1.92	1.98	1.98	1.76	1.73	1.75	1.80	1.88
Palladium	g/t	1.92	–	2.01	1.93	1.96	2.09	2.06	1.95	1.90	1.89	1.93	1.89
Gold	g/t	0.28	–	0.30	0.32	0.31	0.28	0.28	0.28	0.30	0.27	0.28	0.28
Rhodium	g/t	0.13	–	0.13	0.12	0.12	0.13	0.13	0.12	0.12	0.12	0.13	0.13
Copper	% Cu	0.16	–	0.20	0.19	0.18	0.17	0.17	0.16	0.16	0.16	0.16	0.15
Nickel	% Ni	0.32	–	0.39	0.37	0.37	0.34	0.34	0.33	0.33	0.33	0.34	0.31
Sulfur	% S	0.79	–	1.04	0.96	0.95	0.84	0.81	0.85	0.87	0.83	0.81	0.73
3PE+Au	g/t	4.18	–	4.43	4.29	4.31	4.49	4.45	4.11	4.04	4.03	4.14	4.18
Drift and Fill	**kt**	**19,580**	**–**	**–**	**–**	**–**	**–**	**3**	**18**	**71**	**152**	**3,708**	**15,628**
NSR	$/t	176	–	–	–	–	–	148	137	179	178	183	174
Platinum	g/t	2.26	–	–	–	–	–	1.81	1.76	2.05	2.14	2.28	2.25
Palladium	g/t	2.20	–	–	–	–	–	2.01	1.84	2.30	2.31	2.32	2.17
Gold	g/t	0.37	–	–	–	–	–	0.35	0.30	0.31	0.32	0.33	0.38
Rhodium	g/t	0.15	–	–	–	–	–	0.10	0.10	0.13	0.14	0.16	0.15
Copper	% Cu	0.18	–	–	–	–	–	0.18	0.15	0.18	0.17	0.19	0.18
Nickel	% Ni	0.36	–	–	–	–	–	0.32	0.32	0.40	0.38	0.37	0.36
Sulfur	% S	0.90	–	–	–	–	–	0.62	0.50	0.85	0.85	0.93	0.89
3PE+Au	g/t	4.97	–	–	–	–	–	4.28	3.99	4.78	4.91	5.09	4.95

Table 16.37 Total Mine Production

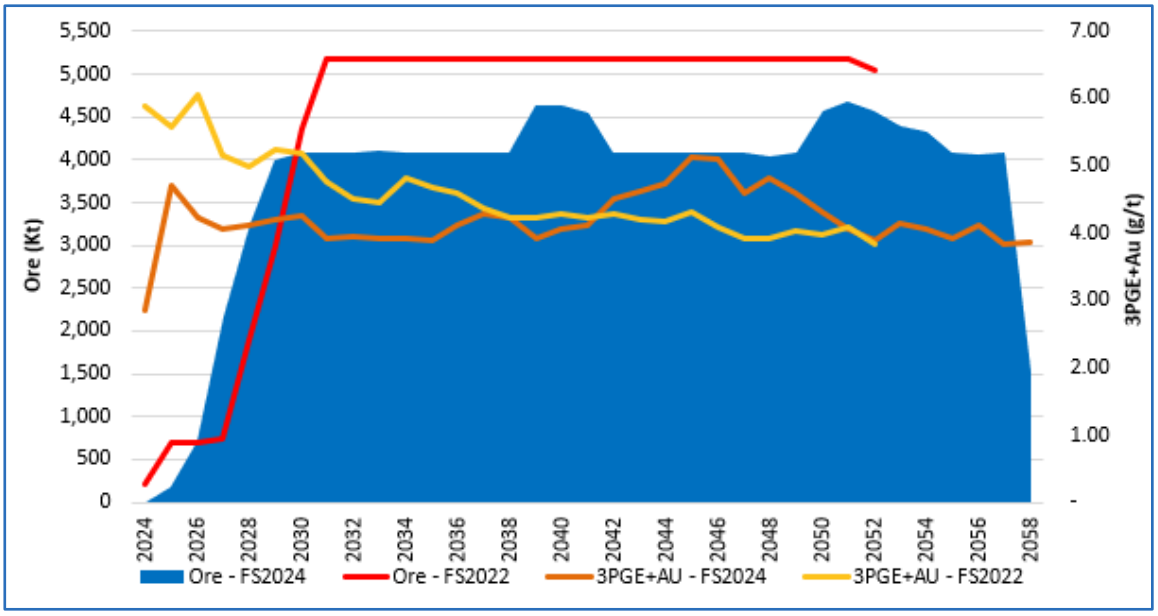
	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Total Ore Mined	**kt**	**129,742**	**172**	**704**	**2,104**	**3,218**	**3,983**	**4,078**	**4,089**	**4,086**	**4,095**	**42,443**	**60,770**
NSR	$/t	141	167	157	141	144	140	143	130	132	132	140	143
Platinum	g/t	1.88	2.12	1.92	1.80	1.83	1.85	1.89	1.67	1.68	1.70	1.82	1.96
Palladium	g/t	1.93	2.06	1.93	1.85	1.89	1.96	1.98	1.85	1.85	1.84	1.93	1.94
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.26	0.27	0.27	0.29	0.26	0.28	0.30
Rhodium	g/t	0.13	0.13	0.12	0.11	0.12	0.13	0.13	0.12	0.11	0.12	0.13	0.14
Copper	% Cu	0.16	0.19	0.19	0.18	0.18	0.16	0.16	0.16	0.16	0.16	0.16	0.16
Nickel	% Ni	0.33	0.37	0.38	0.35	0.36	0.33	0.33	0.32	0.33	0.33	0.34	0.32
Sulfur	% S	0.80	0.95	1.02	0.93	0.91	0.81	0.79	0.83	0.86	0.83	0.82	0.77
3PE+Au	g/t	4.22	4.65	4.28	4.07	4.13	4.20	4.27	3.91	3.93	3.91	4.15	4.34

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 2025 FS economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.

16.5 Development and Production Compared to the Platreef 2022 FS

The development and production schedules of the Platreef 2025 FS were compared to the Platreef 2022 FS and the results are shown in Figure 16.38.

Figure 16.38 Ore Tonnes and 3PE+Au Grades



16.6 Underground Infrastructure

The underground infrastructure includes components, such as ore and waste handling systems, dewatering, maintenance shops, fuelling, ventilation, etc. This Section includes ore and waste handling down to the crushers, ventilation and refrigeration, and production return water.

16.6.1 Mine Dewatering

Mine production return water will include drill water, mine service water, fissure water, backfill flush water, and backfill seepage. The dewatering system for Platreef can be separated into the following components, starting from active development and stoping ends back to the shaft:

- Face dewatering
- Secondary dewatering
- Primary dewatering
- Each system feeds the water back to the next.

16.6.1.1 Mine Dewatering: Water Inflows

The dewatering capacity of the mine is designed to manage all water inflows to the underground operations, including fissure water, mine service water, flushing water, and any potential seepage. The values for water inflow or fissure water used to determine the dewatering requirements were sourced from WSP in Africa (WSP). These values increase progressively as the mine develops. For pumping calculations, a conservative scenario was applied.

The mine service water usage was calculated at 0.19 m³ per tonne of rock mined based on the different mining methods and the blend thereof (Development, drift and fill and longhole stoping). This figure was applied to the mine schedule to forecast the expected service water usage over time. Flushing water for the paste fill system was conservatively estimated based on recommendations from Paterson & Cooke. Seepage from the backfill system is not anticipated.

The peak daily requirement for service water is projected to be 4,050 kl per day, with the majority of this consumption occurring over a 17 hour period.

16.6.1.2 Face Dewatering and Secondary Dewatering

Electric face dewatering pumps and vertical spindle pumps will pump production return water, along with any fissure and flushing water inflows on each sub-level to a system of gravity transfer sumps (GTS) or transfer pump stations (TPS). In general, water above and below 950 m Level is fed to the TPSs located on 950 m Level. The TPSs on 950 m Level, in turn, pump this water to the main pump station also located on 950 m Level.

The return water system is vital for mining operations, as failures can cause risks like operational halts and evacuations. Careful design considerations mitigate these risks by addressing pumping capacities, storage needs, pumping arrangements, settling system specifications, and water column design.

Gravity Transfer System (GTS)

The GTS transfers production return water from various mining levels, typically located above 950 m Level, to the closest transfer pump station by means of gravity. These GTS can be linked into a cascading type of system where required. The drain hole is slightly elevated in the cubby to allow grit buildup in the sump's lower part, which can be cleared out periodically by LHD.

Transfer Pump Station

Transfer pump stations are generally positioned in lower areas where water cannot be transferred by gravity alone to the lower parts of the mine. Each pump station will require a unique capacity design depending on its specific location, accommodating the fissure water extraction area as determined by WSP. Additionally, if the pump station is part of a cascade system, these differences must be considered.

If there is an overflow, water can exit the pump station and flow into the access drift or ramp instead of flooding the pumps. This action transfers the risk to the lowest pump station in the mine and minimises equipment risk.

A removable barrier has been placed in the dam to allow grit to settle on the inflow side. Overflow water will flow to the clear water section of the dam, with most of the grit remaining behind the removable wall for an LHD to clean out. The dam's dimensions are designed to permit a loader to access all sections for removing the settled grit.

In the event of a power failure, dams are capable of receiving water for up to four hours before overflowing, assuming that the dam's capacity was at 70% at the time of the failure. The dam has a capacity of approximately 0.6 ML.

Two submersible pumps, one in operation and one on standby are placed in a sump in the clear water section of the dam. These pumps transfer water to a suction tank on the pump station level, from where two trains of centrifugal pumps, one operating and one standby, draw water from the suction tank and transfer it to the next pump station. The suction tank ensures an adequate net positive suction head (NPSH) for the centrifugal pump trains. Both the submersible pumps and the centrifugal pump trains are designed to handle dirty water, as not all grit will be settled out in the dirty water section of the dam.

An electrical overhead travelling crane is installed in the main pump chamber for maintenance. Additionally, an overhanging crawl beam with hoists is provided for servicing and replacing submersible pumps.

The pump stations on 950 m Level need to deal with the largest volume of water as water from the levels above drain to this level and the water generated below 950 m Level is pumped back up to 950 m Level. The pumps below 950 m Level are, in turn, required to deal with the largest head. A common pump has been selected, and by varying motor installed power and v-belt pulley ratios, required duties can achieved. Where larger flows are required, pump trains will be connected in parallel; where larger heads are required, pumps will be connected in series. Table 16.38 below indicates the pump combinations for the various pump stations.

Table 16.38 Transfer Pump Stations Pump Requirements

Pump No.	Description	Qty Total	Qty Duty	Qty Standby	Pump Train Config.	Flow Rate Q l/s	Flow Rate Reqd. Q l/s	kW Installed
1	850 m Level North East Transfer Pump Station – Level 850 (Block C)	2	1	1	1 in series	5.60	3.77	7.5
2	850 m Level South Central Transfer Pump Station - Level 850 (Block G)	2	1	1	1 in series	9.30	6.21	18.5
3	850 m Level South East Transfer Pump Station - Level 850 (Block G)	4	2	2	2 in series	12.10	8.09	18.5
4	950 m Level Centre Transfer Pump Station - Level 950 (Block D)	4	2	2	2 in parallel	40.70	27.16	15.0
5	950 m Level South Transfer Pump Station - Level 950 (Block E)	4	2	2	2 in parallel	31.80	21.19	18.5
6	950 m Level North Transfer Pump Station - Level 950 (Block B)	4	2	2	2 in parallel	34.10	22.72	7.5
7	1,050 m Level Centre Transfer Pump Station – Level 1,050 (Block D)	6	3	3	3 in series	12.10	8.07	15.0
8	1,050 m Level South Transfer Pump Station - Level 1,050 (Block E)	6	3	3	3 in series	10.30	6.85	15.0
9	1,050 m Level North Transfer Pump Station – Level 1,050 (Block A)	6	3	3	3 in series	9.60	6.37	15.0
10	1,150 m Level South Transfer Pump Station – Level 1,150 (Block E)	6	3	3	3 in series	7.70	5.12	7.5
11	1,150 m Level North Transfer Pump Station – Level 1,150 (Block A)	6	3	3	3 in series	7.90	5.28	15.0

16.6.1.3 Primary Dewatering System

The primary dewatering system handles feed from the secondary pumping system and shaft activities. It includes a degritting station, storage dams, pumps, dewatering pipes via Shaft 1, then Shaft 3, and a surface settling system. Interim (Via Shaft 1) and Phase 2 upgrade (via Shaft 3) will align pumping capacity with production ramp up.

Interim Dewatering (through Shaft 1)

An interim pumping system has been installed in Shaft 1 for the initial development stages before the commissioning of Shaft 3 and the main pumping station.

Mine return water will be collected in two dirty water collection dams located on the 950 m Level. These dams are equipped with a dewatering screen designed to remove a significant amount of grit before the water is pumped to the relay pump station on the 450 m Level. The 450 m Level station will then pump the water directly to the surface into one of two high-rate settlers.

Four Scamont SP600 positive displacement pumps were purchased with two installed as duty/standby units at 950 m Level and 450 m Level pump stations. The pumping flow rate from the system is 67 m³/h of dirty water (based on pump duties).

Phase 2 Life-of-Mine Dewatering Infrastructure

950 m Level Main Pump Station

The main pump station will be further upgraded with eight Scamont SP200 positive displacement (seven duty and one standby) and three Scamont GSB150 self-balancing multi-stage (two duty and one standby) pumps. These will be installed in two phases; based on the current mining profile, the SP200s will be required by the end of 2026, and the GSB150s will be required by the end of 2028. The interim pumping system will be decommissioned thereafter.

The pumps will be installed in line with sufficient clearance to remove a single pump for repairs or replacement.

As per the recommendation of the pump supplier, Scamont Engineering, separate suction lines are provided for the positive displacement pumps and multi-stage centrifugal pumps. Each suction line has strainers and flush points. Similarly, the common delivery lines are provided with flushing points and drains back to the dams.

The positive displacement pumps have a start-up line that dumps water back to the dams before switching over to the main pump column. In addition, the pumps require flush water for the pistons. Potable water is piped to each pump, and the flush water falls into a drain routed to a sump.

An electric submersible pump is installed in a sump near the main pumps to collect and pump spillage and flush water back to one of the main dams.

The pump station is equipped with an electric overhead travelling crane for ease of installation and maintenance of the system.

950 m Level De-Gritting Station

Water pumped from the 950 m Level transfer pump stations reports to the de-gritting screen located north of the main dams. Oversize grit >0.5 mm is removed and deposited over a small wall for removal by a skid-steer-loader (Bobcat) or similar machine. The dirty water flows via gravity through a suitably sized pipe (to avoid settlement) to the main dams. A small sump is allowed for the collection of any spoilage water. This is pumped back to the screen.

Since most of the grit will be settled out prior to this screen, during the detailed design, it is recommended that a fixed linear screen to replace this system be investigated. This will reduce maintenance, cost and complexity.

Surface Settlers

Settling of the dirty water pumped from underground is required. A high-rate settling facility will be provided on surface. The water will not come into contact with the earth, negating the environmental concerns typically associated with settling ponds. This also means a smaller footprint is required.

Water is pumped from underground into the settler's launders. Here, the dirty water is dosed with lime and flocculant. Lime will reduce the potential of hydrogen (pH) of the water, and flocculant will affect coagulation of the suspended solids to promote settling. The clearwater overflow will report to the mine service water tank, where it will be re-circulated to underground. The underflow will report to an agitated mud tank. This tank will supply a filter press, where the grit will be removed from the water and discharged as mud cakes.

16.7 Ore and Waste Handling Systems

The ore and waste handling system will be constructed in distinct phases:

- Current Shaft 1: Relies on load haul dumps (LHDs) tipping directly into the shaft.
- Shaft 3 System Initial: Will introduce a total of four truck tips (2 x on 750 m Level and 2 x 850 m Level) with a crushing station on 950 m Level and a conveyor system feeding into a bin and loading system at Shaft 3.
- Shaft 3 System Future: The system described above will remain in operation, and additional conveyors and crushing stations will be added on 910 m Level, 950 m Level and 1,050 m Level.

16.7.1 Current Shaft 1 Rock Handling System

Blasted material will be dropped to the 950 m Level through ore passes at 750 m Level and 850 m Level waste pass, 850 m Level ore pass, and 750 m Level ore pass. A 14-tonne loader will collect the material from the bottom and tip it directly into the LHD tip at Shaft 1. Rock blasted on or below 950 m Level will be loaded directly into the LHD tip at Shaft 1 or stored in a muck bay near the shaft for later loading. This system allows for the separate handling of ore and waste by allocating the available passes accordingly. The entire system can be quickly switched between ore and waste, requiring only the repositioning of the radial stacking conveyor on the surface.

16.7.2 Shaft 3 Rock Handling System: Initial

The initial rock handling system for Shaft 3 is currently well into the execution phase. This system includes two truck tipping points on 750 m Level, two truck tipping points on 850 m Level, a crushing station on 950 m Level, and a conveyor system feeding into the loading system of Shaft 3.

The silos are vital for providing sufficient storage to ensure that Shaft 3 operates at full capacity. During each shift, trucks and LHDs will fill the silos, and during shift changes, the conveyor system will continue to operate to ensure that all bins are almost emptied before the next shift begins.

After the blasted rocks are resized by the grizzly and rock breakers, the material will be further reduced in size by jaw crushers installed below each silo. At the bottom of each silo, an apron feeder draws material from the silo and feeds it onto a sacrificial conveyor belt equipped with a tramp iron magnet. The tramp material is deposited into a chute and lands in a bin or cassette, while the rock is fed onto a vibrating grizzly feeder (VGF). The undersized material falls onto a conveyor belt below, and the oversized material is directed into the jaw crusher. The resized material joins the finer material on a common conveyor belt beneath each crusher.

A three-conveyor system will transport the material into a 50 t feed bin installed at Shaft 3, which will have an apron feeder and conveyor feeding into a loading flask. The material in the loading flask will discharge into a 19 t skip and be hoisted to the surface.

16.7.3 Shaft 3 Rock Handling System: Future Overview

The mine plan initially focuses on material and areas on 850 m Level and above, and overtime progresses to the lower areas on 950 m Level and below. As mining continues to go deeper, a future rock handling system will be constructed that will establish additional truck tips and crushers in the deeper parts of the mine. In total, three crusher systems will be constructed with additional truck tips on 850 m Level, 750 m Level, 2 x on 950 m Level and 2 x on 1,050 m Level.

Ore from mining areas is truck hauled from internal ore passes to the truck tips mentioned above with 50 tonne trucks. Ore feeds into one of two coarse ore silos. At the bottom of each silo, an apron feeder will transfer the material onto a short conveyor (sacrificial conveyor) equipped with a head-end magnet. The rock will then be transported onto a VGF, where oversized material will be directed into a crusher. The crusher, which accepts input from both sacrificial conveyors coming from each silo, will discharge the crushed material directly into a fine ore silo. The undersized material from the VGF will flow via a chute into the same fine ore silo. The bottom of the fine ore silo will be equipped with an apron feeder that will transfer the crushed material onto a sacrificial conveyor and then onto the main conveyor system, which connects each crusher system to Shaft 3.

Based on the input parameters and results from the simulation, the crusher selected for this duty is an Astec 30 x 55 single toggle jaw crusher. It is equipped with a 160 kW motor, hydraulic gap adjustment and auto-lubrication system.

16.7.4 Conveyors

All the designs of the current Shaft 3 execution conveyors, as well as the newly added conveyor systems connecting the new crushing systems to the current conveyors, have been rationalised to ensure the commonality of equipment (drives, pulleys, idlers, scrapers and belting) as far as is practically possible.

Table 16.39 is a summary of the conveyor schedule indicating the new conveyors. This includes the “sacrificial” crusher feed conveyors and “sacrificial” conveyors under the crusher systems, as well as the main dip conveyors.

Table 16.39 Conveyor Schedule

Conveyor Number	Conveyor Description	Belt Width (mm)	Length (m)	Vert Lift (m)	Capacity (tph)	Speed (m/s)	Belt Class
2622-CV-013	Conveyor No. 2B (Silo Feed Conveyor)	1350	837	124	1250	2.0	ST2000
2622-CV-014	Conveyor No. 4 (Bottom of 950 m Level Crusher)	1350	344	42	1250	2.0	1250/4
2622-CV-015	Conveyor No. 5 (Conveyor 2 from 1,050 Crusher)	1350	520	75	1250	2.0	ST2000
2622-CV-016	Conveyor No. 6 (Conveyor from 1050 Crusher)	1350	352	42	1250	2.0	1250/4
2622-CV-017	Crusher Feed Conveyor (North) 910 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-018	Crusher Feed Conveyor (South) 910 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-019	Crusher Feed Conveyor (North) 950 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-020	Crusher Feed Conveyor (South) 950 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-021	Crusher Feed Conveyor (North) 1,050 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-022	Crusher Feed Conveyor (South) 1,050 m Level	2100	14	0	1250	0.8	1000/4
2622-CV-023	910 m Level Crusher Sacrificial Conveyor	1350	36	3	1250	1.6	1000/4
2622-CV-024	950 m Level Crusher Sacrificial Conveyor	1350	36	3	1250	1.6	1000/4
2622-CV-025	1,050 m Level Crusher Sacrificial Conveyor	1350	36	3	1250	1.6	1000/4
2622-CV-011	Feed Conveyor - Shaft 3 Existing System	1350	35	5.4	750	1.5	1000/4

16.7.5 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 and be directed to the correct off-loading area.

On request, the material control system will load spares, consumables, or other materials into cassettes according to the area and section for delivery to underground sections. Due to the size limitation of Shaft 1, custom-designed cassettes are required to optimise available cage space. Additionally, converting a single-deck cage to a double-deck cage is necessary to transport longer materials.

Once Shaft 2 is operational, the utility vehicle cassettes will be utilised for material transport. The cassettes 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" and "empty cassettes" areas close to the shaft bank.

16.7.5.1 Underground Material Handling System

Shaft 1 cage will be changed out with a double deck cage once Shaft 3 is in operation to facilitate higher capacity loads to the production levels. Material, if exceeding the rope's weight limit, can be slung down the shaft without the cage. Material handling involves slinging, palletizing, and loading in tankers or containers. Large mobile equipment is disassembled, slung down the shaft, and reassembled in assembly bays on different levels. A multi-purpose vehicle with a forklift attachment loads material into the cage, and overhead crawls with chain hoists assist in handling large equipment in assembly bays.

Once Shaft 2 is commissioned, mobile equipment will be able to drive in and out of the cage, loading full cassettes directly into the cage. Cassettes are removed from the cage by a forklift, and underground utility vehicles transport them to working areas.

16.7.5.2 Explosives Handling and Distribution

Explosives are critical for underground mining, and the type, delivery, and storage of the same require special design considerations. The underground mine will use the two-component emulsion system, which consists of a base product and a sensitizer that is combined at the face.

Explosives are classified according to their respective hazard classification type.

- Explosive 1.1B: Non-electric detonators, electronic detonators, or any type of blasting cap.
- Explosive 1.1D: Water-gel explosives, ammonium nitrate fuel oil (ANFO), cast boosters, detonating cord.
- Oxidizer 5.1: Oxidizers and base emulsion.

Each type of explosive will be transported underground separately and via different methods. Class 1 explosives, includes Explosive 1.1B and 1.1D, will be transported underground, making use of purpose-built explosives cassettes. Base emulsion will be piped down from surface into storage tanks underground and further distributed via emulsion cassettes. However, initially, the emulsion will be transported underground using explosive cassettes that are purpose-built to fit into the cage. These cassettes will be handled by a forklift in and out of the cage and loaded onto utility vehicles for further distribution to working areas.

Class 1 Explosives

Delivery of Class 1 explosives is assumed to be twice per week or, according to discussions with African Explosives Limited (AEL), packaged explosives can be delivered to site with 72 hours' notice, if required. Surface delivery of Class 1 explosives will be to the surface explosives transfer shed. Once the pallets are offloaded on surface, reloading for transport underground will start immediately.

Currently, there is a surface magazine for detonators and/or ancillary explosives located approximately 500 m away from the shaft. The surface explosives facility will be replaced by an explosive transfer shed that will offload delivered accessories directly into explosives utility vehicles and transported underground. The offloading facilities on surface will be completely fenced off with lockable gates, warning signs, lights and fire extinguishers according to best practices and similar to other operating mines while also adhering to the explosives act requirements.

Three magazines will be located underground on 750 m Level, 850 m Level and 950 m Level. Each magazine will have two separate explosives bays, one for detonators and one for package explosives. The recommendation for only two magazines is made to reduce unrestricted access by unauthorised personnel, simplify inventory control and product issuance, and improve the security capabilities of a limited number of explosives storage areas.

Each of the Class 1 explosives bays has been designed with a storage capacity of 25 pallets. The Class 1 explosives bays will be of equal size and depth.

Bulk Emulsion and Sensitizer: Current System

Currently, the emulsion and sensitizer are stored in tanks on surface located near the shaft. Emulsion trucks will deliver emulsion every second day; however, sufficient storage capacity for four days is available.

Emulsion Vertical Drop System

Emulsion and sensitizer are delivered to site via bulk tanker truck to the 50 m³ emulsion silo. Emulsion is transferred underground at a controlled rate using a pump, whereas sensitizer and water are transferred under pure gravity. Loading of the mobile charging unit is achieved via gravity discharge. The charging unit then drives to the required mining area.

16.7.6 Mine Services

Services supplied to underground workings are outlined in this section, including fire water, service water, and potable water.

16.7.6.1 Compressed Air

Compressed air will not be distributed throughout the mine due to this being prone to leakages and general wastage.

All production equipment is equipped with onboard compressors, where required, and will not require additional compressed air for drilling, flushing or spraying.

16.7.6.2 Underground Firewater System

Firewater will be stored on surface and distributed to underground through Shaft 3 by means of a 150NB column. The fire system is based on the fire protection design criteria.

Firewater Tanks

Firewater will be drawn from a dedicated source on surface and gravity-fed to the underground reticulation system, ensuring the availability of 675 m³ at all times. The reservoir will be constructed with an internal division, sub-dividing the tank into two equal sections, thus guaranteeing at least 50% availability at all times during maintenance and/or mechanical damage. 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 tanks have been designed to supply dedicated firewater for a duration of 90 minutes at maximum flow.

Firewater Reticulation System

Firewater will be gravity-fed from surface via a South African National Standards (SANS) 62 MED WT galvanised pipeline flanged to SANS 1123 class 1600/3 flanges. This line will tie into the dedicated fire water column in Shaft 3.

Firewater is provided primarily for the protection of the conveyor belting, and pressure will be reduced to the required design levels to provide adequate suppression at required duties.

Firewater pipework reticulation on the levels (post pressure reduction) will be via SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be class 16. All isolation/section valves will be Underwriters Laboratories (UL) listed/Factory Mutual (FM) approved.

Fire deluge zones for the conveyor system will be fed off the fire water column. Similarly, 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 8 m/s through any pipe and 6 m/s through any valve. Cognisance has been taken to ensure compliance with these limitations when sizing the ring main.

Fire Detection in Sub-Stations/Motor Control Centres

All sub-stations and motor control centres (MCCs) have a smoke detection system in the room and 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 conditions 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 con system etc.).

Fire Detection on Conveyors

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.

Types of Fire Suppression Systems

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 without restricting the operation.

16.7.6.3 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 LOM. 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. Reticulation on the levels, post pressure reduction, will be via high-density polyethylene (HDPE) piping.

Service Water

Mine return water is pumped to surface via Shaft 3 into a high-rate settling plant, the water is then pumped to a holding tank. This water is then gravity-fed back underground via Shaft 3, through a 150NB (ASTM A106 Grade B, Schedule 80) shaft pipe column, to the respective production levels. Reticulation on the levels, post pressure reduction, will be via HDPE piping.

Service water consumption includes drilling, wash down and dust suppression. Allowances were made for water loss in seepage, ventilation and water in rock. The water requirement was calculated from first principles to obtain a volume required per tonne of rock mined. This was calculated at 0.19 m³ of service water per tonne of rock mined. This number was applied to the mine schedule to calculate the service water expected to be used over time.

16.7.7 Underground Infrastructure

Underground facilities and associated infrastructure with applicable surface facilities support are addressed below.

16.7.7.1 Diesel Fuel and Lubricants Distribution

The mine uses diesel-powered mobile equipment, requiring a system to supply diesel to surface and underground equipment. Diesel will be stored in bulk tanks at two surface areas: one near the emergency generators and another beside the vehicle workshop. A refuelling station is included in the surface storage facility.

Underground vehicles will refuel at stations on the 750 m, 850 m, and 950 m Levels. Diesel, transmission oil, and engine oil will be piped to these levels via Shaft 1, while other lubricants will be transported in cassettes. The 750 m Level will have one refuelling bay due to lower production and fewer vehicles, while the 850 m and 950 m Levels will each have two bays.

Currently fuel is distributed from a surface storage tank near the bank, feeding pipes with energy dissipaters and temporary fuelling systems installed on 750 m and 850 m Levels. A permanent system will extend this setup, with refuelling at the main workshops on the 750 m, 850 m, and 950 m Levels.

Underground Fuels

The shaft fuel delivery pipe column to 750 m Level will be an "Energy Dissipation" pipe to prevent high flow velocities and pressure build-up. From 750 m to 850 m and 950 m Levels, a normal 50 mm pipe will be used.

Surface storage tanks were designed with sufficient fuel storage capacity to operate the mine for approximately ten production days. There are four tanks with a capacity of 83 m³ each, giving a total storage of 332 m³.

Underground storage tanks were designed with sufficient fuel storage capacity to operate the mine at the specific underground levels for the legal maximum of three days operating capacity. The envelope available for slinging limits the physical size of these tanks. The resulting volume of each of the tanks is 3.2 m³.

Diesel consumption was calculated using the maximum fleet operating on a specific level. Machine fuel consumption figures were obtained from the original equipment manufacturers (OEMs) based on a consumption per hour figure.

Lubrication Distribution: Surface and Underground

Lubrication oil, according to requirements, will be stored in bulk tanks on surface and dispensed to the surface and underground workshops.

Bulk lubes will be transferred from surface to the underground workshop storage tanks via dedicated pipelines in Shaft 1.

Surface storage tanks were designed with sufficient lube storage capacity to operate the mine for approximately 30 days (per grade of oil).

Fuel Management System

A fuel management system is part of the dispensing control system, designed to monitor and control diesel and lubrication use. Each nozzle has a tag detector to log vehicle identity, fuel and lube amount, time, location, and operator details. The system stores records for a set period and generates reports in various formats. It is fully compatible with the control system, offering live monitoring both in the control room and remotely.

16.7.7.2 Underground Workshops

Mobile equipment will stay underground for its entire life cycle, receiving maintenance in underground workshops. Machines only exit the mine for full refurbishment or disposal.

The mine will have three types of workshops:

- Main surface workshop:
  - Service and repairs for surface vehicles.
  - Rebuild of surface and underground vehicles (this may be performed off-site at the OEM).
- Main underground workshops on 750 m, 850 m and 950 m Levels:
  - Servicing of primary equipment, secondary equipment and LDVs.
  - Planned and unplanned maintenance.
  - Major components replacement (rock drill, cylinders, motor, brakes, transmission, compressor, water pump, diesel engine, buckets).
  - Tyre replacement.

- Satellite workshops near the working areas:
  - Slow-moving equipment only (drill rig, prod rig, shotcrete, charge-up unit).
  - Planned and unplanned maintenance.
  - Hourly services up to 500 hours.

The number of bays needed for underground workshops depends on fleet size and service requirements.

Production fleet vehicles primarily operating at the production face, such as drill rigs, bolters, and LHDs, will undergo servicing and minor maintenance at satellite workshops where feasible. These workshops will be constructed as close to the working areas as possible. For major services and repairs, all vehicles will be directed to the main workshop. Similarly, all other vehicles will report to the designated main workshop for both minor and major services and repairs.

The workshop layouts are designed to facilitate a one-directional flow within the main shaft infrastructure and in proximity to the return air vent raise. This strategic placement positions the workshop centrally near the main shaft to optimise logistics.

In the event of a fire within a workshop, fire doors will be activated to isolate the area. The proto team will enter from the air intake side and combat the fire towards the ventilation return side.

Tyres will be stored on the surface, with minimal quantities kept underground in a designated bay. Tyre repairs and fittings will take place on the surface and be transported to and from the workshops daily as needed. The multi-purpose utility vehicle (MPV) is equipped with a tyre handler to assist with wheel changes at the site or in case of a breakdown. The MPV will retrieve tyres from the tyre bay and return used or damaged tyres to the bay.

As the mine develops further from the main infrastructure, it is necessary to establish satellite workshops for minor breakdowns and applicable servicing up to 500 hours. Satellite workshops will comprise of two bays with an oil separator. One bay will be equipped with a lifting arrangement and the other with a wash bay.

16.7.7.3 Concrete and Shotcrete Facility and Distribution

A batch plant located on surface is required to supply shotcrete and concrete to the underground workings for support and construction. Stopes will be filled with waste rock or paste fill. For this purpose, bulkheads will be constructed at the base of the stopes using fibre shotcrete. Underground civil construction material will also be supplied from this plant. This plant is designed to adjust the mix according to the requirements, with flushing facilities to clean out the complete system prior to any changeover.

The batch plant is equipped with a compulsory planetary mixer L1000, capable of mixing a variety of concrete requirements, including steel and synthetic fibres, although if fibres and/or steel reinforcement is required for underground applications, it will be added underground. The mixer can produce high-quality shotcrete, ready-mix, mortar, and earth fry concrete. Production capacity varies from 10 m³/h up to 40 m³/h depending on the mixing period, materials used and requirements. The plant includes three aggregate silos filled by a wheel loader. Cement is added by a screw system from the cement silo.

There is an optional additive system for two liquid additives, including tanks, pumps and scale. This plant can be equipped with a dosing system for steel or synthetic fibres. The system will weigh and dose fibres automatically according to the requirements. Fibres are tumbled in the mixer to ensure homogeneous concrete and the best spraying result.

The optional micro silica dosing system enables for specific concrete options. Micro silica is dosed from bulk bags to scale.

Concrete is mixed into a wet sludge and discharged into one of two boreholes feeding the underground receiving areas, located in dedicated excavations on 750 m, 850 m and 950 m Levels.

16.7.7.4 Refuge Stations

Refuge chambers will be required <750 m from any workplace, in accordance with South African regulations. 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.

16.7.7.5 Sewage Handling

Underground sewerage will comprise two systems: fixed flushing toilets at the main workshops and mobile flushing (non-chemical) toilets for the remainder of underground workings.

The mobile toilets are designed as utility vehicle attachments and are easily manoeuvrable. Each unit is fitted with a 900 L sump and pumped empty into a sewage tank mounted on a cassette carrier. This tank is transported to surface and emptied into a sewerage disposal system on surface.

16.7.8 Power and Communication Systems

16.7.8.1 Power Requirements

A bottom-up estimating methodology was used to determine the predicted electrical power required and notified maximum demand for the various surface and underground installations. The estimated running load (kW) is derived from total connected load, which is reduced by excluding standby circuits and further reduced to absorbed power running loads as stipulated by the process and mechanical data.

16.7.8.2 Electrical Substations and Power Distribution

Power is distributed at 33 kV from the 33 kV Consumer Substation to the 33 kV Mining substation via three feeders consisting of cables on cable racking. Each feeder consists of two 3c x 185 mm² cross-linked polyethylene (XLPE) cable and rated for a load of 30 MVA. The distribution is designed to include a redundant supply.

16.7.8.3 33 kV Mining Substation

The 33 kV mining substation consists of 33 kV ABB GIS and supplies the following loads:

- Three 33/11 kV 30 MVA Transformers for the 11 kV mining substation.
- Feeders to ventilation fan substation VR6.
- Feeder to backfill plant substation.

16.7.8.4 11 kV Mining Substation

Power to all the surface and underground loads for Area 2000 is supplied from the 11 kV Mining substation. The substation is supplied via three 30 MVA transformers and is designed to have one transformer for redundancy. Incomers circuit breakers for the emergency generator plant and solar plant are allowed for. The following loads are supplied from the 11 kV mining substation:

- Feeders to underground substations.
- Ventilation substation VR4.
- Winders Shaft 1.
- Personnel and material Winder Shaft 2.
- Winders Shaft 3.
- Six feeders for power factor correction (PFC).
- Mini substations for surface 525 V loads.
- Mini substations for surface 400 V loads.

The 11 kV mining substation is equipped with two 33/11 kV 30 MVA transformers. The load on this substation will exceed 30 MVA in 2029, when the third 30 MVA transformer should be installed to maintain redundancy. The cost for feeder cables and transformer is allowed for in the Phase 2 estimate.

16.7.8.5 11 kV Underground Distribution

An 11 kV feed is installed from the 11 kV mining MV substation to an 11 kV substation placed at each main level underground. Five 3 x 240 mm² XLPE cables are already installed in Shaft 1, but the cables are not commissioned. Two additional 3c x 240 mm² XLPE cables will be installed in Shaft 3 for redundancy. The cables will be terminated and commissioned once the underground substations are installed. The design is to have one feeder cable per incomer circuit breaker for redundancy and flexibility.

Mining equipment is supplied from a 630 kVA 11 kV/1,000 V mini substation, gulley rig panels and local starter panels.

16.7.8.6 Mini Substations

525 V Mini Substations

All 525 V mini substations will consist of an 11 kV RMU, suitably rated 11 kV/550 V transformer and a 525 V compartment. The RMU shall be of the vacuum/sulfur hexafluoride (SF6) interrupted type. The neutral of 11 kV/550 V transformers will be connected to earth via a neutral earthing resistor (NER) to limit the earth fault. ADIT relays will be installed to trip the 11 kV transformer feeder in the event of an earth fault not being cleared by the 525 V system.

1,000 V Mini Substations

All 1,000 V mini substations will consist of an 11 kV RMU, suitably rated 11 kV/1,000 V transformer and a 1,000 V compartment. The RMU shall be of the vacuum/SF6 interrupted type. The neutral of 11 kV/1,000 V transformers will be connected to earth via an NER to limit the earth fault. ADIT relays will be installed to trip the 11 kV transformer feeder in the event of an earth fault not being cleared by the 1,000 V system.

400 V Mini Substations

All 400 V mini substations will consist of an 11 kV RMU, suitably rated 11 kV/400 V air natural air natural (ANAN) transformer and a 400 V compartment. The RMU shall be of the vacuum/SF6 interrupted type. The neutral of 11 kV/400 V transformers will be connected to earth.

16.7.8.7 Emergency Generator Plant

The existing emergency generator plant consists of the following generators, and the control and switching of the generator system is done as part of Phase 1. No additional generators are required as part of the Phase 2 study.

- 3 x 1 MVA generators.
- 2 x 1.25 MVA generators.

The existing generator plant will be connected to the 11 kV mining substation and emergency power to underground pumping will be via the distribution system. The distribution system is designed to have a fully redundant system, and no additional cables are required under emergency conditions.

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.

- Evaluating batch design to optimise cement and admixture usage minimising fill curing rates.
- Reducing the angle of the footwall and hanging wall in production stopes as a benefit of the definition drilling programme.

16.8.1 Recommendations

Mining recommendations:

- Optimise the definition drilling programme required for the initial mining areas.
- Continue mine planning work to reevaluate development and stopes to optimise the mine.
- Monitor fragmentation during the development stage to reduce the need for secondary breaking on development rock.
- Monitor and optimise the first development through the Tshukudu Fault.
- Develop an operating procedure to allow waste rock to go into secondary longhole stopes.
- Set up a programme for ore pass monitoring and rehabilitation 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

Following on from the previous 2017 DFS and 2022 PDP FS, further progression of the Platreef mining and processing strategy for the current 2025 IDP has resulted in a production plan that incorporates a phased ramp up in four phases to ultimately target a combined processing capacity of 10.67 Mtpa.

The Phase 1 Project is currently being executed and includes the construction of Shaft 1 and a 0.77 Mtpa concentrator which comprises a conventional three-stage crushing and ball milling circuit followed by flotation (MF1). The design is as per the 2022 PDP FS but with the inclusion of Jameson Cell dilution cleaning for the final flotation circuit cleaning duties. Phase 1 will be followed by the addition of 3.3 Mtpa of processing capacity in Phase 2 to target a combined concentrator production rate of 4.07 Mtpa.

This section details the process and engineering design basis for the Phase 2 Project, at FS level, which includes the construction of Shaft 3 and Shaft 2 (personnel and material only) in combination with the concentrator capacity expansion. The Phase 2 concentrator design includes all processing requirements from receipt of the ROM material from stockpiles and or ROM silo through to final concentrate load out and production of filtered tailings. The handling of filtered tailings and DTSF operation is not within the concentrator scope. Phase 2 will include the addition of:

- A new 4.07 Mtpa crushing circuit incorporating secondary cone crushing and a tertiary HPGR. The new Phase 2 crushing circuit will feed both the Phase 1 and Phase 2 milling circuit modules, and the Phase 1 crushing circuit will become redundant.
- A new 3.3 Mtpa milling and flotation module comprised of two stages of ball milling in series and flotation (MMF).
- A new 4.07 Mtpa tailings dewatering and pumping circuit in combination with a tailing's filtration plant located at the Rietfontein Dry Tailings Storage Facility (DTSF). The new Phase 2 tailings handling circuit will process the combined tailings and the Phase 1 tailings handling circuit will become redundant.

The process design has been developed based on the testwork findings and assessments, various desktop level trade-off studies and design information from the historical study phases in combination with relevant DRA design information. The flow sheet as summarised in Figure 17.1 is well known in industry and has historically been proven as a suitable processing route for various platinum ores.

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

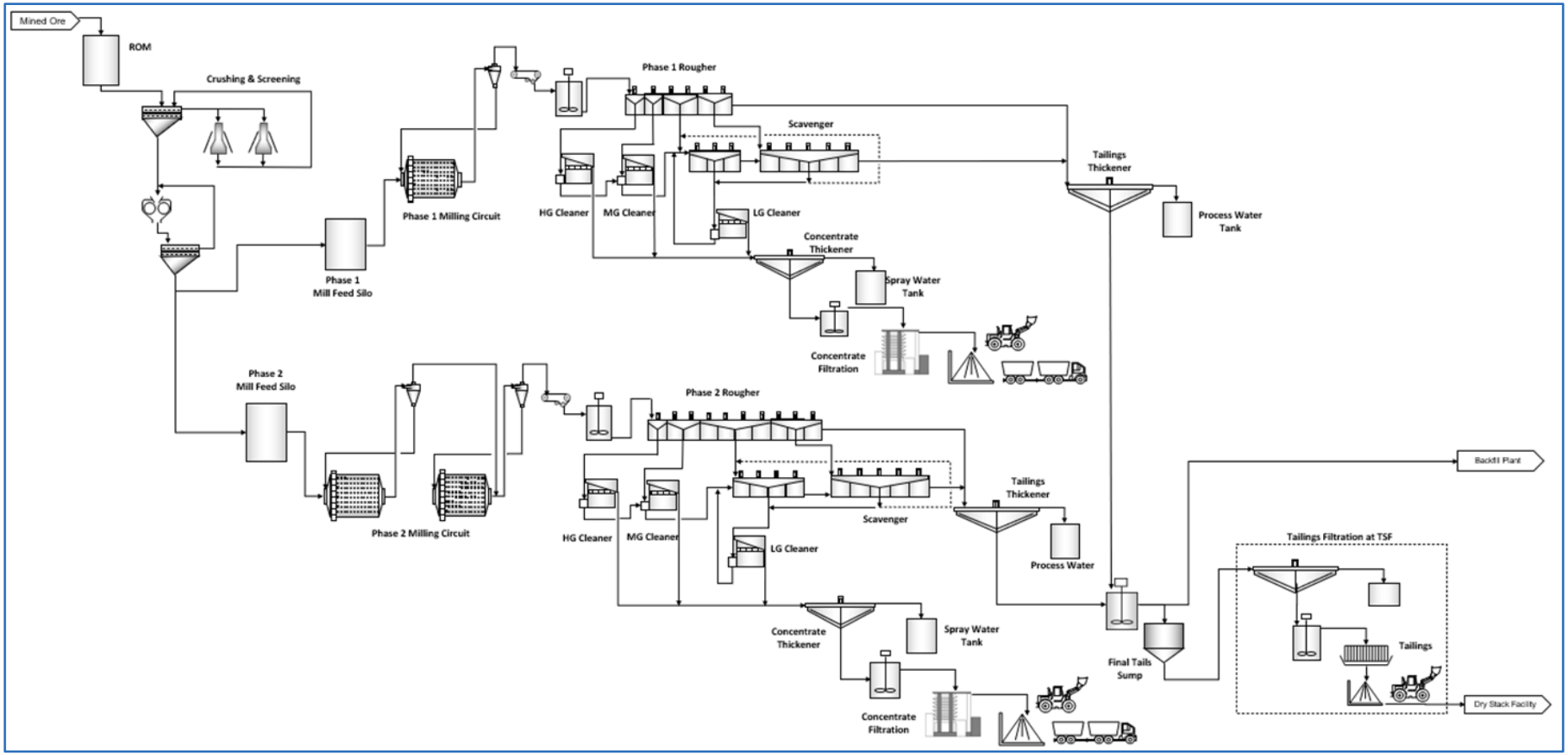
The engineering design has taken cognisance of the environmental and social impacts regarding noise, dust, light, vibration and visual pollution. Dust suppression is present on all ROM transfer points. Silos are specified with concrete roofs and the Phase 2 conveyors are sheeted to minimise noise, dust, and light pollution.

The final concentrator layouts were selected, 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 Phase 2 concentrator plant has been designed in accordance with the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities.

The future Project Phases 3 and 4, currently at PEA level, will aim to further expand production to 10.67Mtpa. This will include bringing Shaft 2 (Rock hoisting) online in combination with expansion of the crushing circuit and the addition of two 3.3 Mtpa concentrator milling and flotation modules which will be identical to the Phase 2 module. The concentrator tailings dewatering and pumping circuit capacity will also be expanded and two additional tailings filtration modules (located at the DTSF) will be installed, one each in Phases 3 and 4 respectively.

Figure 17.1 4.07 Mtpa Phase 2 Simplified Concentrator Flow Sheet



DRA, 2024

17.2 Introduction

Several studies have previously been undertaken for the Platreef Project, which can be summarised as follows:

- 2017 Platreef Definitive Feasibility Study (DFS): The 2017 DFS was based on the development of a large-scale, mechanised, 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). 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 the construction of Shaft 2, which was to be the mine's main production shaft, had also commenced.
- 2022 Platreef Phased Development Plan (PDP) FS: The 2022 PDP FS considered a higher throughput of 5.2 Mtpa. The design was based on a phased production ramp up which included the construction of an initial 0.77 Mtpa concentrator plant to treat uncrushed ROM material hoisted through Shaft 1. This would be followed by the addition of the 2 off 2.2 Mtpa concentrator modules as per the original 2017 DFS, to process primary crushed ROM material hoisted through Shaft 2. The 2017 FS process design remained unchanged except for the required changes to cater for a Dry Tailings Storage Facility (DTSF).

The current 2025 IDP Platreef mining and processing strategy has built on from these previous studies, resulting in a production plan that incorporates a phased ramp up in four phases to ultimately target a combined processing capacity of 10.67 Mtpa. The production ramp up, will include the phased introduction of additional processing capacity to accommodate the increased mining production, and additional concentrator modules will be commissioned and brought online as sustainable concentrator feed material becomes available as follows:

- The Phase 1 Project is currently in implementation and includes the construction of Shaft 1 and a 0.77 Mtpa concentrator.
- The Phase 2 Project will include the construction of Shaft 3 and Shaft 2 (personnel and material only) and the addition of 3.3 Mtpa of processing capacity to target a combined concentrator production rate of 4.07 Mtpa. The Phase 2 Project is currently presented at FS level.
- The future Project Phases 3 and 4 will aim to further expand production to 10.67. This will include bringing Shaft 2 fully online (rock hoisting) in combination with further expansion of the crushing circuit and the addition of two 3.3 Mtpa concentrator milling and flotation modules which will be identical to the Phase 2 module. The concentrator tailings dewatering and pumping circuit capacity will also be expanded and two additional tailings filtration modules (located at the DTSF) will be installed, one each in Phases 3 and 4 respectively. Phases 3 and 4 are currently at PEA level.

This Section outlines the Phase 2 concentrator process design basis as well as the engineering design. The process design caters for the following key aspects:

- ROM handling and storage
- Secondary and tertiary crushing and screening
- Milling

- Flotation
- Concentrate thickening, filtration and storage
- Tailings thickening and slurry transfer
- Tailings thickening and filtration (located at the DTSF)
- Air, water and reagent services

17.3 Concentrator Process Design

The 0.77 Mtpa Phase 1 concentrator plant (currently in implementation phase) includes a conventional three-stage crushing and ball milling circuit followed by flotation (MF1). The design is aligned to the 2022 PDP FS but with the inclusion of Jameson Cell dilution cleaning for the final flotation circuit cleaning duties aligned to the testwork findings.

Phase 2 will include the addition of a new 4.07 Mtpa crushing circuit incorporating secondary cone crushing (primary crushing in mining scope) and a tertiary High-Pressure Grinding Roll (HPGR) followed by two stages of ball milling in series and flotation (MMF). The Phase 2 concentrator plant scope includes the installation of a 4.07 Mtpa tailings dewatering and pumping circuit in combination with a tailing's filtration plant located at the Rietfontein DTSF.

The Phase 2 processing philosophy is based on a 4.07 Mtpa throughput while making provision for a phased expansion to 10.67 Mtpa using a modular approach as follows:

- The crushing circuit for the Phase 2 FS will have 4.07 Mtpa installed capacity, with space to upgrade to 10.67 Mtpa.
- The new 3.3 Mtpa Phase 2 concentrator module will have a dedicated two-stage milling and flotation circuit. Two additional 3.3 Mtpa milling and flotation modules will be added in the future to increase throughput to 10.67 Mtpa.
- One thickener sized for 3.3 Mtpa will be included in Phase 2. Additional concentrate thickening capacity will be added when throughput increases to 10.67 Mtpa.
- One tailings thickener and one clarifier will be included in Phase 2. Additional tailings thickening capacity will be added as total throughput increases to 10.67 Mtpa.
- The tailings disposal circuit (tank and pipelines) will be able to pump the Phase 1 (0.77 Mtpa) and Phase 2 (3.3 Mtpa) tailings to the DTSF and feed the backfill plant, space provision will be made for additional pump trains and pipelines to cater for future phases.
- The Phase 2 tailings filtration circuit located at the DTSF (Rietfontein) will be sized for 4.07 Mtpa and space provision will be allowed for the future 10.67 Mtpa capacity expansion.
- Reagent mixing tanks will be sized to include future expansions, and dosing/day tanks will be sized and selected for one 3.3 Mtpa module. Additional dosing/day tanks will be added with each additional module.
- Air compressors, blowers, and dryers, as well as compressors for the concentrate filter, will only be included for the 3.3 Mtpa circuit, with space allowance for the future Phases 3 and 4 upgrades.

- Process and clean water circuits will only allow for the Phase 2 requirements, with space allowance for the future Phases 3 and 4 upgrades.
- Plant infrastructure includes capacity in the central control room, various substations and transformer bays, plant offices, workshops extension, stores extension, change house, canteen and access control.

The concentrator throughput for Phases 1, 2, 3 and 4 are summarised in Table 17.1

Table 17.1 Summary of the Concentrator Throughput Requirements

Criteria	Units	Value
Plant Throughput		
Design Throughput Phase 1 (Implementation)	Mtpa (dry)	0.77
Design Throughput Phase 2	Mtpa (dry)	0.77 + 3.3 = 4.07
Design Throughput Phase 3 (Future)	Mtpa (dry)	0.77 + 3.3 + 3.3 = 7.37
Design Throughput Phase 4 (Future)	Mtpa (dry)	0.77 + 3.3 + 3.3 + 3.3 = 10.67

The Phase 2 concentrator design criteria for the 4.07 Mtpa throughput is summarised in Table 17.2. The key aspects of concentrator plant design are discussed below.

Table 17.2 Summary of the Phase 2 Concentrator Design Criteria

Criteria	Units	Value
Plant Throughput		
Design Throughput Phase 2	Mtpa (dry)	0.77 + 3.3 = 4.07
Head Grades (Min – Max/Average)		
Platinum	g/t	0.95 – 2.35/1.88
Palladium	g/t	0.97 – 2.44/1.93
Rhodium	g/t	0.07 – 0.18/0.13
Gold	g/t	0.16 – 0.38/0.29
3PE+Au	g/t	2.14 – 5.27/4.22
Copper	%	0.11 – 0.19/0.16
Nickel	%	0.23 – 0.38/0.33
Sulfur	%	0.56 – 0.97/0.80
Overall Recovery (Min – Max/Average)		
3PE+Au Recovery	%	79.4 – 89.6/85.8
Nickel Recovery	%	58.7 – 75.7/70.9
Copper Recovery	%	80.4 – 90.3/87.3

Criteria	Units	Value
Concentrate Production		
3PE+Au Grade Target	g/t	85
Nickel Grade Expected	%	2.5 – 4.2
Copper Grade Expected	%	4.0 – 6.9
Mass Pull Expected	% Feed	2.8 – 5.5
Filtered Concentrate Moisture Target	%w/w	12 - 14
Crushing Circuit Operating Schedule		
Phase 2 Capacity	Mtpa	4.07
Operating Days per Annum	days	365
Crushing Circuit Running Time	%	68.5
Overall Running Time	h/annum	6 000
Design Circuit Feed Rate	t/h (dry)	678
Milling and Flotation Circuit Operating Schedule		
Phase 2 Capacity	Mtpa	0.77 + 3.3 = 4.07
Operating Days per Annum	days	365
Milling Circuit Running Time	%	91.3
Overall Running Time	h/annum	8 000
Circuit Feed Rate 0.77 Mtpa Milling & Flotation Module	t/h (dry)	96.3
Circuit Feed Rate 3.3 Mtpa Milling & Flotation Module	t/h (dry)	413
Tailings Filter Plant (Located at Rietfontein DTSF)		
Design Average Mass Pull to Tailings	% Feed	95.7
Filter Plant Design Availability	%	80
Filtered Tailings Moisture Target	%w/w	12 – 14

17.3.1 ROM Handling and Crushing

In Phase 2, the concentrator will receive primary crushed ore (250 mm top size) from the ROM ore silo which will be supplied from the mining operations through Shaft 3 hoisting and conveyed to the ROM silo. Primary crushing is underground. Shaft 1 will hoist waste and low grade ore and Shaft 2 will hoist personnel and material only.

The Phase 2 crushing circuit has a design capacity of 4.07 Mtpa and will include a secondary cone crushing circuit followed by an HPGR in a tertiary crushing duty. The HPGR will be operated in closed circuit to produce a mill feed size of 100% passing 13 mm. The new Phase 2 crushing circuit will feed both the Phase 1 and Phase 2 milling circuit modules. The 0.77 Mtpa Phase 1 crushing circuit will become redundant.

The 2017 FS and Phase 1, 0.77 Mtpa concentrator design was based on a conventional three-stage crushing circuit utilising cone crushers for the secondary and tertiary crushing duty. HPGR testwork and trade-off assessments conducted during the previous 2022 PDP FS, showed that an HPGR in a tertiary crushing application, offered the potential for an approximate 7% comminution circuit operating cost saving. Based on this, a decision was made to include an HPGR for the tertiary crushing duty in the Phase 2.

The secondary cone crushing circuit sizing has been based on a crusher work index (CWi) of 22 kWh/t, which represents the 85th percentile CWi value from the 2017 FS comminution variability testwork. The HPGR circuit simulations and sizing calculations were conducted by the DRA in-house comminution consultant based on the semi-pilot scale HPGR (SMALLWAL) testwork conducted by Thyssenkrupp for the 2022 PDP FS. A single Ø2.0 m x 1.0 m, 2 x 1250 kW HPGR, operating at 3.3–3.6 N/mm^2 and a mass flow rate of 322 t.s/h.m^3 has been selected for the 4.07 Mtpa Phase 2 tertiary crushing duty.

The crushing circuit will be expanded by installing additional crushing, screening and HPGR equipment to further increase production to 10.67 Mtpa in the future for the Phases 3 and 4 capacity expansion.

17.3.2 Milling

The new 3.3 Mtpa Phase 2 milling circuit will be comprised of a two stages of ball milling in series and a cyclone cluster operated in closed circuit. The circuit will be fed with 100% passing 13 mm tertiary HPGR product material and will produce a classification cyclone overflow product size of 80%, passing 75 µm.

The mills have been sized to cater for the 85th percentile ore hardness as determined from the comminution variability test data. The recommended mill size for each mill (primary and secondary operated in series) is Ø6.25 m x 9.37 effective grinding length (EGL) with an overflow discharge liner arrangement and 7 MW variable speed geared pinion drives. The ball charge, together with the variable speed motor capability, allows for variation in the feed material, required grind size and mill throughput.

A summary of the expected primary and secondary ball mill performance when treating ore of the 85th percentile hardness, is presented in Table 17.3.

Table 17.3 Summary of Ball Mill Performance Criteria

Duty	Specification	Ball Charge % by volume	Expected Gross Mill Power Draw (kW)
Primary Mill	Ø6.25 m x 9.37 m EGL, Overflow Ball Mill, 7MW (VSD)	30	6,033
Secondary Mill	Ø6.25 m x 9.37 m, Overflow Ball Mill, 7MW (VSD)	30	5,931

A single 3.3 Mtpa ball milling circuit has been selected for Phase 2 with provision to increase production to 10.67 Mtpa through the addition of a new 3.3 Mtpa concentrator milling module in Phases 3 and 4 respectively. These modules will be identical to the Phase 2 module.

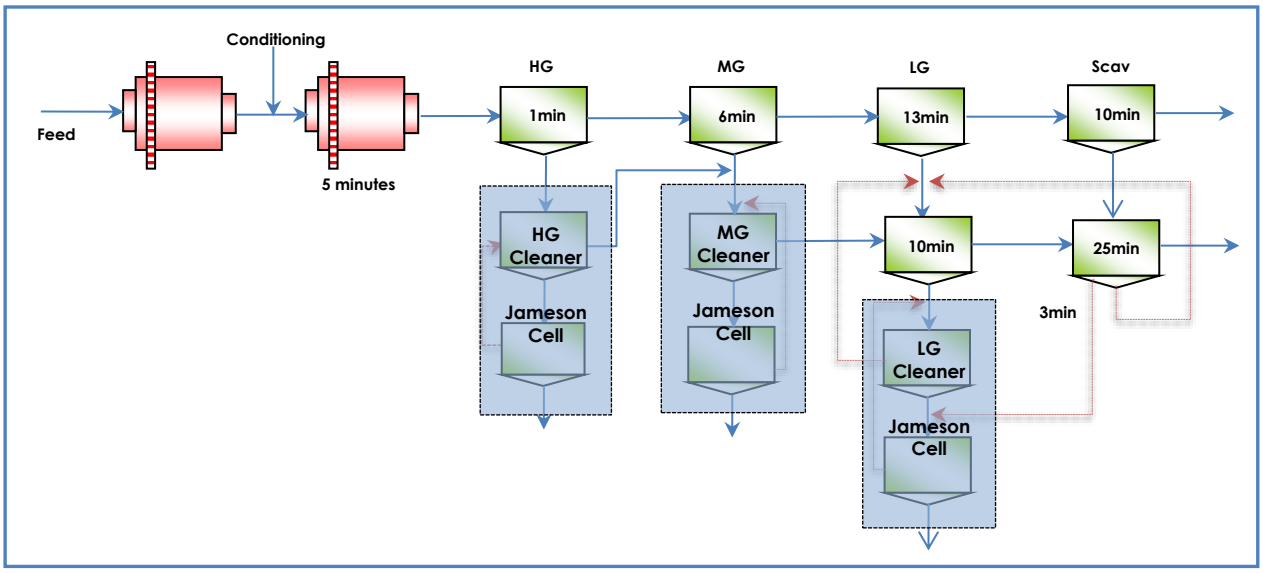
Testwork has indicated that the flotation circuit response is sensitive to the grinding media type used. It was noted that the flotation performance improved when using high chrome media (Cr>16%) as compared to carbon steel media. Based on the testwork findings, the design and operating cost estimates 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.

17.3.3 Flotation Circuit

The optimised flotation circuit flow sheet as developed during the metallurgical testwork 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 17.2. 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 17.2 Optimised Flotation Flow Sheet as Derived from Testwork



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A single 3.3 Mtpa flotation circuit has been selected for the Phase 2 expansion with provision to increase production to 10.67 Mtpa through the addition of a new 3.3 Mtpa flotation module in Phases 3 and 4 respectively. These modules will be identical to the Phase 2 module.

The historical 2017 DFS and 2022 PDP FS flotation circuit flow sheet incorporated conventional tank flotation cells in combination for with box flotation cells for the final re-cleaner and re-re-cleaner flotation stages. Subsequent to this, additional testwork was conducted by XPS in 2022. This testwork showed that a single Jameson cell was able to replicate or better the performance of two-stages of conventional cleaning. Jameson Cells have thus been installed for the HG cleaner, MG cleaner and LG re-cleaner flotation duty for the Phase 1 Project which is currently in implementation. This same approach has also been adopted for the Phase 2 flotation circuit design.

The sizing of the mechanically agitated flotation cells is based on the residence time requirements while also ensuring that the froth carry rate (FCR) and lip loading (LL) fall within typical design standards. The Jameson cells were sized by the vendor based on bench-scale open circuit testwork results and throughput requirements.

The flotation circuit residence times for the mechanically agitated cells have been based on the bench-scale flotation circuit residence times. A residence time scale-up factor of 2.5 was applied to determine the design residence time requirements for scale operations. The flotation pumping systems and flotation circuits were designed based on the mass pull ranges achieved during bench-scale flotation testwork.

The flotation circuit flow sheet is based on the copper collector reagent regime as derived from testwork and comprises:

- Copper collector
- Sodium Isopropyl Xanthate (SIPX) collector

- Aero promoter
- Frother
- Depressant

Preliminary mini pilot plant tests during the 2022 PDP FS indicated that sodium isobutyl xanthate (SIBX) reagent is a potential alternative to replace the copper collector, SIPX collector and Aero promoter. This was also noted during the original flow sheet development phase but was not adopted due to froth stability concerns. The mini pilot plant commissioning however, achieved adequate froth stability, but performance was not optimised and did not fully replicate the copper collector reagent suite performance. The alternative SIBX reagent suite may be considered for further trials during full scale operations.

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 slurry piping within the plant is specified to be lined to minimise any slurry contact.

17.3.4 Concentrate Thickening and Filtration

The concentrate thickening and filtration circuit is based on the installation of a new concentrate thickener and pressure filter to treat the combined concentrate from the new 3.3 Mtpa Phase 2 flotation circuit. The Phase 1 concentrate thickening and filtration circuit will continue treating the Phase 1 flotation circuit concentrates.

Based on the bench-scale MO testwork (2021) in combination with benchmarked data, a Ø33 m concentrate thickener has been selected catering for a unit area thickening rate of 0.04 t/h/m². A 120 m² Larox pressure filter has been selected based on a recommended design filtration rate of 360 kg/h/m². The sizing has conservatively been based on mass pull of 6.1% of mill feed. The nominal concentrate mass pull for Phase 2 is expected to range from 2.5–5.5% of mill feed and average approximately 4.1% of mill feed.

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.

17.3.5 Tailings Dewatering and Transfer

The initial Phase 1, 0.77 Mtpa process plant design includes a tailings thickener and vacuum filter located at the concentrator. During Phase 1 (0.77 Mtpa), the tailings filter cake will be trucked to the DTSF, located at the mine site. No slurry tailings from the concentrator plant will be required for mine backfill during this phase.

Once the production rate is increased to 4.07 Mtpa in Phase 2 backfill will transition to utilising slurry tailings from the concentrator plants to make-up the CPF mix. The backfill plant will operate intermittently depending on mining backfill requirements and is expected to utilise approximately 32% of the concentrator tailings over LOM.

The Phase 2 concentrator flow sheet includes a new tailings thickener and overflow clarifier which will operate in combination with the Phase 1, 0.77 Mtpa thickening circuit to achieve a total mill throughput of 4.07 Mtpa.

A Ø36 m tailings thickener has been selected based on a unit area thickening rate of 0.42 t/m²/h and a target underflow solids concentration of 55 to 65% (w/w). The design also includes a Ø12 m clarifier to remove solids from the thickener overflow.

In Phase 2, the 0.77 Mtpa Phase 1 vacuum filtration circuit will not be operational. The thickened slurry streams from each flotation circuit will be combined in a new slurry transfer circuit, sized for the total 4.07 Mtpa mill throughput. The combined thickened tailings will be pumped to the backfill plant or alternately to a new filter plant located at the Rietfontein DTSF.

To accommodate the requirement to pump the combined slurry tailings to either backfill and or the Rietfontein DSTF, the design incorporates an agitated backfill feed tank and slurry transfer pumps which will pump the combined tailings to the backfill plant at a solid's concentration of 60 to 65% (w/w). When full, the backfill tank will overflow directly into an agitated tailings transfer tank which will pump the tailings to the Rietfontein DTSF filter plant at a solid's concentration of 50 to 55% (w/w) via a booster station. This allows for pumping the combined tailings to the DTSF when the concentrator is producing more tailings than required in the paste mix, or when the backfill plant is down for maintenance. The tailings pumping systems include provision for a flushing system to minimise settling of solids in pipelines. 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.

Two tailings pump trains (running/standby) and a single pipeline will be installed for pumping the tailings slurry to the backfill plant. Two tailings' pipelines and pump trains (running/standby) will be installed at both the concentrator and booster station for pumping the tailings slurry to the DTSF. The design includes space provision for additional backfill feed and tailings transfer pump trains and pipelines to cater for the future Phases 3 and 4 capacity expansion.

17.3.6 Tailings Filtration

The Phase 2 flow sheet includes a new tailings receiving, thickening and pressure filtration circuit located at the Rietfontein DTSF, which is sized for a 4.07 Mtpa mill throughput rate. The circuit will receive the combined pumped concentrator flotation tailings slurry at a solid's concentration of 55 to 60% (w/w) and produce a filtered tailings product with a moisture content of 12 to 14% (w/w).

A Ø38 m thickener has been selected based on a unit area thickening rate of 0.42 t/m²/h and a target underflow solids concentration of 60 to 65% (w/w). Four off 816 m² plate and frame pressure filters have been selected based on the optimal testwork filtration rate of 246 to 303 kg/h/m² in combination with appropriate run time and design factors which make provision for operational and maintenance requirements. The design incorporates two agitated filter feed surge tanks catering for 1.4 hours residence time (based on thickener underflow).

The design also includes space provision for additional tailings slurry receiving, thickening and filtration capacity for the future Phases 3 and 4 capacity expansion.

The handling and disposal of the filtered tailings falls within the DTSF consultant scope.

17.4 Concentrator Plant Production Schedule

The concentrator plant production schedules for the Platreef 2025 FS showing feed tonnes and grade, recoveries and metal production is presented in Table 17.4 and Table 17.5.

Table 17.4 Platreef 2025 FS Concentrator Feed

	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrator Feed	**kt**	**129,742**	**84**	**715**	**1,265**	**3,988**	**4,070**	**4,070**	**4,070**	**4,070**	**4,070**	**40,700**	**62,641**
Platinum	g/t	1.88	2.16	1.93	1.79	1.84	1.85	1.89	1.67	1.68	1.70	1.85	1.93
Palladium	g/t	1.93	2.11	1.93	1.82	1.89	1.96	1.98	1.85	1.85	1.84	1.97	1.92
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.26	0.27	0.27	0.29	0.26	0.28	0.30
Rhodium	g/t	0.13	0.14	0.12	0.11	0.12	0.13	0.13	0.12	0.11	0.12	0.13	0.13
Copper	% Cu	0.16	0.19	0.19	0.18	0.18	0.16	0.16	0.16	0.16	0.16	0.17	0.16
Nickel	% Ni	0.33	0.38	0.38	0.35	0.36	0.33	0.33	0.32	0.33	0.33	0.34	0.32
Sulfur	% S	0.80	0.97	1.00	0.93	0.92	0.81	0.79	0.83	0.86	0.83	0.83	0.77
3PE+Au	g/t	4.22	4.74	4.28	4.02	4.14	4.20	4.27	3.91	3.93	3.91	4.23	4.29

Table 17.5 Platreef 2025 FS Concentrator Recoveries

	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrator Recoveries													
Platinum	%	86.88	88.88	86.79	85.07	85.96	86.59	86.83	85.49	85.53	85.45	86.69	87.07
Palladium	%	86.29	88.65	86.24	84.30	85.34	86.02	86.31	84.59	84.67	84.57	86.12	86.45
Gold	%	78.33	78.80	78.16	76.90	77.65	78.20	78.35	77.44	77.48	77.43	78.25	78.47
Rhodium	%	79.78	82.88	79.76	77.56	78.66	79.39	79.73	77.78	77.85	77.73	79.52	79.94
Copper	%	87.33	88.48	90.20	88.23	88.85	87.51	87.52	87.30	87.45	87.45	87.95	86.58
Nickel	%	70.81	74.03	75.37	72.42	73.39	71.00	70.76	70.29	70.52	70.80	72.04	69.53
Sulfur	%	70.40	72.72	75.11	72.84	73.05	70.62	70.00	71.20	72.03	71.12	70.99	69.34
3PE+Au	%	85.81	87.89	85.73	83.90	84.88	85.58	85.83	84.28	84.31	84.27	85.64	85.97
Mass Pull	%	–	6.54	4.37	3.98	4.12	4.20	4.27	3.86	3.88	3.85	4.24	4.36
Concentrate Produced	kt	5,523	5	31	50	164	171	174	157	158	157	1,724	2,731

Table 17.6 Platreef 2025 FS Concentrator Flotation Concentrate Grade

	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrator Grade													
Platinum	g/t	38.28	29.39	38.25	38.11	38.28	38.17	38.34	37.05	37.07	37.60	37.95	38.69
Palladium	g/t	39.11	28.55	38.08	38.56	39.15	40.17	39.94	40.65	40.35	40.29	40.03	38.22
Gold	g/t	5.33	4.01	5.45	5.79	5.55	4.93	5.02	5.45	5.74	5.26	5.20	5.41
Rhodium	g/t	2.43	1.75	2.23	2.19	2.23	2.38	2.36	2.33	2.30	2.34	2.44	2.47
Copper	% Cu	3.31	2.58	3.95	3.92	3.85	3.40	3.34	3.65	3.67	3.69	3.47	3.09
Nickel	% Ni	5.47	4.29	6.47	6.39	6.35	5.60	5.45	5.92	5.95	6.05	5.80	5.08
Sulfur	% S	13.28	10.75	17.19	16.95	16.23	13.64	12.96	15.38	16.04	15.33	13.84	12.24
3PE+Au	g/t	85.15	63.70	84.01	84.65	85.21	85.65	85.66	85.48	85.46	85.50	85.62	84.79

Table 17.7 Platreef 2025 FS Contained Metal

	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Recovered Metal													
Platinum	koz	6,797	5	38	62	202	210	214	187	188	190	2,103	3,397
Palladium	koz	6,944	5	38	62	207	221	223	205	205	203	2,218	3,356
Gold	koz	947	1	5	9	29	27	28	27	29	27	288	475
Rhodium	koz	432	0	2	4	12	13	13	12	12	12	135	217
Copper	klb	402,911	310	2,719	4,354	13,934	12,795	12,820	12,637	12,763	12,761	131,804	186,015
Nickel	klb	665,535	517	4,462	7,097	23,022	21,090	20,892	20,485	20,694	20,931	220,306	306,041
Sulfur	klb	1,617,135	1,295	11,848	18,833	58,811	51,410	49,708	53,237	55,775	52,998	525,972	737,249
3PE+Au	koz	15,120	11	84	137	450	471	479	432	433	431	4,745	7,446

17.5 Process Description

The process descriptions for the Platreef 2025 FS are based on the detailed process flow diagrams (PFDs) that include mass balances. A summary block flow diagram for the Phase 2 4.07 Mtpa concentrator plant is presented in Figure 17.1.

The process description for the 0.77 Mtpa Phase 1 concentrator plant which is currently in the implementation phase, is aligned to the 2022 PDP FS but with the inclusion of Jameson Cell dilution cleaning for the final flotation circuit cleaning duties aligned to the testwork findings.

The process description as presented below focuses on the additional equipment which will be installed for the Phase 2 capacity expansion.

17.5.1 ROM Handling

The ROM and crushing circuit for the Phase 2 FS will have 4.07 Mtpa installed capacity and allows space provision for the planned future expansion to 10.67 Mtpa.

The circuit has the capability to convey ROM material from Shaft 3, Shaft 1 will hoist waste and low grade ore, whilst Shaft 2 will cater for personnel and material only, the winders and equipment required for hoisting rock from Shaft 2 is excluded from Phase 2 and will be included in Phase 3. The ROM feed conveyor is included in Phase 2 and the underside of the discharge chute of this conveyor is the battery limit between mining and the concentrator plant.

Primary crushed ROM material with a top size (F_{100}) of 250 mm from underground will be conveyed to a single 15,000 tonne ROM silo for storage ahead of the new Phase 2 concentrator screening and crushing circuit. The design also includes provision for a ROM stockpile.

17.5.2 Crushing and Screening

The new Phase 2 concentrator screening and crushing circuit consists of secondary cone and tertiary High Pressure Grinding Roll (HPGR) crushing stages each in closed circuit with secondary and tertiary screening stages respectively. This circuit will produce a mill feed size of 100% passing 13 mm for both the Phase 1 and Phase 2 milling circuit modules. The 0.77 Mtpa Phase 1 crushing circuit will become redundant in Phase 2.

Primary crushed ROM feed material will be extracted from the ROM silo at a controlled rate via two (running/standby) variable-speed apron feeders onto a conveyor which feeds the concentrator secondary screening circuit feed bin. The bin will be equipped with hydraulically driven spile bars. The design also includes provision to reclaim material from the ROM stockpile via a ROM reclaim bin and apron feeder that discharges material onto the secondary screening bin feed conveyor. This conveyor will be fitted with a weightometer, a cross-belt self-cleaning magnet, and a metal detector.

Material will be withdrawn from the secondary screening circuit feed bin via a variable-speed vibratory feeder which will discharge onto a double deck secondary screen. The +50 mm secondary screen oversize will be conveyed to the secondary cone crushing circuit comprised of two cone crushers operated in parallel. Each cone crusher will be fed via a feed bin fitted with a variable speed vibrating withdrawal feeder and the crusher product will discharge back onto the secondary screening bin feed conveyor.

The secondary crushing screen –50 mm undersize will be conveyed to a tertiary crushing circuit feed bin. This conveyor will be fitted with a cross-belt, self-cleaning magnet and metal detector for the removal of tramp metal prior to tertiary crushing. A weightometer will also be installed on this conveyor and the bin will be equipped with hydraulically driven spile bars.

The tertiary crushing circuit is comprised of a single HPGR for the Phase 2 throughput. The tertiary HPGR will be fed via a dedicated feed bin fitted with a variable speed belt feeder. The HPGR feed bin will also be equipped with hydraulically driven spile bars. The belt feeder will be equipped with a metal detector and flopper gate to remove any misplaced metal from the HPGR feed. The HPGR product will be conveyed to the tertiary screening circuit feed bin.

Material will be withdrawn from the tertiary screening circuit feed bin via a variable-speed vibratory feeder which will discharge onto a double deck tertiary screen. The +13 mm tertiary screen oversize will discharge directly onto the HPGR circuit feed conveyor which feeds the HPGR feed bin. The –13 mm tertiary crushing screen undersize will be conveyed to a transfer tower where this product can either be transferred to the Phase 1 mill feed tie-in conveyor or to the Phase 2 mill silo feed conveyor.

The secondary crushing area is equipped with an overhead crane and hoists for maintenance purposes. Hoists will also be installed in the tertiary crushing building for maintenance purposes. The secondary and tertiary crushing areas are both equipped with dust suppression units.

In addition to the Phase 2 equipment described above, the design includes space provision to add a secondary screen, secondary cone crusher, tertiary HPGR and tertiary screen for the future 10.67 Mtpa capacity expansion. In future, the Phase 2 mill silo feed conveyor will also be able to feed the Phase 3 and Phase 4 mill silo. Conveyors have been sized to cater for the 10.67 Mtpa throughput.

17.5.3 Milling

The crushing and screening circuit product (P_{100} of 13 mm) for the 0.77 Mtpa Phase 1 concentrator plant milling circuit will be transferred to the Phase 1 milling circuit which includes a 1,600 tonne mill feed silo and primary 5.48 mØ x 7.8 m EGL, 4.5 MW grate discharge ball mill which will be operated in closed circuit with a classification cyclone cluster. The 0.77 Mtpa milling circuit module has a design capacity of 96.3 dtph.

The crushing and screening circuit product (P_{100} of 13 mm) for the new 3.3 Mtpa concentrator plant milling circuit will be stored in an 8,000 tonne mill feed silo, drawn down by way of variable speed belt feeders which transfer material onto the primary mill feed conveyor.

The 3.3 Mtpa milling module for the Phase 2 expansion has a design throughput rate of 413 dtph and comprises an identical primary and secondary 6.25 mØ x 9.37 m EGL overflow ball mill. Each mill will be fitted with a 7 MW variable speed geared pinion drive. The mills will be operated in series and both the primary and secondary milling stage will be operated in closed circuit with a classification cyclone cluster.

The mill feed material (F_{100} of 13 mm) is fed to the primary mill feed hopper where process water is added for in-mill density control. The primary mill feed conveyor is equipped with a weightometer for mill feed rate control. Steel balls will be loaded onto the primary mill feed conveyor through the mill feed ball addition hopper and ball loading magnet. Process water will be added to the primary mill feed hopper for in-mill density control.

The primary mill product will discharge into a sump via a screen for scats removal before being pumped to a classification cyclone cluster. The primary classification cyclone cluster will produce an overflow product of P_{80} of approximately 150–180 µm as feed to the secondary milling circuit. The cyclone underflow will gravitate to the primary mill feed hopper for further regrinding. Copper collector reagent will be added into the primary mill feed hopper to assist in maximising copper recovery in the high grade flotation circuit.

The primary cyclone overflow will be combined with secondary mill product in the secondary ball mill discharge sump before being pumped to the secondary mill classification cyclone cluster. The secondary mill classification cyclone cluster will produce an overflow product of P_{80} of 75 µm which will gravitate to the rougher flotation feed tank via a two-stage sampling system. Secondary classification cyclone underflow will be fed to the secondary mill by gravity. The secondary mill product discharge will also be screened for scats removal prior to classification.

Mill scats from both mills will be deposited into a scats bunker and reclaimed via front-end loader. The scats will be taken to waste handling area, or alternatively, scats can be re-loaded onto the mill feed conveyor.

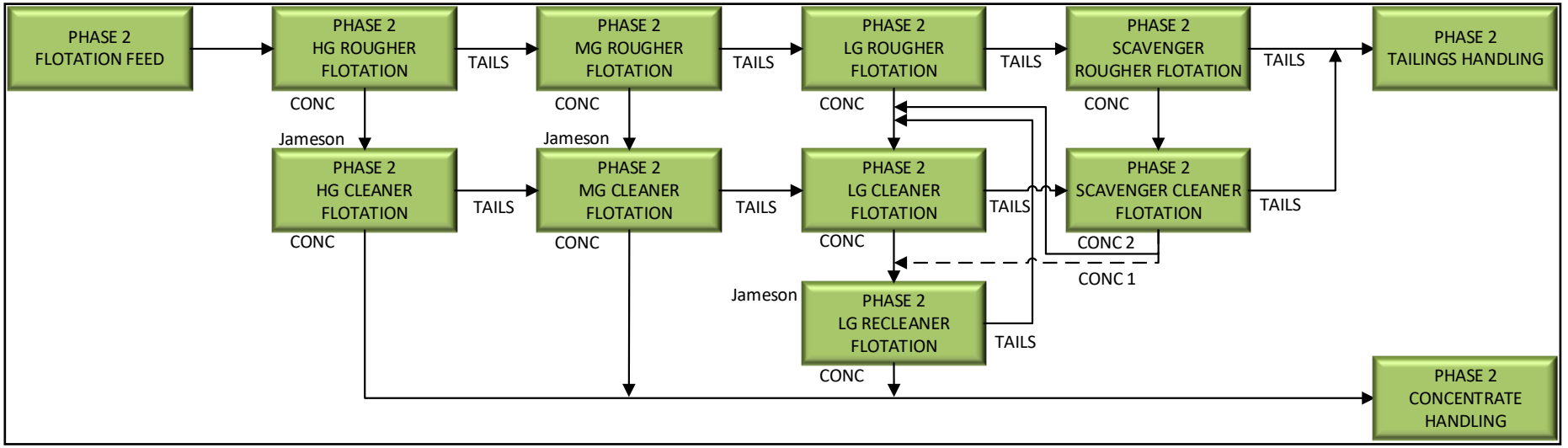
The design includes space provision to increase production to 10.67 Mtpa through the addition of an additional 3.3 Mtpa concentrator milling module in Phases 3 and 4 respectively. These modules will be identical to the Phase 2 module.

17.5.4 Flotation

The new 3.3 Mtpa flotation circuit for the Phase 2 capacity expansion will have the same overall flow sheet that has been adopted for the 0.77 Mtpa Phase 1 flotation circuit. There are variances in the cleaner circuit mass balance for the 0.77 Mtpa and 3.3 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 approximately 4.2 g/t once the 3.3 Mtpa concentrator comes online.

The new 3.3 Mtpa flotation circuit for the Phase 2 capacity expansion is described below and a high level summary flow sheet is presented in Figure 17.3.

Figure 17.3 Flotation Circuit High Level Flow Sheet



DRA, 2024

The rougher flotation feed, at approximately 30% solids (w/w), will be pumped from the rougher flotation feed tank to the rougher flotation circuit. The rougher flotation bank has a total residence time of 75 minutes and is comprised of 1 x 100 m³ and 9 x 160 m³, mechanically agitated forced air, flotation tank cells.

Four rougher concentrates will be produced namely a high grade (HG), medium grade (MG), low grade (LG) and scavenger concentrate that report to the HG, MG, LG and Scavenger cleaner circuits respectively.

The rougher flotation tailings will gravitate to a tailings sump via a two-stage sampling system. The rougher tailings, together with the scavenger cleaner tailings, will be pumped to the final tailings thickening and disposal circuit.

Collector, depressant, and promotor will be added into the MG and LG rougher feed boxes.

HG rougher concentrate will be pumped to the HG cleaner Jameson flotation cell. Depressant and frother are added to the HG cleaner feed box. The HG Jameson cell cleaner concentrate will be pumped to the concentrate thickener. The HG cleaner tailings will gravitate to a collection sump before being pumped to the MG cleaner feed box.

MG rougher concentrate and HG cleaner tailings will be combined with depressant, frother and collector in the MG cleaner feed box. A Jameson flotation cell has been selected for the MG cleaner duty. The MG Jameson cell cleaner concentrate will be pumped to the concentrate thickener. The MG cleaner tailings will gravitate to a collection sump before being pumped to the LG cleaner feed box.

LG rougher concentrate, MG cleaner tailings and LG re-cleaner tailings will report to the LG cleaner feed box, where depressant, collector and frother will be added. The LG cleaner circuit has a design residence time of 25 minutes and is comprised of 4 x 130 m³, mechanically agitated forced air, flotation tank cells.

The LG cleaner concentrate will be collected in a sump and pumped to the LG re-cleaner Jameson flotation cell where it will be combined with the scavenger cleaner concentrate, depressant and frother in the feed box. The LG cleaner tailings will flow to the scavenger cleaner feed box by gravity.

LG Jameson cell re-cleaner concentrate will be pumped to the concentrate thickener. The LG re-cleaner tailings will gravitate to a collection sump before being pumped back to the LG cleaner feed box or alternately to the scavenger cleaner feed box, dependant on the plant operational parameters and concentrate grade requirements. This routing will be done with a valve arrangement.

Scavenger rougher concentrate and LG cleaner tailings will report to the scavenger cleaner feed box where depressant, collector and frother will be added. The scavenger cleaner circuit has a design residence time of 62.5 minutes and is comprised of 5 x 130 m³, mechanically agitated forced air, flotation tank cells.

Concentrate from the first scavenger cleaner cell is collected in the scavenger cleaner concentrate collection box that is equipped with dart valves allowing the concentrate stream to either be routed to the LG cleaner concentrate sump or alternately to the scavenger cleaner concentrate sump.

The concentrate from the remaining cells will be collected in a sump and recycled back to the LG cleaner feed box, or alternately to the scavenger cleaner feed box, dependant on the plant operational parameters and concentrate grade requirements. This routing will be done with a valve arrangement.

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 Dewatering and Product Handling

The new 3.3 Mtpa concentrate dewatering and product handling circuit for the Phase 2 capacity expansion will have the same overall flow sheet that has been adopted for the 0.77 Mtpa Phase 1 flotation circuit.

The combined HG cleaner, MG cleaner and LG re-cleaner flotation concentrates will report to a concentrate thickener via a sampler and trash removal screen. The screened concentrate, together with diluted flocculant, will gravitate to the concentrate thickener feed box.

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 a 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 filtrate reports to the concentrate thickener. The concentrate cake product is loaded onto trucks and sampled by an auger sampler before dispatch.

Holding large volumes of concentrate stock increases the security risk, unnecessarily increases the metal pipeline inventory and elevates the risk for dust losses. It is important to ensure regular daily dispatch to the smelter to prevent filter cake becoming bone-dry to limit dust formation. Regular smelter dispatch and good housekeeping are imperative as per other operations.

The design allows space provision to add an identical 3.3 Mtpa concentrate dewatering and product handling module in both Phases 3 and 4 respectively.

17.5.6 Tailings Dewatering and Transfer

The combined scavenger rougher and cleaner flotation tails from the new 3.3 Mtpa flotation module will be pumped to a Ø36 m tailings thickener via a feed box. Effluent from the backfill circuit, together with diluted flocculant, will also be added into the thickener feed box.

The tailings thickener overflow will gravitate to a Ø12 m tailings clarifier via a feed box, along with any return water (pumped) from the DTSF. Diluted flocculant will be added to the tailing's clarifier feed box and centre well to assist in the flocculation of the solids. The tailings clarifier overflow will gravitate to a process water tank for recycle as process water in the concentrator.

Both the tailings thickener underflow and tailings clarifier underflow at 60–65% solids (w/w), will be pumped to an agitated backfill feed tank via a final tailings sampler. Additionally thickened tailings from the 0.77 Mtpa Phase 1 concentrator, at 60–65% solids (w/w), will also be pumped to the backfill feed tank.

The combined thickened concentrator tailings in the backfill feed tank will be pumped to the backfill plant at a slurry solids concentration of 60–65% (w/w). When the concentrator plants are producing more tailings than required in the backfill CPF mix, or when the backfill, plant is down for maintenance, the backfill tank will fill up and overflow directly into an agitated tailings transfer tank. The thickened tailings will be diluted to 50–55% solids (w/w) and pumped to a new tailings filter plant located at the Rietfontein DTSF.

The combined tailings will be pumped to the Rietfontein DTSF via a booster station using duty/standby pump trains. Each pump train at both the concentrator plant and booster station, will consist of four centrifugal pumps in series with a dedicated tailings pipeline per pump train. The last pump in each train will be fitted with a variable speed drive and will always be running, while the other pumps will be controlled based to target the required flow rate. The booster station will deliver the tailings to the filtration plant thickener feed box.

Allowance has been made for gland seal service water (GSW), make-up water and flushing water supply as well as spillage handling facilities.

The design includes space provision to add additional thickeners, clarifiers and tailings pump trains and booster stations for the future 10.67 Mtpa capacity expansion.

17.5.7 Tailings Filter Plant

The new tailings filter plant for the Phase 2 capacity expansion will be located at the Rietfontein DTSF.

The combined 0.77 Mtpa and 3.3 Mtpa concentrator tailings slurry, at 50–55% (w/w) solids will be pumped to a Ø38 m thickener via the booster pumping station. Filtrate from the filtration circuit, together with diluted flocculant and thickener overflow recycle will also be added into the thickener feed box, resulting in a diluted feed measuring approximately 15–20% solids (w/w).

The tailings filter plant thickener underflow at 60% solids (w/w) will be pumped to 2 x 500 m³ filter feed tanks. The overflow product from the thickener will provide process water for the tailings filter plant.

Thickened slurry from the filter feed tanks will be pumped to the filtration circuit comprised of 4 x 816 m² plate and frame pressure filters. The tailings filter cake product, with a moisture content of 12–14% (w/w) will discharge onto a transfer conveyor, which feeds a tailings load out conveyor that discharges to the DTSF. The tailings transfer conveyor throw out stockpile, stacking conveyor and tailings filter cake reclaim and handling are within the DTSF scope. The tailings filtrate will be recycled to the filter plant feed thickener.

The design includes space provision for additional thickening and filtration capacity for the future 10.67 Mtpa capacity expansion.

17.5.8 Water Circuit

Process water is stored in a process water tank, which is fed with tailings clarifier overflow water and excess flotation spray/wash water. The 3.3 Mtpa Phase 2 milling and flotation module will be equipped with a dedicated process water pumping installation. Clean water from the clean water tank provides for the process water make-up requirements.

The Masodi water treatment plant will filter water from the Masodi water supply system which will be used as fresh make-up water for the clean water circuit. A potable water treatment plant, located within the infrastructure scope will treat borehole water to produce potable water, which is distributed throughout the mine site.

A dedicated fire water tank located in the concentrator plant area supplies fire water for firefighting purposes.

17.5.9 Air Services

Low pressure blower air for the flotation circuits will be 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.

The drying air for both the concentrate and tailings filtration circuits will be drawn from dedicated compressors and air receivers. Filter-pressing air is supplied by high-pressure compressors delivering compressed air at approximately 1,600 kPa(g). The compressors and air receivers are situated adjacent to the relevant filtration building located at the concentrator (concentrate filtration) and DSTSF (tailings filtration).

17.5.10 Reagents

Phase 2 will include the addition of dedicated reagent systems for the new 3.3 Mtpa milling and flotation module. The design is aligned to the approach adopted for Phase 1 and 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 will be received in bulk and transferred into a dry silo. 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.

Liquid SIPX collector will be received in bulk and transferred into a receiving tank. The collector from the receiving tank will be transferred to an agitated mixing tank and diluted to the required dosing strength of 15% w/v using reagent water. After mixing, the collector solution will be transferred to a collector day tanks, from where it will be pumped to the respective dosing points.

Copper collector, promotor and frother will each be received in 1 m³ intermediate bulk containers (IBC), added manually into a dosing tank, and pumped to the respective dosing points.

The SIPX collector and depressant reagent mixing tanks for Phase 2 have been sized based on the requirements for the future 10.67 Mtpa expansion, while the dosing/day tanks have been sized for a single 3.3 Mtpa module. Additional dosing/day tanks will be added with each additional module. Similarly additional dosing tanks will be added for copper collector, promotor and frother with each additional module.

The flocculant reagent mixing systems will be expanded in future with the addition of a new system for each 3.3 Mtpa milling and flotation and tailings filtration module.

17.5.11 Sampling and Analysis

The design includes automated samplers on the mill feed, flotation feed and flotation tailings and an auger sampler on the final concentrate stream for metallurgical accounting purposes. In addition, the design includes online analysis of the final concentrate and flotation circuit Jameson cell concentrate streams to allow for real-time analysis of the concentrate grade.

A sample preparation facility is located at the concentrator with a large, centralised laboratory on the mine site to service the mine and concentrator.

17.5.12 Control Philosophy

High level concentrator plant control philosophy and preliminary 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 the 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 suppression has been considered across the stockpiles, rock handling and processing 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 concentrator process 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 Phase 2 concentrator plant has been designed in accordance with the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities. The modular expansion approach for 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 temporary construction electrical supply of 5 MVA and a 100 MVA supply was added as part of Phase1 of the project. The 5MVA construction supply will be decommissioned once all the changeovers to the 100 MVA supply point are complete. These change-overs are in progress and scheduled for completion by June 2025.

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 waste water 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). Each of the buildings and or facilities has its own associated terrace; there are thus no bulk earthworks terraces.
- Concentrator plant infrastructure: All supporting infrastructure located within the concentrator plant area (i.e., roads, buildings, process water dams, services, terraces, and electrical reticulation).
- General site infrastructure: Bulk supply infrastructure found within the waste management area, the general offices area and all supporting infrastructure, which links the three areas together (site-wide). Additionally, the bulk supply of water, tailings piping, return water pipeline and power is 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.
- Ivanplats Platreef 4 Mtpa Phase 2 Feasibility Study Report - Section 10 - Surface Infrastructure, DRA Report Number JZADXR7434-STU-REP-075, January 2025, by DRA Projects.
- 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
- Crushed G5 and G7 material for platform construction.

18.1.3 Surface Structure

The buildings have been classified into the following categories:

- Brick and mortar buildings.
- Structural steel and cladded buildings.

Prefabricated electrical 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.

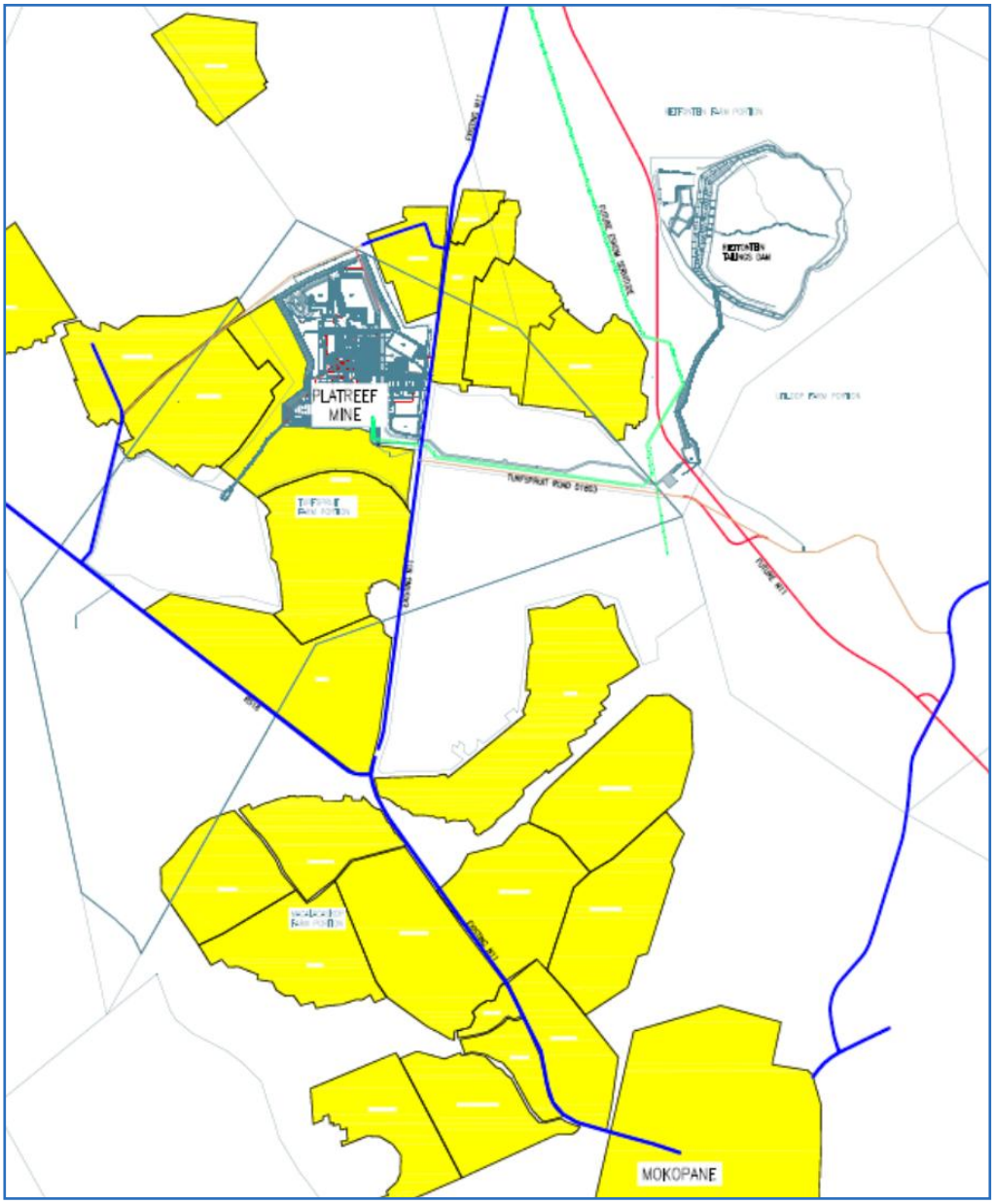
18.2 Surrounding Infrastructure

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

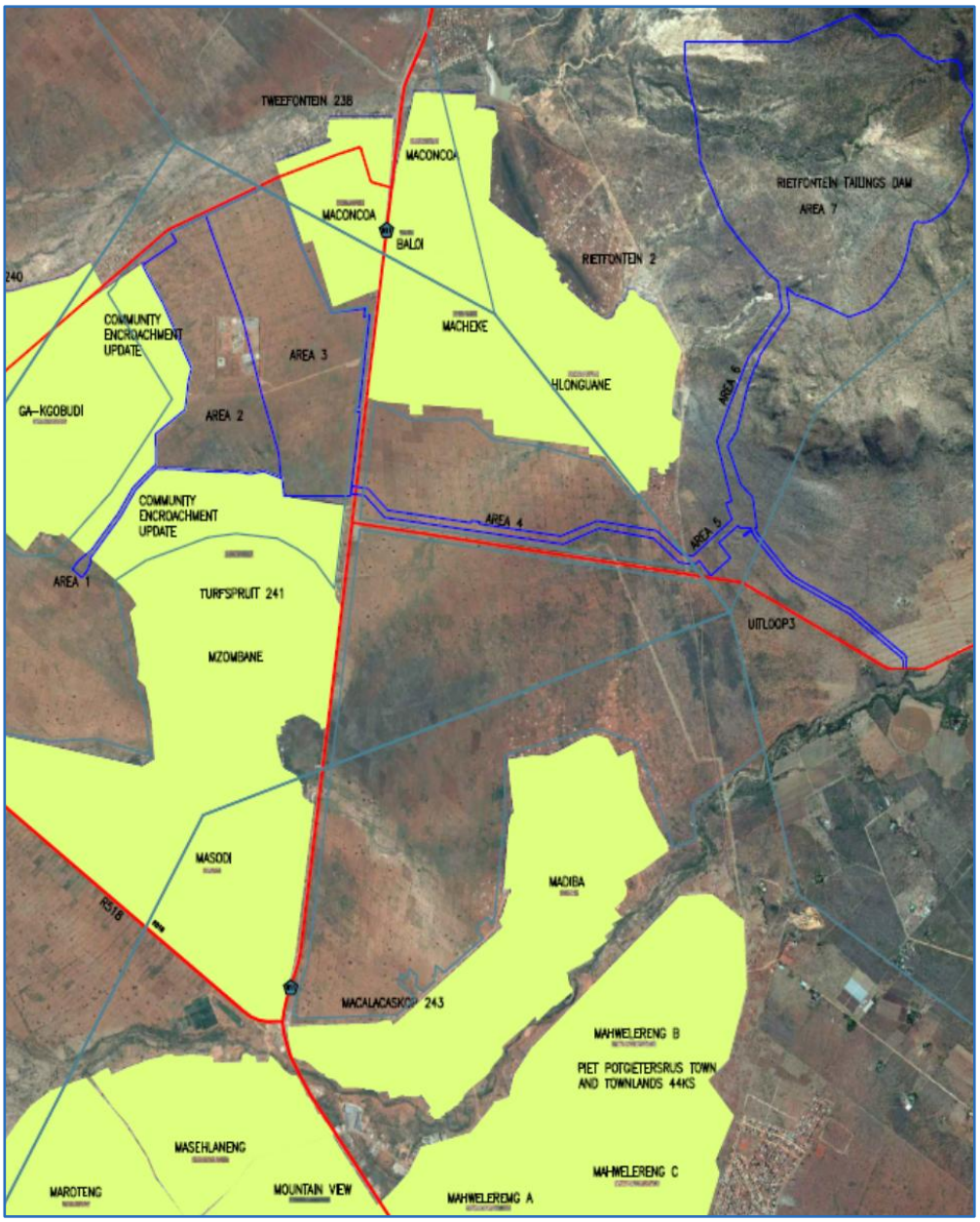
The mine lease area is situated in the Mogalakwena River valley, with the anticipated mining area situated in lease area as per Figure 18.2. Mountainous areas run to the east and west of the lease area, while the mining area itself is relatively flat. Mountainous areas are to be found in the north-eastern 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 masl in the Mogalakwena River floodplain in the south-western corner to 1,359 masl 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 updated from the previous feasibility studies in 2017 and 2022 and were developed with a holistic view of the complete mine lease area and considering directly affected local communities. The original plans were adapted to suit the phased implementation and operation of the Project, taking cognisance of the existing infrastructure on the site as well as the Phase 1 and Phase 2 construction activities currently underway.

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 includes 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 include Shaft 2, which during steady state operations, will hoist men and material whilst Shaft 1 will hoist waste and low grade material and Shaft 3 will be used to hoist ore. The remainder of the infrastructure will be constructed in order to support the larger mining operation, the additional 3.3 Mtpa concentrator plant and the ramp up of mining production to steady state.

Layouts for the mining area include a shaft area, concentrator plants, DTSF and various general infrastructure sections. All layout design work was undertaken in close cooperation with the Platreef project owner's team.

The following major factors specifically influenced the determination of the plot and block plan layouts:

- Allowance of sufficient footprint for further expansion of the and Phase 2 concentrator plant.
- Department of Water and Sanitation (DWS) legislative requirements prohibiting infrastructure from being located in the 1 in 100-year flood lines.
- The proximity of neighbouring communities and their potential encroachment onto mine lease areas.
- Various licence applications and environmental and social requirements.

The Phase 2 concentrator plant was positioned on the plot plan, allowing future space for an expansion, with the position of the DTSF identified at Rietfontein. A suitable tailings pipeline corridor was pinpointed and placed onto the plot plans and routed accordingly to minimise grave relocation and heritage site impacts.

18.4.2 Plot Plans

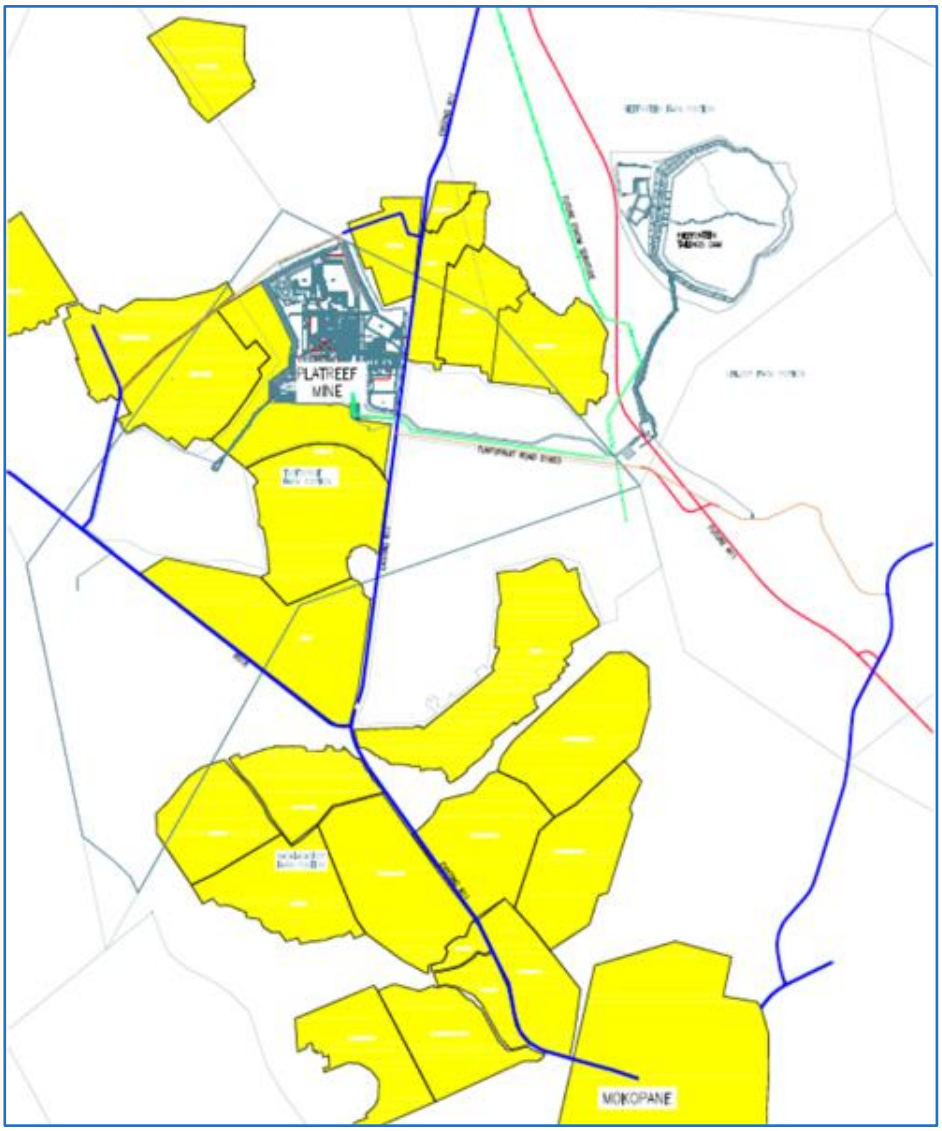
The plot plans give consideration to three main regional areas:

- The Platreef mine
- The Rietfontein TSF Site

The Tailings Pipeline servitude connecting these two areas

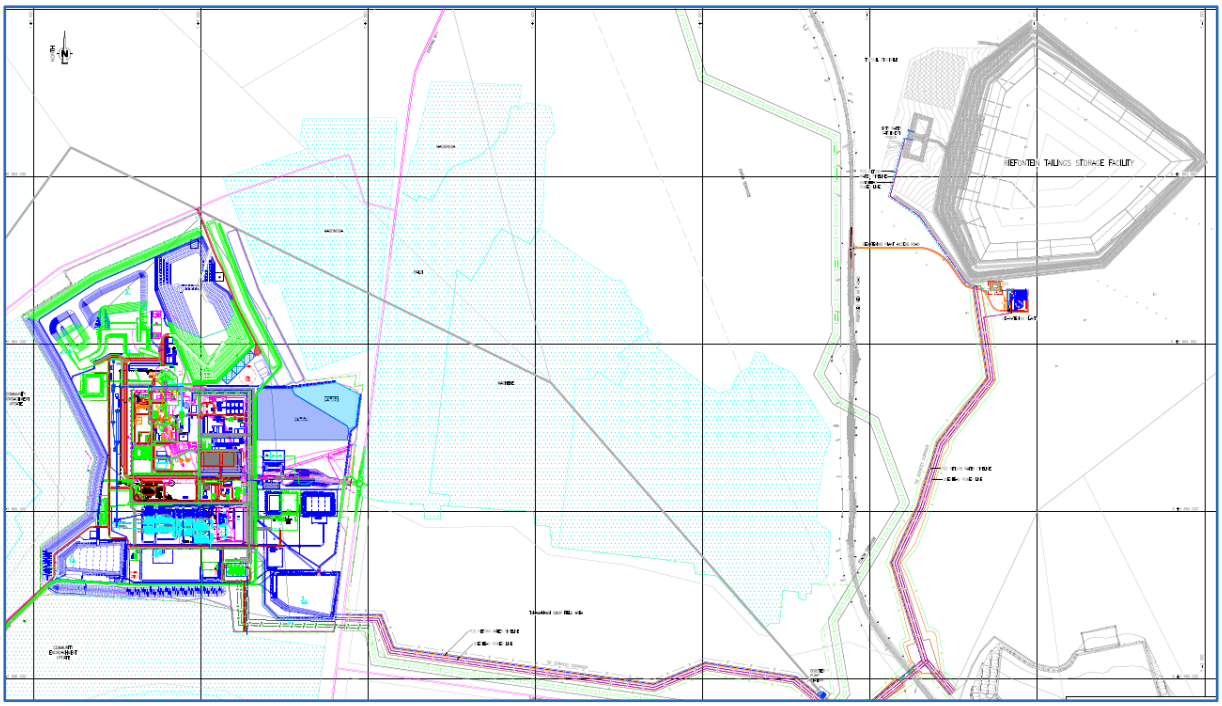
Figure 18.3 indicates the Platreef Project's location within the local region and surrounding communities whilst Figure 18.4 depicts the overall plot plan of the Platreef Mine site, Rietfontein TSF and the tailings pipeline servitude.

Figure 18.3 Platreef Project Location within Local Region



DRA, 2022

Figure 18.4 Mine and Rietfontein TSF Area Plot Plan



DRA, 2025

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 which incorporates the surface infrastructure for the various project phases.

The current Phase 2 FS mining area block plan has been updated from the previous feasibility study block plans with the following key changes being implemented:

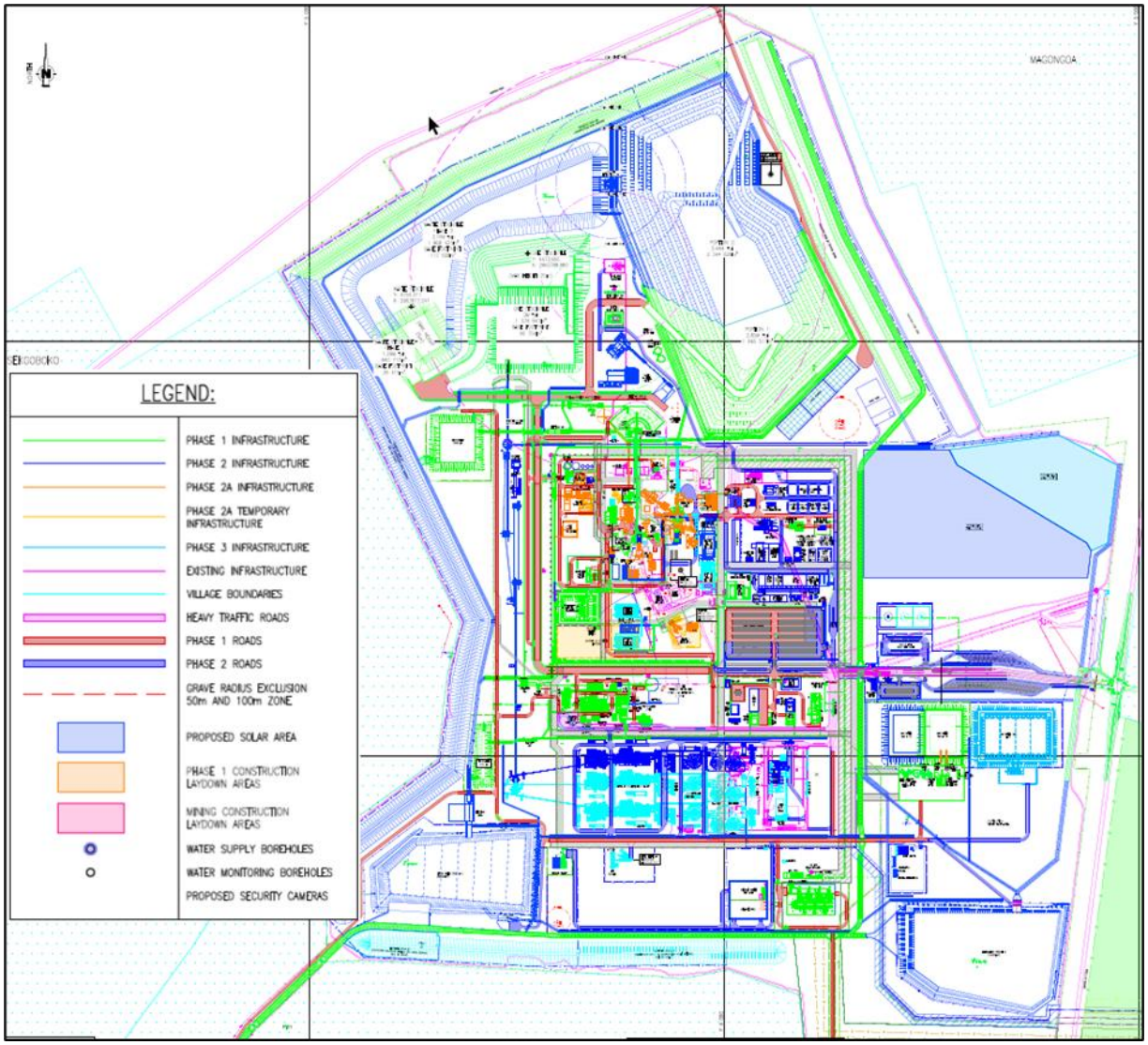
- The locations of the waste stockpile and ore stockpile have been switched.
- The paste backfill plant and paste fill cubbies have been relocated adjacent to the DTSF.
- The change house will be done in two stages, with two separate buildings being constructed. The first building will be part of Phase 2, and the second building will be constructed as part of a future phase.
- The shaft office is a duplication of the main office, although neither form part of the Phase 2 study.
- The core yard has moved to south of the DTSF.

- The main entrance has changed significantly with the removal of the traffic circles and the inclusion of the incoming and outgoing truck parking, bus and taxi drop-off, helipad and gate house.
- The concentrate filtration plant has moved to the area east of the thickener.

The following infrastructure considered previously in the 2022 feasibility study have been removed from the current Phase 2 study scope:

- Leach Pond
- Compost Tip
- Material Recovery Facility
- Waste Tyre Store
- Landfill
- Berm 2

Figure 18.5 Mine Plot Plan



DRA, 2025

An irregular-shaped fence line and perimeter berm indicates the available footprint to the west of the N11, wherein the Phase 2 Platreef 4.07 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 bulk water entering adjacent to the N11 intersection access road to the mine. The treated municipal grey water enters 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.

Storm water and pollution control systems are integrated in the mining area in adherence to the Integrated Water Use Licence Application (IWULA) requirements and suitably located for the site topography. These systems comprise large drains and channels (mostly running north to south), a stormwater attenuation pond in the south-eastern corner (attenuating clean stormwater runoff from upstream of the Mine) and a stormwater pollution 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.

Perimeter berms are located on the southern, western and northern perimeters of the mining area to mitigate visual impacts, dust and noise in accordance with the Environmental Management Programme (EMP)specialist study requirements for the IWMLA. Waste rock and ore stockpiles are also situated in the northern section of the mining area.

The positions of Shaft 1 and Shaft 2 are fixed and are located towards the centre of the mining area. An area approximately 400 m long and 500 m wide contains 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 Plant immediately south of the shaft area and the 4.4 Mtpa (and space allocation for further expansion) south of this.

18.5 Tailings Storage Facilities

The Phased Development Plan will see the development of two Dry Stack Tailings 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. As part of the Platreef 2025 FS, Ivanplats (Pty) Ltd (Ivanplats) appointed WSP Africa (Pty) Ltd (WSP) to undertake the engineering FS design for the Phase 2 (Rietfontein) DTSF.

The below sub-sections contain further details regarding the Phase 1 and Rietfontien DTSF.

18.5.1 Phase 1 DTSF

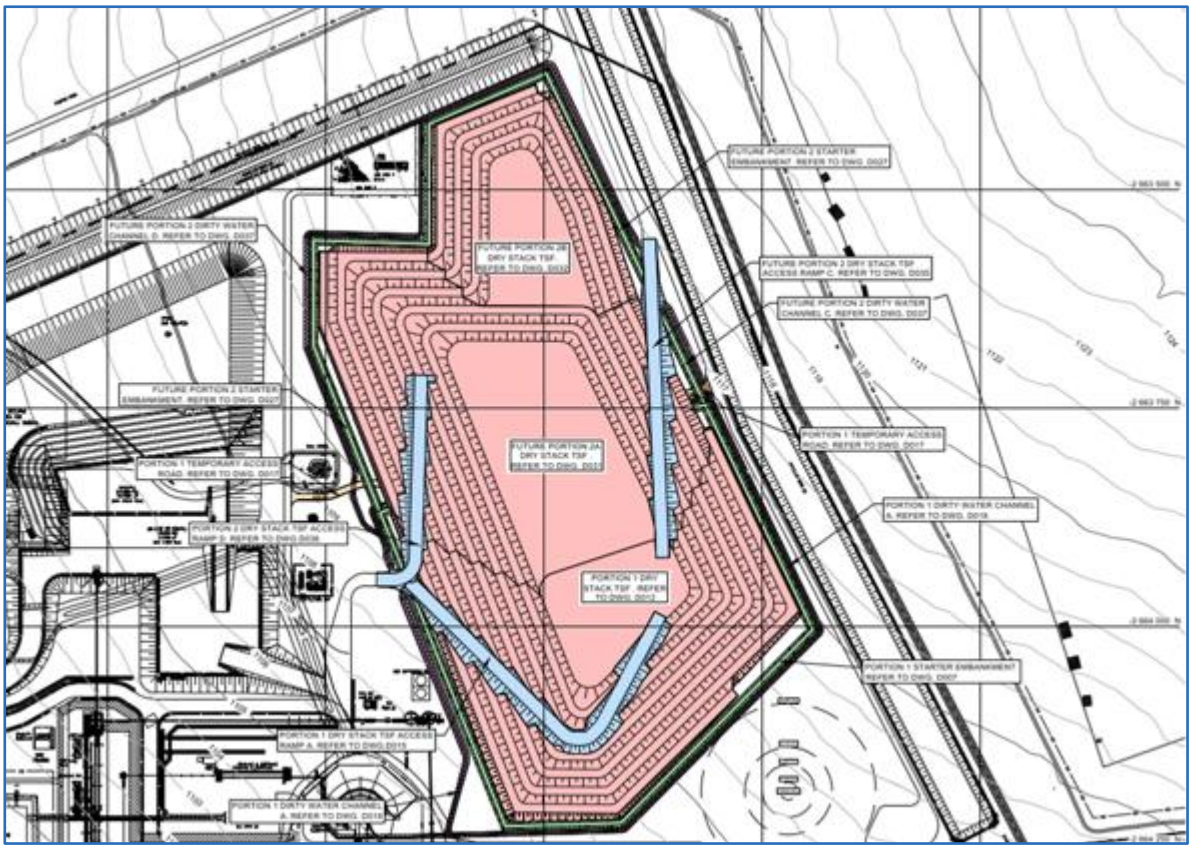
The Preliminary Economic Assessment (PEA) completed in 2020 for the Platreef Phased Development plan proposed to use the site previously allocated for a waste rock dump as the Phase 1 DTSF. This concept was developed further in the Platreef 2022 FS. Following the completion of the 2022 FS, the detailed design for construction of the DTSF was completed in 2023 by WSP. The Phase 1 DTSF is located at the Mine site, shown in Figure 18.6.

The design of the Phase 1 DTSF includes three portions, namely, Portion 1, Portion 2A and Portion 2B. The construction of Portion 1 of the Phase 1 DTSF is currently underway. The capacity and life span of each portion of the Phase 1 DTSF is summarised below:

- Portion 1:
  - Capacity: 3.05 Mt
  - Lifespan: 4.3 years
- Portion 2A:
  - Capacity: 2.90 Mt
  - Lifespan: 4.1 years
- Portion 2B
  - Capacity: 0.57 Mt
  - Lifespan: 0.8 years

The updated production schedule derived from the Platreef 2025 FS stipulates that only the first three years of tailings production will be accommodated by the Phase 1 DTSF and the remainder of the life-of-mine production tonnages will be sent to the Rietfontein DTSF for storage. Therefore, only Portion 1 of Phase 1 DTSF will be constructed.

Figure 18.6 Phase 1 DTSF Layout



18.5.2 Phase 2 (Rietfontein) DTSF

Initially, a hybrid paddock deposition methodology was proposed. However, Ivanplats decided to change the deposition methodology from upstream design to dry stacking during the Platreef 2022 FS. Due to the change in tailings production schedule (plant expansion to 4.07 Mtpa), the engineering design for the Rietfontein DTSF was then revised and updated by WSP as part of the Platreef 2025 FS. The Rietfontein DTSF was not included in the Platreef 2025 PEA or Phase 3 of the Project (plant expansion to 10.67 Mtpa). However, if Ivanplats implements the 10.67 Mtpa plant expansion, the maximum capacity will likely remain the same, but the lifespan will be reduced.

The Rietfontein DTSF site is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed DTSF site was chosen following a site selection study and is considered to be a feasible site based on all applicable engineering and environmental standards for tailings storage facilities. The site layout is indicated in Figure 18.7.

Figure 18.7 Rietfontein Site Layout

The DTSF has an operating life of 36 years, during this time approximately 90.4 Mt of tailings will be stored within the DTSF, the remainder of the tailings will be used as backfill in the underground mine. The construction of the DTSF will follow a phased approach in order to limit the initial expenditure. The DTSF will be constructed in two lifts and four phases, presented in Figure 18.8:

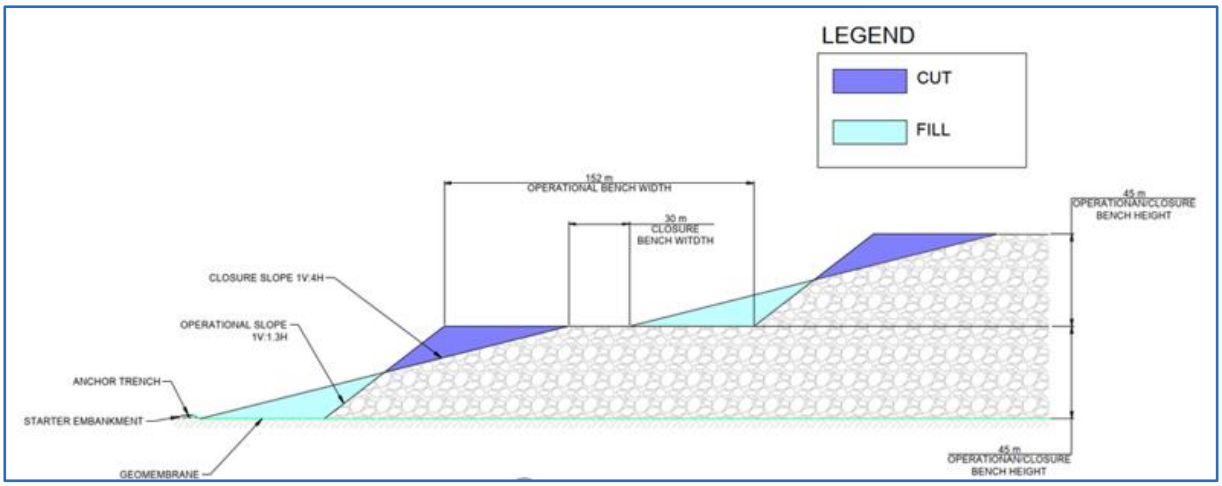
- The first lift will accommodate tailings for the first 12 years of deposition at the Rietfontein DTSF and be constructed in three phases, referred to as Portion 1, Portion 2 and Portion 3.
- The second lift will accommodate tailings from Year 13 to end of life-of-mine (LOM) and will be constructed in one phase, referred to as Portion 4.

Figure 18.8 Rietfontein DTSF Development Plan

The tailings are to be dewatered with thickeners and pressure filter technologies located at the Rietfontein site. The dewatered material will then be transported and deposited onto the DTSF from the filter plant via conveyors. The facility will be accessed via an access ramp located on the most eastern point of the DTSF.

The conveyor system will deposit the material at an angle of repose, of 1V:1.3H, and the outer slopes will be battered (cut-to-fill) to a 1V:4H gradient as part of concurrent rehabilitation. A typical cross section through the DTSF is shown in Figure 18.9, depicting the operational and proposed closure profile slopes. The bench width between the first and second lift post-rehabilitation will be 30 m. The first lift will be constructed to an elevation of 1,248 masl and the second lift will be constructed to an elevation of 1,293 masl.

Figure 18.9 Typical Cross Section through the DTSF



The DTSF is to be lined with a reinforced geosynthetic clay liner (GCL) overlain with a high-density polyethylene (HDPE) liner. A Class C barrier system will be needed due to the tailings classified as Type 3 waste. This system is compliant with the prescribed South African National Environmental Management Waste Act, 2008 (NEMWA) R636.

The following are the summary of the key features of the DTSF Phase 1:

- Compacted earth embankment structures around the perimeter demarcating the phase development.
- Perimeter fencing around the DTSF.
- Access road and access ramps.
- A barrier system.
- A leachate collection system.
- A dirty water channel.
- A subsoil drainage system.
- Stormwater management system.

18.5.3 Conclusions and Recommendations

The use of the dry stack method extracts water from the tailings prior to placement, the dewatered 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 DTSF compared with a hydraulic deposition method.

The slope stability analysis completed as part of the FS indicates that the DTSF design satisfies the stability requirements as per the Canadian Dam Association (CDA) and the Australian National Committee on Large Dams (ANCOLD) guidelines.

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.
- Review and refining of the 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 sources of bulk water during the construction period are local groundwater abstracted from licensed boreholes on Ivanplats, Uitloop and Turfspruit properties, as well as stormwater 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.

On 17 January 2022, Ivanhoe concluded an agreement with the MLM to use local treated effluent water as a bulk water supply for the Platreef project. A minimum of 3,000 m³/d will be supplied for the first phase of production at Platreef.

Under the terms of the agreement, Ivanplats provided 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 m3 for the first 10 million litres per day to offset a portion of the initial capital contributed.

A further agreement was signed between Ivanplats and the MLM on 15 June 2023 to increase the supply to the Ivanplats constructed Masodi Waste Water Treatment Plant (WWTP), by an additional 3,000m3/d via the installation of a new sewer gravity pipeline. This, together with additional sewer reticulation upgrades, will increase the sewage inflow to the Masodi WWTW up to approximately 10,000 m³/d, depending on the roll out of the supporting municipal infrastructure. The associated pumps at Masodi WWTP, and piping infrastructure to deliver the treated sewage effluent to the Platreef site, have been installed as part of the Phase 1 construction.

Although 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, Ivanplats is not currently considering participating in the Phase 2B pipeline with sufficient bulk water supply seemingly available via the agreed allocation of treated effluent from the Masodi WWTW.

18.6.2 Bulk Water Requirement

For the operational phase of the mine, an updated dynamic daily time step water balance model was applied to develop an average water balance for Phase 2 (January 2028 to LOM). The simulation shows that, on average, 9,809 m³/d water is required to operate the mine. The sources of water are an average of 5 602 m³/d bulk water supply (treated sewage effluent) from the Masodi WWTW to supply the site filtered water requirements, 222 m³/d from the four site boreholes to supply the site operational potable water requirements and the remaining 3,985 m³/d is supplied from moisture in the ore, from rainfall/runoff collected on-site in the stormwater ponds and groundwater ingress to the workings.

The bulk water requirements will increase to approximately 6,800 m³/d as the Phase 2 concentrator ramps up. The variation in the make-up volume required is due to the seasonality and annual variation of the rainfall. The reduction in the water requirement occurs during the wet season and in wet years when the rain and runoff volumes are available on-site to meet the filtered water requirements.

Provision has been made to further treat the water received from the Masodi WWTW via an on-site filtration plant for use in the process plant.

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.

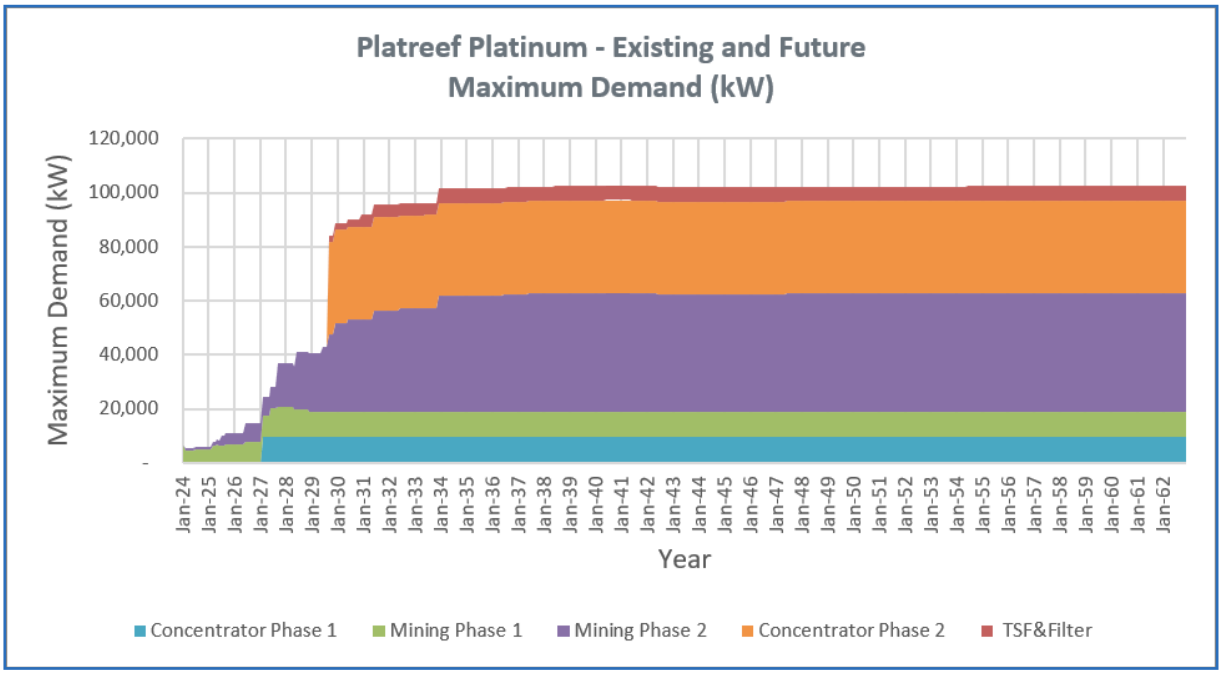
The results of these calculations are presented in Table 18.1.

Table 18.1 Notified Maximum Demand

Description	Run kW	Run kVA	Run kVAr	Target kVAr @ PF=0.96	kVA (with PFC)
Concentrator Phase 1	9,580	10,527	4,024	2,794	9,979
Concentrator Phase 2	26,330	28,352	9,680	7,672	27,425
Crushing and Screening Phase 2	8,026	8,558	2,706	2,341	8,361
Tailings Filtration and Storage	5,344	6,277	3,193	1,559	5,567
A 6000 Phase 1	519	609	307	151	541
A 6000 Phase 2	2,196	2,560	1,302	641	2,288
Backfill Phase 1	719	786	289	210	749
Backfill Phase 2	4,485	5,062	2,187	1,308	4,672
Refrigeration	5,526	6,431	3,247	1,612	5,756
Mining Services - Surface	2,448	2,919	1,582	714	2,550
Rock Handling Phase 2	2,385	2,771	1,319	696	2,484
Sinking 3#	1,352	1,690	1,014	394	1,408
Winders 1#	1,934	3,128	2,443	564	2,015
Winders 3#	4,140	5,162	3,081	1,208	4,313
Dewatering	1,248	1,472	777	364	1,300
Primary Ventilation Phase 1	1,418	1,649	843	414	1,477
Ventilation VR#4	4,449	4,620	1,201	1,298	4,635
Ventilation VR#6	4,449	4,621	1,201	1,298	4,635
Ventilation VR#7	4,450	4,622	1,202	1,298	4,635
Secondary Ventilation Phase 2	8,200	9,437	4,605	2,392	8,541
Production Phase 2	5,098	6,010	3,176	1,487	5,310
Mining Services - Underground	1,793	2,171	1,221	523	1,868
Total	**106,091**	**119,435**	**50,602**	**30,936**	**110,510**

The electrical load build-up is presented in Figure 18.10. The electrical power requirement over the LOM is presented in the operational expenditure.

Figure 18.10 Electrical Load Build Up

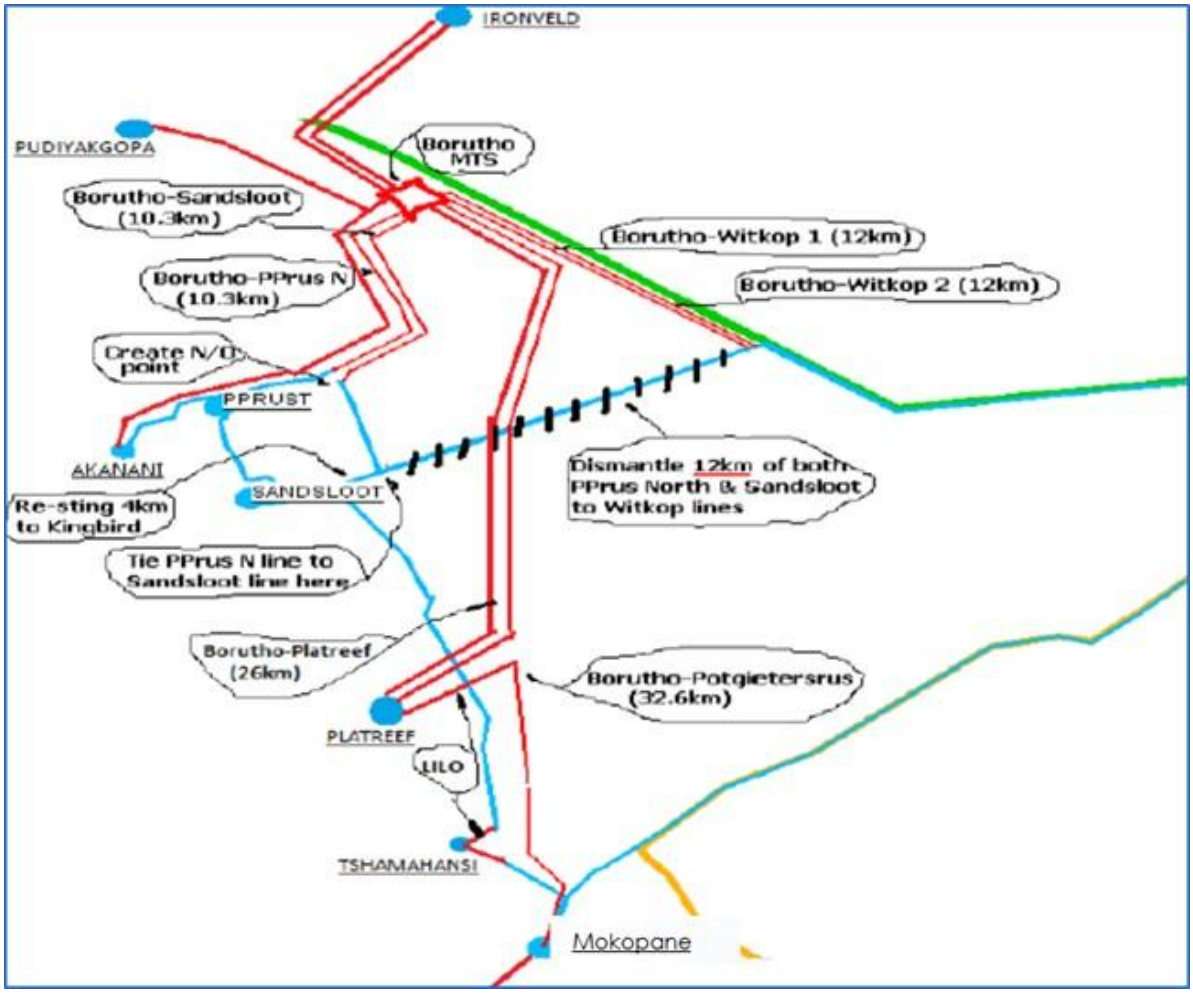


18.7.2 Bulk Power Supply

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.

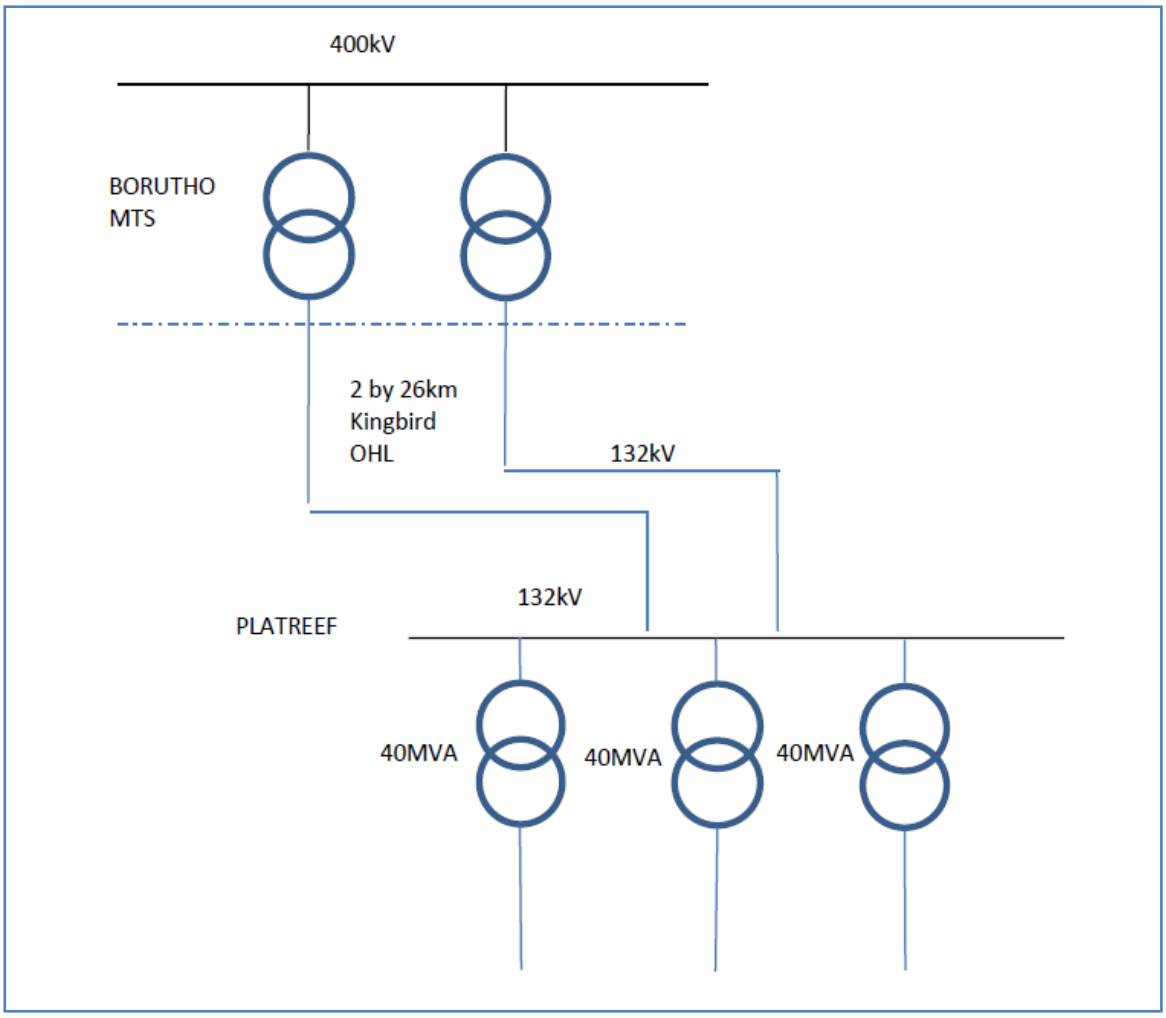
Upon completion of the works, these will be handed over to Eskom to form part of the utilities supply network. As represented in Figure 18.11 and Figure 18.12, the Platreef Project is to be fed from the Eskom Borutho Main Transmission Station (MTS). Two 132 kV overhead line (OHL) feeder bays have been provided by Eskom.

Figure 18.11 Eskom Supply Network



ESKOM, 2017

Figure 18.12 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.

Construction of the overhead lines and substation for the 100 MVA supply commenced during Phase 1 and commissioning was completed August 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

Ivanplats has entered into a 10-year agreement with an independent power producer (IPP) for a minimum commitment of 48,300 MWh per annum of solar power. The solar power will be wheeled through the Eskom network. The reduction in the cost for the solar power is allowed for in the electrical operating expenditure (Opex) cost. The scheduled start date for the wheeling is June 2026.

It is anticipated that there will be a similar supply agreement when the initial agreement runs out after 10 years.

In addition to the renewable energy supplied by the IPP, the construction of an on-site 9,589 MWh per annum solar power plant is in progress, and the scheduled commissioning date is January 2025. The cost savings due to the on-site solar power plant is included in the electrical Opex cost.

18.7.4 MV Distribution to Surface Substations

Power will be distributed from the 33 kV Main Intake substation (33 kV Consumer substation) to the following substations at 33kV. A dual supply to each substation is allowed for:

- Mining 33 kV substation.
- Concentrator 33 kV substation.
- VR6 and VR7 ventilation fan substations.
- Tailings booster substation.
- Tailings filtration and dry stack storage facility.
- Backfill filtration and paste plant.

18.7.5 Construction Power Supply

The temporary 5 MVA construction power supply was sourced at 33 kV from Eskom, the South African national power utility. This temporary supply will be decommissioned once all the loads have been changed over to the new 33kV substation. Completion is scheduled for June 2025.

18.7.6 Emergency Power

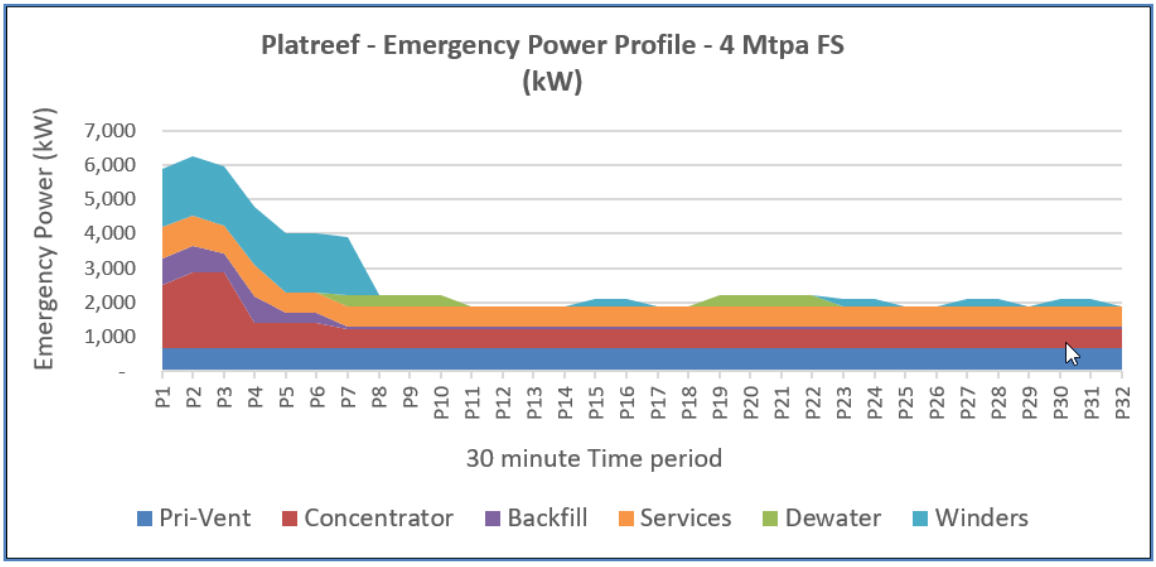
The capacity of the existing generator power plant was reviewed, and the standby power is adequate to comply with the requirements of the Mines Health and Safety Act in terms of standby supply for winders and main ventilation fans. The available generator plant consists of 3x1 MW generators and 2 x 1.25 MW generators, totally 5.5 MW.

A summary of equipment requiring emergency power is listed in Table 18.2, and the emergency power profile is indicated in Figure 18.13.

Table 18.2 Electrical Power Requirements

Description	KW
Shaft 1 Man Winder	1,711
750 m Level 2 x 110 kW Booster Fans 850 m Level 2 x 110 kW Booster Fans 950 m Level 2 x 110 kW Booster Fans	500
4 x 160 kW Scamont Dewatering Pump on 950 m Level	307
All Slurry Tank Agitator Drives in the Concentrator Plant Flushing of Tailing Disposal lines	1,848
Slurry Tank Agitator Drives in Backfill Filtration and Paste Plant Flushing of Backfill Lines	779
General Services	903
Total	**6,048**

Figure 18.13 Electrical Profile – Emergency Power



Note: the timescale is in 30-min intervals

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

A geotechnical investigation was previously completed by SRK Consulting (South Africa) (Pty) Ltd (SRK) (project number 489587 Rev 1, "Geotechnical Investigation for Platreef Mokopane DFS: Plant Area", hereafter referred to as the surface geotechnical report). The report indicated that the observed collapsible colluvial soil typically present in the upper 0.3 to 2.6 m of the site soil profile is unlikely to have a safe bearing load capacity of 150 kPa. It further recommended that that all applicable colluvial material be removed to spoil. Residual granite rock was also encountered below the colluvial material.

Shallow rock outcrops were encountered on the southern part of the site, including at the concentrator plant and stormwater pond no. 3 locations. The Phase 1 construction methodology entailed using Nonex (fast burning of pyrotechnic compound instead of explosive detonation) and mechanical rock breaking to remove these rock outcrops. This was due to the proximity of the neighbouring villages and licencing and permitting constraints. This had a major adverse impact on earthworks cost and duration.

To mitigate the cost impact of hard rock breaking in lieu of blasting, the FS Phase 2 concentrator terrace level was lifted to prevent excessive hard rock excavations. The lifting of the terrace levels did not impact the FS designed buried services.

The option to proceed with traditional rock blasting has been used in the pricing of the earthworks scope with the caveat that the blasting permit will be in place in time to proceed with this method. It must also be noted that a contingency amount has been catered for if the blasting permit is not in place by the time the construction commences, and rock breaking reverts to using Nonex.

18.8.3 Perimeter Berm

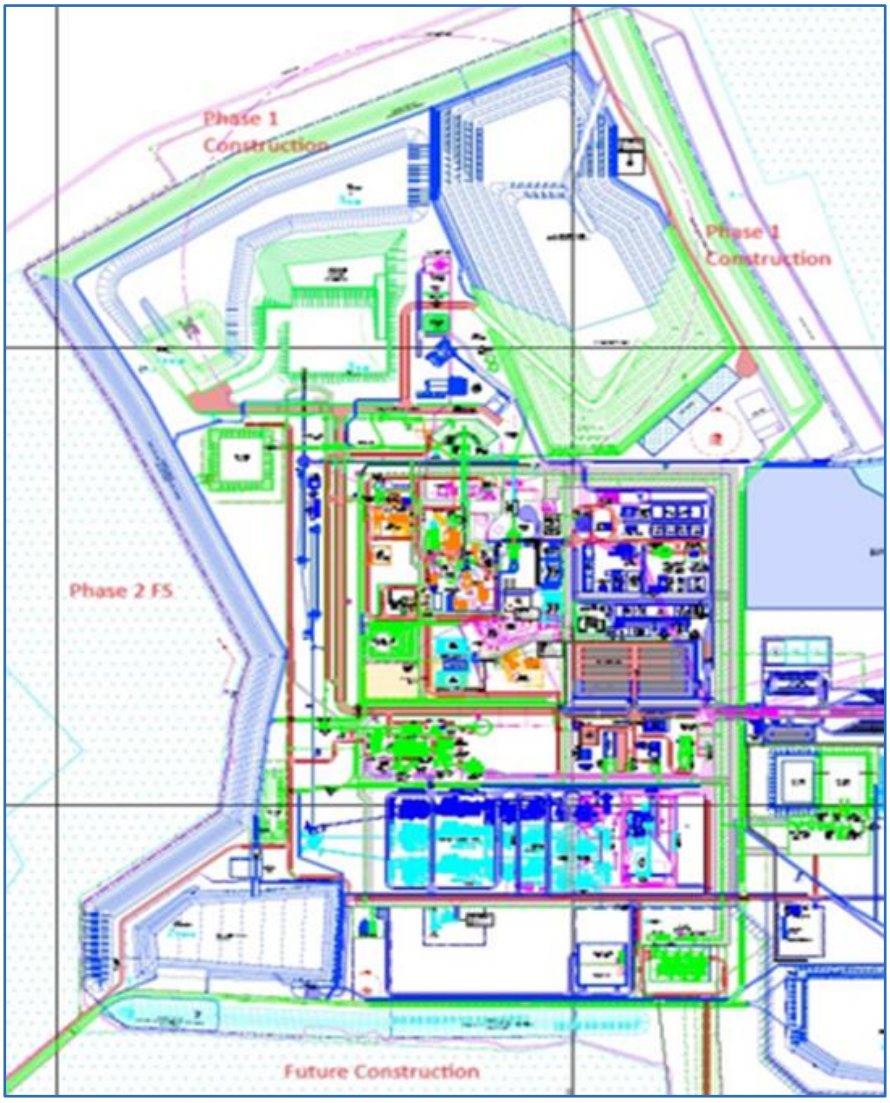
To mitigate the noise, dust and the visual impact of the mine, the relevant specialist studies were performed, modelling an approximate 7.5 to maximum of 10 m high berm on the periphery of the mining area as shown in Figure 18.14. The since-approved Environmental Management Plan (EMP) and subsequent Environmental Authorisation (EA), have both been granted on the notion that this berm will be constructed.

In addition, one of the key requirements of the visual impact study is to clad the berm with topsoil and to hydro-seed the completed sections as the berm is being built.

This perimeter berm doubles as a suitable stockpile for overburden generated during construction as well as surplus waste rock from underground development.

The perimeter berm is divided in four sections as shown in Figure 18.14 with the sections scheduled to be built at different times. Sections 3 and 4 are being constructed in Phase 1 construction, with Section 1 forming part of the Phase 2 implementation and Section 2 to be built during the final construction stages of the project.

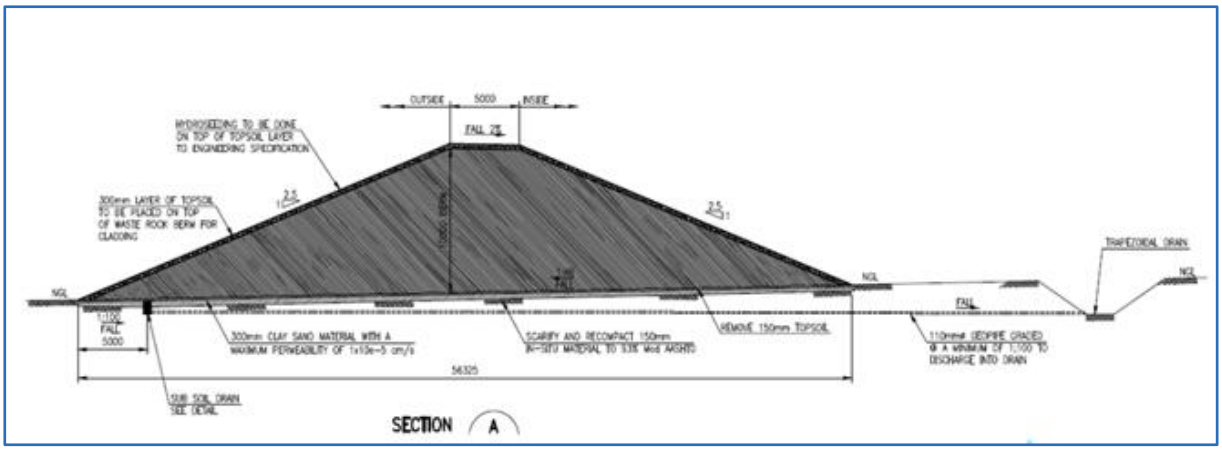
Figure 18.14 Perimeter Berms



DRA, 2025

Figure 18.15 shows a typical cross section through the berm.

Figure 18.15 Typical Berm Section



This design still includes flexibility to change the sequence of building the berm should later developments require the Project to do so.

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)
- Crushed G5 and G7 material for platform construction

The overall stockpiling strategy in terms of design and operation has been undertaken in line with the EMP requirements, for minimising dust and visual impact. The height at which material is discharged from conveyors for stockpiling purposes is to be kept at a minimum.

Materials deposited onto temporary stockpiles are spread further by means of mobile earthmoving equipment (trucks, loaders, and dozers) to build individual-type stockpiles.

Stockpiles have been designed with the required footing and surface drainage as per the project specification, as stipulated in the approved water usage license from the DWS.

A further requirement for waste rock is to reuse this as earthworks backfill material. To cater for this, waste rock will be crushed and screened to suitable-sized materials.

The earthwork contractor will establish a mobile crushing and screening plant.

18.8.5 Earthworks Commodity Flow Philosophy and Strategy

On account of the various earthworks commodities required during the construction phase, a basic commodity flow model was developed for the project, as this directly impacts on the overall capital estimate. The objective of this model is to indicate the estimated nett quantities of material/ commodities requiring to be stockpiled, whether temporarily or permanently. It further indicates commodity requirements and whether the importation of commodities is required onto the site.

The different type of earthworks commodities relevant are as follows:

Topsoil

Topsoil is defined as the first 150 to 200 mm of material removed (cut to stockpile) from any excavation done on surface. The remainder of the topsoil needs to be stockpiled in a defined area for topsoil storage only.

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. The depth of subsoil within the excavation is determined by either the bottom of the excavation level or when the refusal level (hard material) has been intersected. Subsoil could also be used in certain circumstances as fill material.

Hard Excavation

Hard excavation is the removal of material within the excavation which cannot be efficiently extracted or loaded by a track-type excavator. This type of excavation generally includes excavations in material such as formations of unweathered rock, 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.

The traditional drilling and blasting method will be adopted in the removal of the outcrops of hard rock instead of the Nonex and mechanical procedure being used on Phase 1 execution. The two primary reasons for the change in methodology stem from:

- High cost of high costs associated with using Nonex.
- The slow rate of progress associated with Nonex.
- The main risk with implementing the drilling and blasting method lies in obtaining the blasting permit in time, prior to it being needed. In the Phase 1 construction, blasting was prohibited due to permitting and licencing approval constraints.
- A contingency sum has been set aside if the Nonex option is required to be implemented.

G5 and G7 Material

G5 and G7 are types of materials used in layer works that comply with certain material properties and classification, which could be obtained naturally from a borrow pit or rocky type of material that requires crushing. With no borrow pits available, all G5 and G7 materials required will be obtained from waste rock and blasted rock that is to be crushed. This material will be obtained on site by establishing a crushing plant to crush available waste rock.

Waste Rock

Waste rock is material produced from underground mining activities and hoisted to surface with a 3PE+Au content below the mine cut-off grade. This material will be routed to the waste rock stockpile.

Ore

Ore is material 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 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

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 m³ 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.

- Cut to stockpile:
  - Cut to stockpile is loading of material generated from excavations, hauled, dumped, and stored in a dedicated spoil or stockpile area.
- Cut to fill:
  - 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 total volume of cut material
  - There is approximately 1,2 million m3 shortfall of fill material needed for the road works as well as the building foundations.
  - The planned strategy will be crushing and screening of the waste rock from the underground mining development and using this as the required fill material.

18.8.5.3 Underground Earthworks Material Flow

Ore and waste rock will be hoisted to surface, the ore will be directed to the ore stockpile or the Concentrator Plant, while waste rock will be directed to the waste stockpile from where it will either be used for the berms or sent to the screening and crushing plant for engineered fill.

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 (m^3)	Required End of Phase 1 (m^3)	Required for Phase 2 (m^3)	Required in Final Construction Phase (m^3)	Required LOM (m^3)	Total Spare (+) or Shortfall (-) (m^3)
Perimeter Berm 1	532,101	0	532,101		532,101	+1,573,162
Perimeter Berm 2	234,802	0		234,802	234,802	+1,338,360
Perimeter Berm 3	300,000	300,000			300,000	0
Perimeter Berm 4	340,000	340,000			340,000	0
Perimeter Berm Total	1,406,903	640,000	766,903		1,406,903	0
Phase 1 Waste Rock Stockpile	2 368,421	660,713			660,713	0
Phase 1 LG Ore Stockpile	52,631	52,632			1,578,947	
Phase 2 Waste Stockpile	2,105,263	0	1,200,000		1,968,421	0

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 640,000 m³ surface materials (topsoil, subsoil and hard material) and 766,903 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 N11 is a paved, dual carriage road intersecting with the N1 at Mokopane and the N4 near Middelburg. The N11 is the main access road passing the site, and as this is a national road, it falls under the jurisdiction of South African National Roads Agency Limited (SANRAL). An intersection with the N11 onto the mine property has been constructed in line with SANRAL requirements, which allows for heavy vehicles to turn north and south back onto the N11. Access to the planned DTSF is from a new intersection planned on the future N11 bypass road. The access road to the DTSF and new tailings filtration plant will be an asphalt-surfaced road to accommodate day-to-day maintenance traffic.

Delivery vehicles will travel mostly on the main entrance road to the required delivery points and parking areas, with internal roads reserved for the delivery of equipment from stores to the applicable work area. The internal roads and parking take account of the traffic flow inside the mine area. Security gates separate areas in order to control access without reducing serviceability and production.

Different types of surface finish and layer works have been designed for differing types of applications, and road uses within the mine area.

The following have been used as part of the mine design and layout:

- Asphalt road layer works are used for the main entrance access road (Figure 18.16).
- Paving road Type 1 layer works are used for internal access roads in and around the shaft, concentrator plant, workshop, and store areas (Figure 18.17).
- Gravel road layer works are used mainly in the outer portions of Areas 3000 and 6000 for maintenance access (Figure 18.18).
- Haul road layer works are used for hauling required in the rock handling area between the different types of stockpiles as well as for hauling dry tails from the 770 ktpa concentrator plant to the DTSF during Phase 1 (Figure 18.19).

- Service road layer works are used for security access road around the perimeter and south-western clean water discharge drain no. 3, with minimal expected traffic (Figure 18.20).

Figure 18.16 Asphalt Road Layer Works

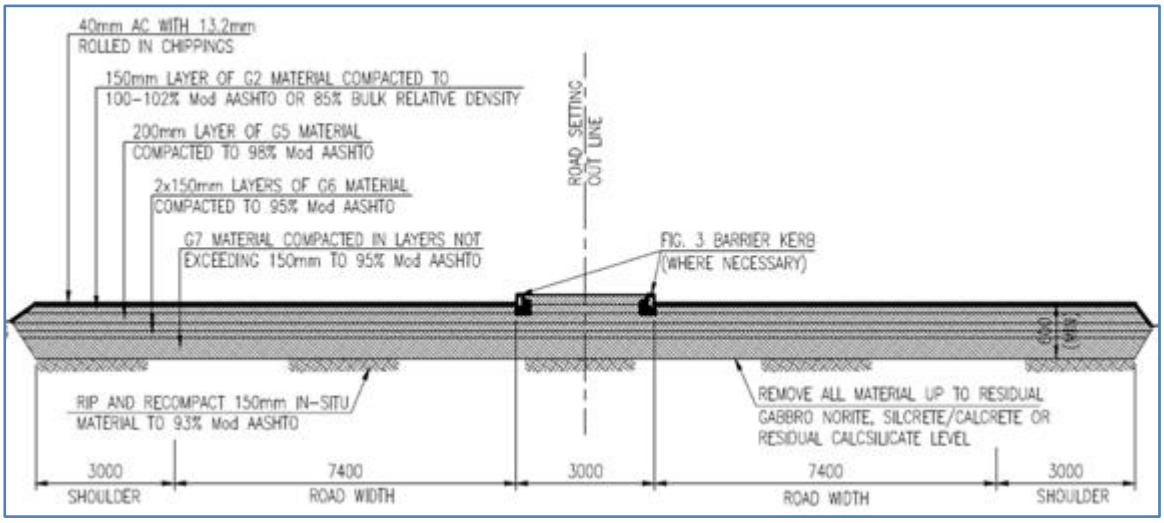


Figure 18.17 Paving Road Type 1 Layer Works

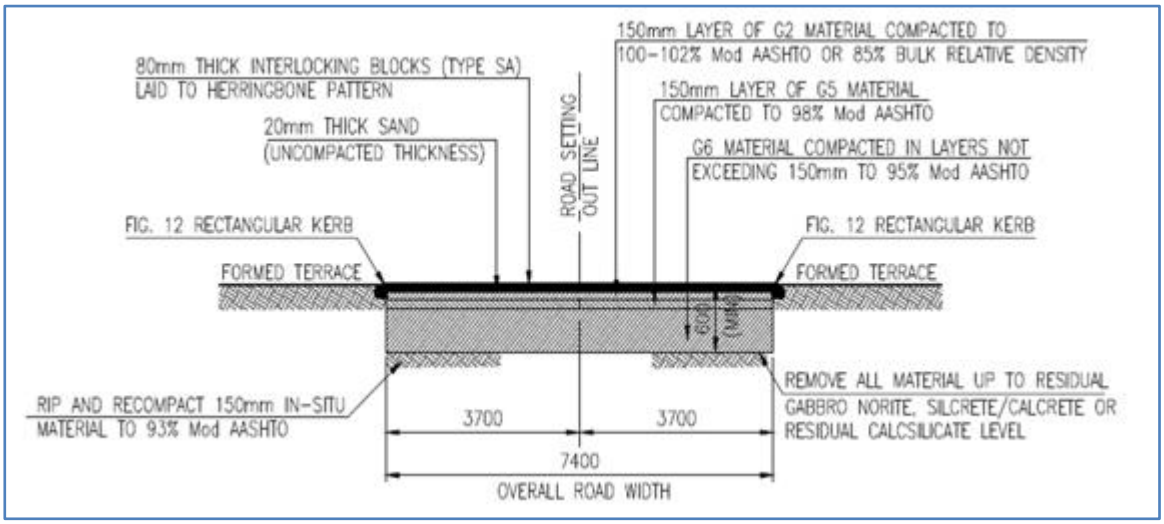


Figure 18.18 Gravel Road Layer Works

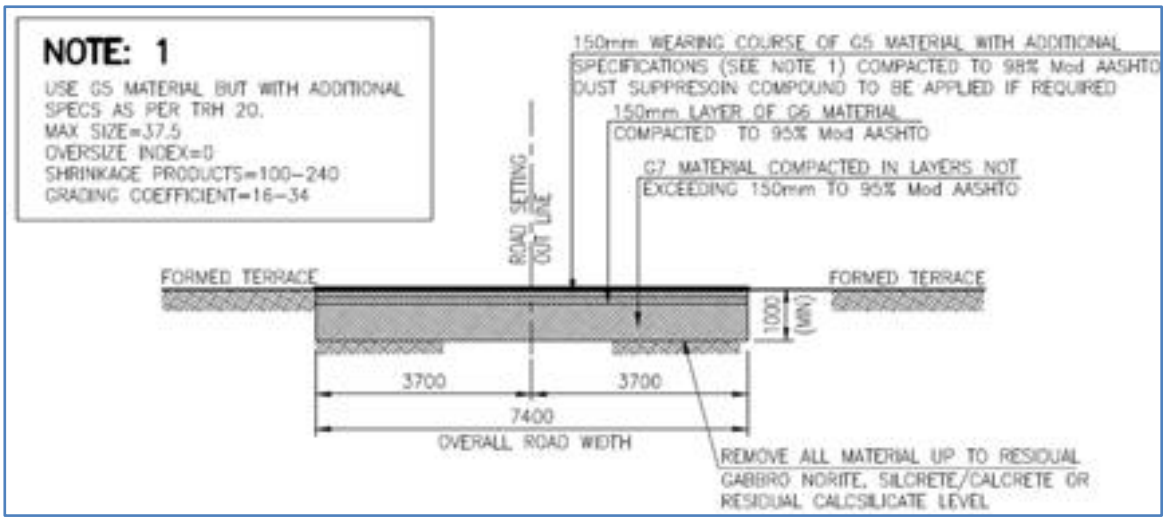


Figure 18.19 Haul Road Layer Works

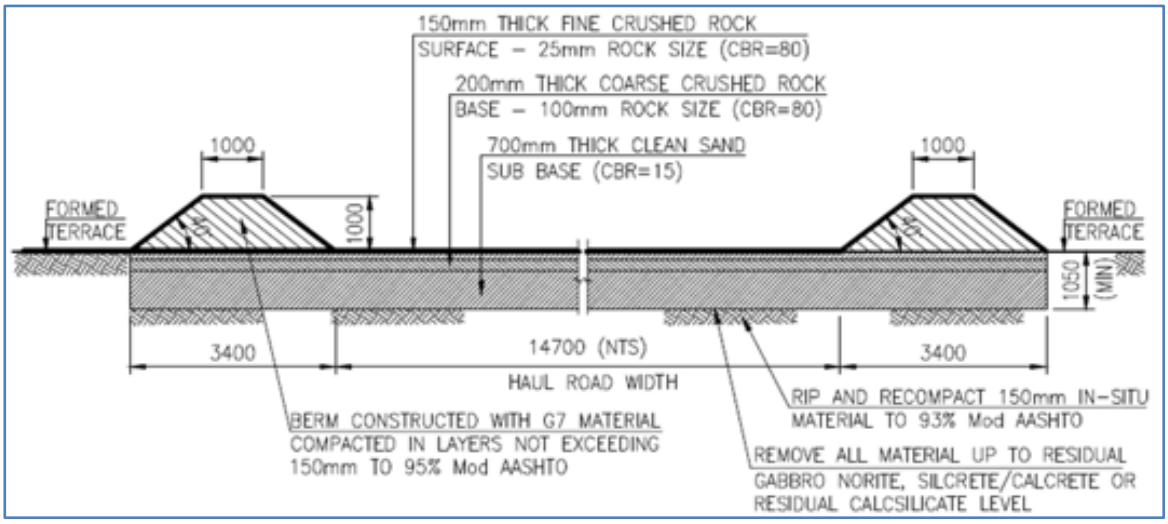
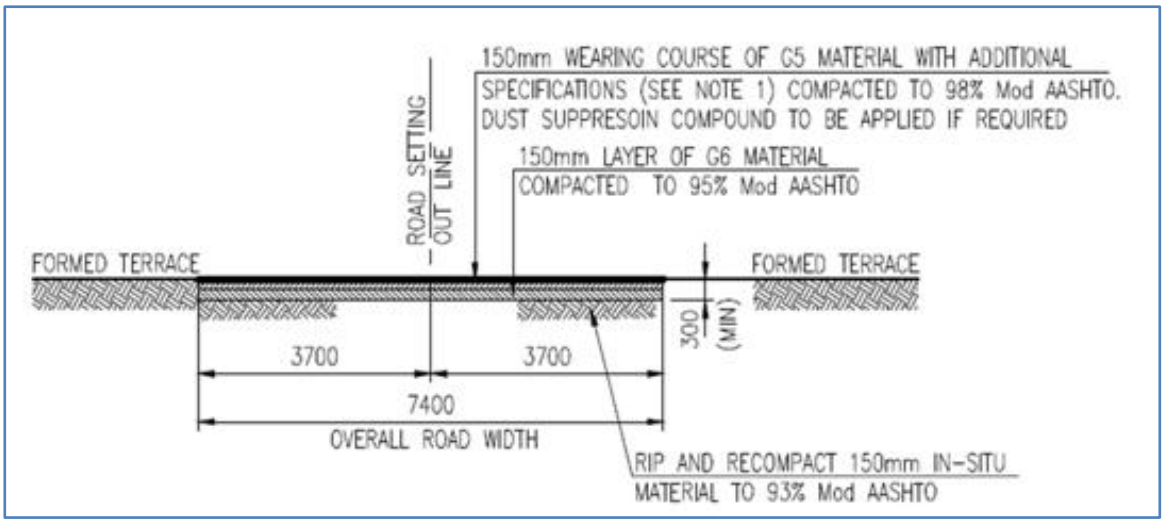


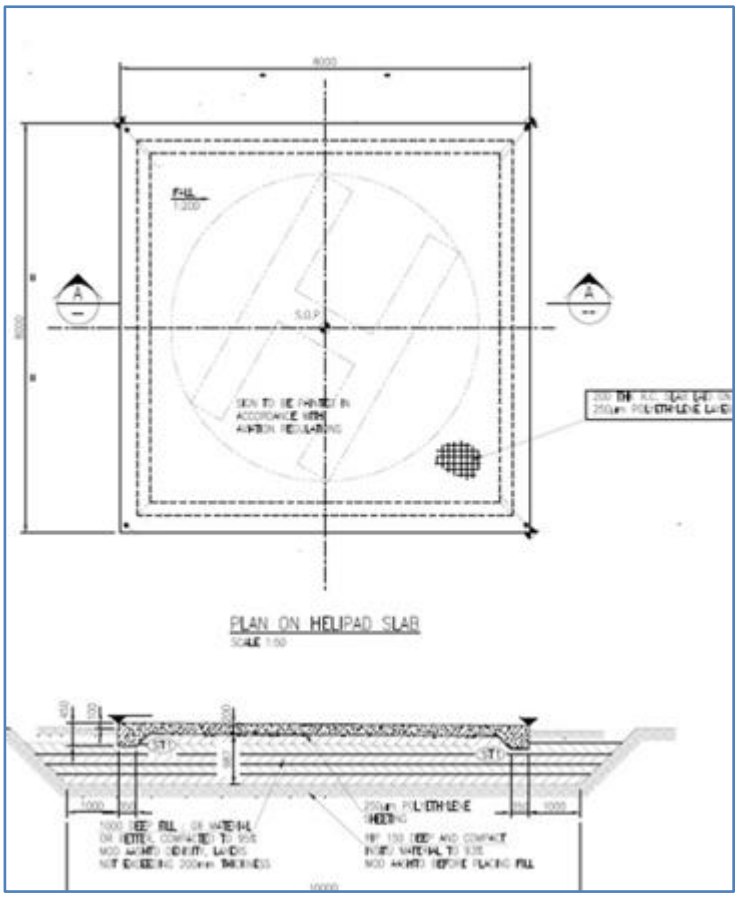
Figure 18.20 Service Road Layer Works



18.8.7 Helipad

There are three helipads designed, of which only one forms part of the Phase 2 study. The remaining two will be constructed in the future. Figure 18 20 below is a typical Helipad design.

Figure 18.21 Typical Helipad Design



The design of the helipad will be further progressed during the execution phase of the project. The minimum criteria for designing the helipad will include ensuring structural integrity to support the maximum take-off weight of the design helicopter, maintaining obstacle-free approach and departure paths, and adhering to regulatory standards and guidelines (as determined by the South African-Civil Aviation Authority) for dimensions, markings, and lighting. Additionally, the design will incorporate effective drainage, fire safety measures, and accessibility for emergency services to ensure operational safety and compliance.

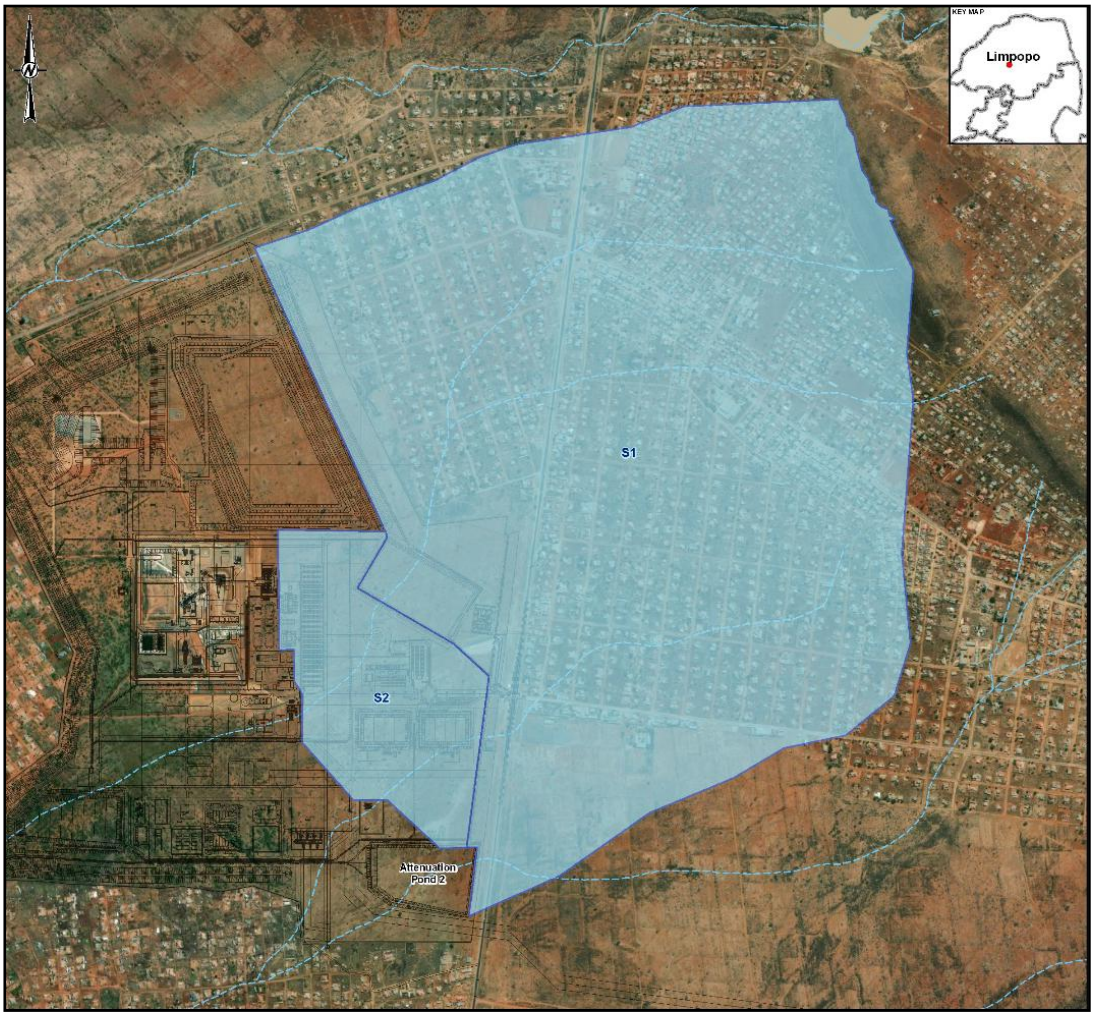
Other key criteria to be assessed include, but are not limited to:

- Ensuring the helipad is located at a sufficient distance from sensitive areas or residential zones to minimise noise and vibration impacts.
- Evaluating prevailing wind directions and turbulence.
- Ensuring compliance with prescribed gradient limits for safe approach and departure paths.
- Identifying and addressing potential obstructions within the flight paths

18.8.8 Stormwater Management

The mine area receives clean stormwater run-off 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 run-off is kept separate from the polluted run-off 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 18.22 below, which depicts the north-eastern clean water catchment areas that affect the mine site.

Figure 18.22 Clean Water Catchment Areas



When the mine infrastructure is constructed, a concentrated flow of rainwater will be created, which needs to be managed via a stormwater management system to negate its possible effect on surrounding communities.

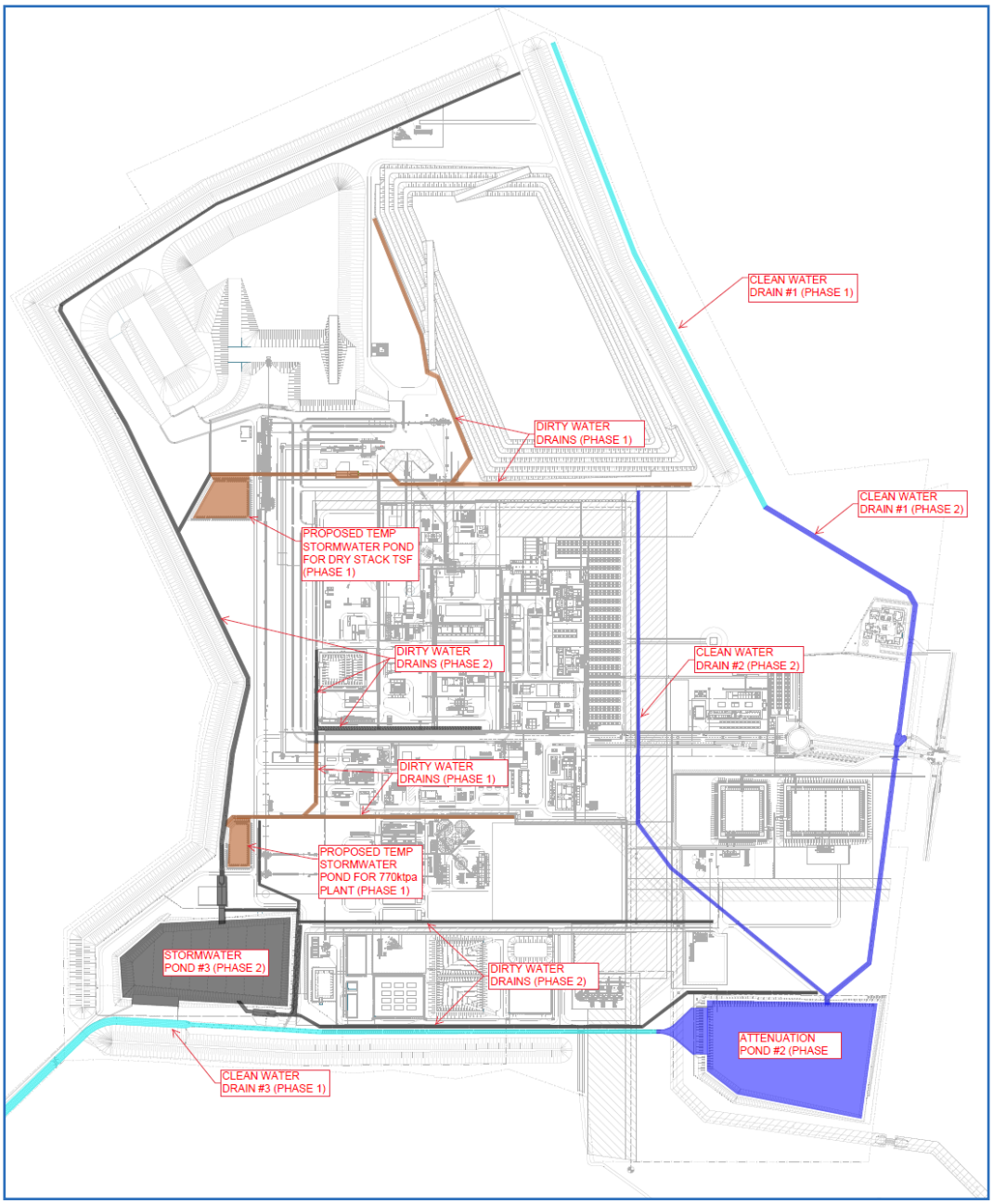
WSP, as the hydrology consultants, have modelled this run-off and made recommendations for the position of cut-off drains and an attenuation dam, based on the following:

- A 1 in 100-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 conditions.
- A 1 in 50-year stormwater event was used to calculate the stormwater runoff 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 depth with a minimum of 0.8 m has been applied for stormwater containment structures. Freeboard with respect to water storage dams can be defined as the vertical distance between the full supply level (level at which the dam overflows) and the lowest point on the dam wall crest level.
- The stormwater management system consists of the following:
- Clean Water Cut-Off Drains #1 and #2.
- Clean Water Discharge Drain #3.
- Dirty Stormwater Runoff Drains.

Refer to Figure 18.23 for the proposed overall primary stormwater drains and ponds layout.

The DWS has approved the water use licence for the abstraction of water and disposal of waste water as detailed in the amendment of licence number 07/A61G/GCJAIBF/ 6975, dated 15 August 2023.

Figure 18.23 Overall Primary Stormwater Drains and Ponds



18.8.8.1 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. There is a risk of erosion and possible damage of the storm water drain during events larger than the 1 in 10-year storm event.

Inspection of the drain following such events should be included in the mine operational procedures.

18.8.8.2 Dirty Stormwater Run-Off Drains

The dirty stormwater runoff drains are a network of containment channels running through the mining area, collecting all contaminated runoff water and directing it towards the dirty stormwater pond #3. These drains vary in size, 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 (areas such as the initial DTSF, stockpiles, shafts, concentrator plants and waste facilities) within the overall mining area. The secondary drains are designed to cater for 1:10-year storm events, and these drains are located within the sub-areas.

During Phase 1, the primary drains around the initial DTSF and the concentrator plant have been constructed. These drains 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 constructed as part of early Phase 2 construction works. The existing stormwater drains and pond will cater for the inner shaft area.

18.8.9 Water Storage Facilities

Different types of dams have been designed in accordance with their varied requirements. The main water storage facilities, lining type and storage capacities are described below for the different areas of the mine.

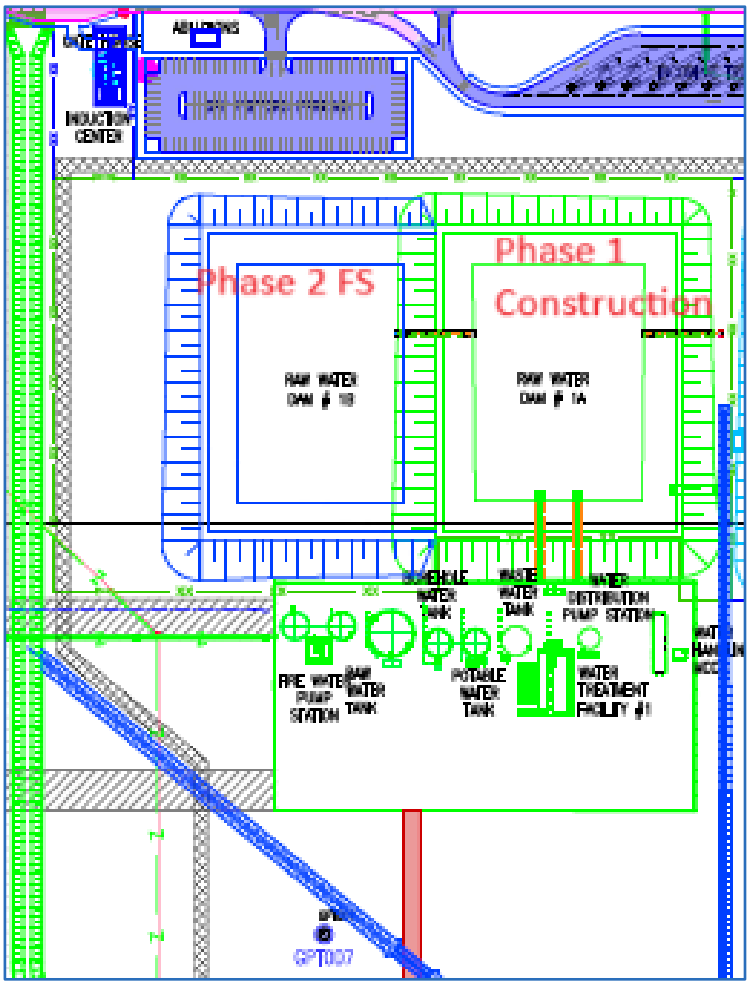
18.8.10 Raw Water Storage Dams

The design provided for two raw water storage dams. These dams, #1A and #1B are 4.5 m deep and high-density polyethylene (HDPE) lined with a network of subsoil drains beneath the liner.

Dam #1A is being constructed under Phase 1 execution, and Dam #1B will be constructed as part of Phase 2. Refer to Figure 18.23 below. The two raw water dams are required to ensure a maximum storage capacity of 49,000 m³ can be attained with the ability to store two commodity types of water supply, namely clean water from boreholes and grey water (treated effluent) from the Masodi WWTW.

With the anticipated average water demand of 9,809 m³ per day required during full production, the combined raw water storage capacity will be able to sustain operations for approximately five days in case of an emergency.

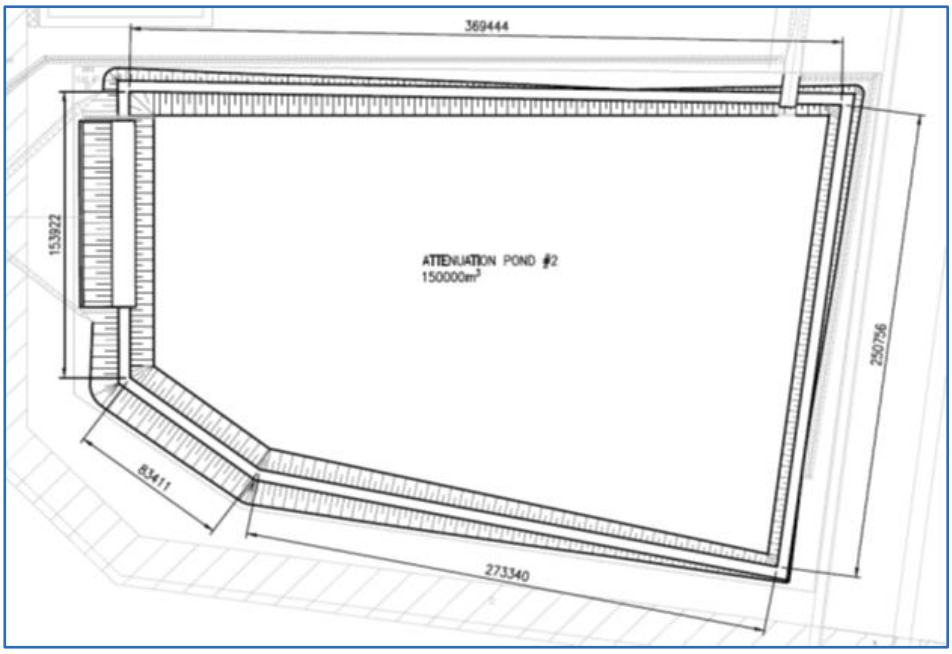
Figure 18.24 Revised Raw Water Dam #1B



18.8.10.1 Clean Stormwater Attenuation Pond

The design includes for an Attenuation Pond (#2), which has been sized by WSP at 150,000 m3 in order to release the collected upstream clean water at a reduced flow rate and velocity into the environment. Refer to Figure 18.25 below.

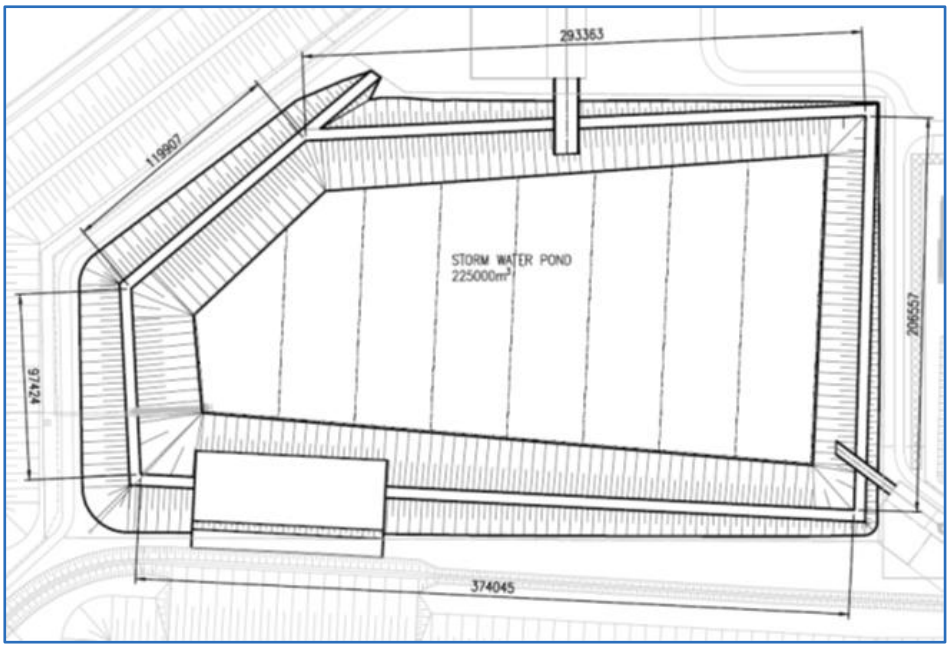
Figure 18.25 Attenuation Pond No. 2



18.8.10.2 Stormwater Pond #3

A single large HDPE-lined stormwater pond #3 (Figure 18 25 below) is located in the south-western corner of the mining area. The dam capacity is 225,000 m³, and water flows into the dam through two silt traps, one on the northern side and one at the south-eastern corner, catering for the dirty stormwater runoff from the mining area. Subsoil drains are installed underneath the HDPE lining. This forms part of the Phase 2 Study.

Figure 18.26 Stormwater Pond #3



18.8.11 Services

18.8.11.1 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 bulk water supply to the mine is via treated sewage effluent from the Masodi 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 has been constructed at the mine site.

The WWTW supplied treated sewage effluent water will be pumped into a water filtration feed tank, the water will then be disinfected and filtered in order to have a turbidity not 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 as described above in Section 18.8.9.1.

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 is abstracted from four 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 m³/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 222 m³/day.

18.8.11.2 Assay Laboratory

Phase 2

A centralised mine site laboratory will be constructed 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.

All samples from geology; mining; and concentrator will be submitted to the centralised mine site laboratory for sample preparation and assay analysis.

Infrastructure samples comprise 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, groundwater, 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.

18.8.11.3 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 IT sleeves between Phase 1 infrastructure and six electrical sleeves at various road crossings.
- Masodi WWTW treated effluent supply line for bulk water supply to the mine area.
- Borehole water supply.

18.8.11.4 Sewer Reticulation

The domestic sewage requirements for the mine area are designed to enable collection from various facility points and routed through an unplasticized polyvinyl chloride (uPVC) gravity-fed buried pipe system of 100 to 250 nominal bore (NB), depending on the flow, to a concrete-lined sump.

The effluent collected in the sump is pumped to a sewage treatment plant located in the waste management area. Manholes have been allowed for every 50 m in order to clean and maintain the pipelines. The plant will be situated within the integrated waste management area.

The treatment plant incorporates a biological treatment process with a septic tank performing primary treatment by settling solids and breaking down organic matter, while bioreactors use microorganisms to further break down pollutants, and clarifier separates solids from the treated water before disinfection and reuse.

Limited sewage sludge is generated from the treatment plant. Any biological sludge that is washed through the bioreactors is settled out in the clarifier and is returned to the first septic tank via an automated process with a submersible pump controlled by a timer switch. The volume of the bioreactors is turned over every hour without any requirements for additional desludging.

The septic tank will be de-sludged every 3–5 years with the frequency determined by monitoring the sludge levels. The septic tank can be emptied into a sludge-drying bag on a sludge-drying bed (3m x 3m). The liquid will be collected and pumped back into the plant for treatment. The dried sludge can be buried in a landfill site or taken to the municipal water treatment plant

Treated sewage effluent will be discharged to the Stormwater Pond #3 and ultimately recirculated for re-use. The treated sewage effluent water will comply to discharge quality standards allowing for discharge into the south-western clean water discharge drain #3 in the event of excess water accumulation in the dirty Stormwater Pond #3.

The Phase 2 sewer treatment plant is based on a modular design concept to cater for the phased development plan (PDP) approach, which is identical to the plant constructed in Phase 1 and it will be located adjacent to the Phase 1 constructed plant.

18.8.12 Weighbridges

Two 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.

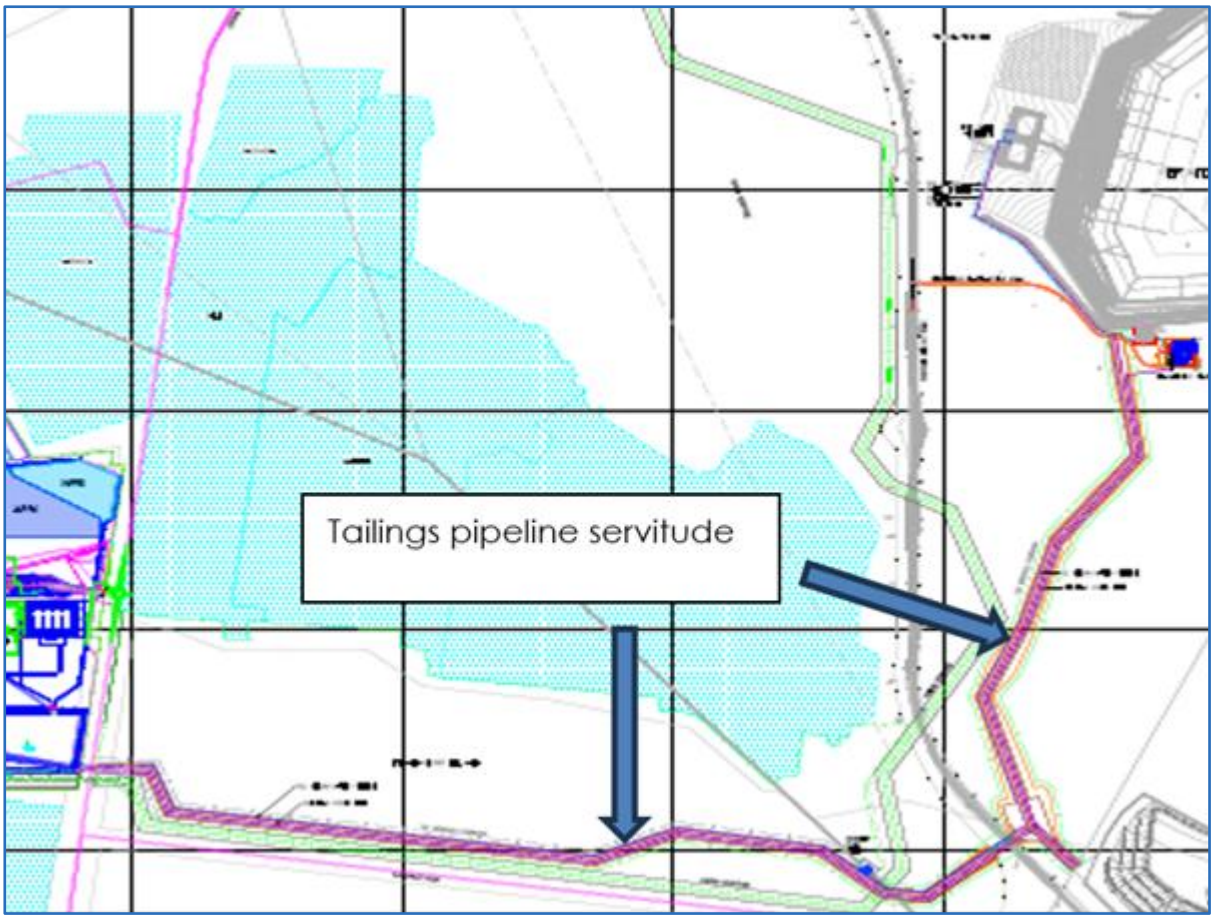
18.8.13 TSF Pipeline Servitude

A TSF pipeline servitude has been developed in order to pump tailings to the full production TSF at Rietfontein. This servitude will also cater for return water from the TSF for reuse by the concentrator plants. This servitude forms part of Phase 2.

18.8.13.1 Earthworks

The tailings pipeline servitude is split into 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, up to the TSF approximately 7.5 km north-east of the mine area. Refer to Figure 18.27 indicating the layout of the tailings pipeline servitude.

Figure 18.27 Tailings Pipeline 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 maintenance crossing access over the tailings pipeline servitude

18.8.13.2 Overland Piping

Tailings from the concentrator plant are pumped to a booster station situated approximately 5 km from the concentrator plant. The total length of the tailings servitude is approximately 7.5 km between the concentrator plant and the TSF.

- The tailings return water is pumped from the TSF return water dam back to the concentrator plants via various pipelines implemented across different phases of the project development.

- The various overland pipelines include:
- 300D Steel pipe, 10 mm HDPE lined. Common tailings stand by line.
- 300D Steel pipe, 10 mm HDPE lined Phase 2 tailings disposal pipeline to dry stack filtration plant.
- 300D Steel pipe, 10 mm HDPE lined Phase 3 tailings disposal pipeline to dry stack filtration plant.
- 300D Steel pipe, 10 mm HDPE lined Phase 4 tailings disposal pipeline to dry stack filtration plant.
- 315D HDPE PN 20 Common standby line return water to the process plant.
- 315D HDPE PN 20 Phase 2 return water to the process plant.
- 315D HDPE PN 20 Phase 3 return water to the process plant.
- 315D HDPE PN 20 Phase 4 return water to the process plant.
- 150D steel pipe, clean water standby line.
- 150D Phase 2 clean water supply to dry stack tailing filtration plant.
- 150D Phase 3 clean water supply to dry stack tailing filtration plant.

These pipelines are routed in the tailings servitude, the TSF corridor servitude and lie on the cleared ground (welded pipes) or on precast concrete sleepers at 6m intervals if welded.

18.9 Buildings

The buildings previously designed and drawn as part of the 2022 Feasibility Studies have been optimised where possible and are categorised into the following building types:

- Brick and mortar buildings.
- Structural steel and cladding.
- Prefabricated electrical buildings.

The previously proposed prefabricated buildings have been changed to either brick and mortar or structural steel buildings IBR sheeted cladding during the current Phase 2 FS.

The electrical buildings are the only buildings that remain prefabricated.

18.9.1 Brick and Mortar 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.

The buildings have been divided into two groups: large buildings (Table 18.4) and Small Buildings (Table 18.5) indicated below.

Table 18.4 Large Buildings

WBS Number	Building Description
Area 2000	
257201	Change House
Area 2000	
257201	Plant Office

Table 18.5 Small Buildings

WBS Number	Building Description
257202	Control Room and Auditorium
257204	Medical Room and Mine Rescue Room
257206	Banksman Cabin
257805	Shaft Area Gate House
318502	Ablution
341104	Canteen
341105	Plant Gate House
618116	Ablutions Store Area
618111	Mine Entrance Ablution
618129	Gate House and Induction Centre
618105	Main Entrance Vehicle Gate House

18.9.2 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.

The main store has been optimised to include the following:

- Original equipment manufacturer returns
- Capitol store
- Sub-component store

- Main store
- Ropes and accessories

Table 18.6 below lists all the workshops and stores per area as well as the proposed execution phase.

Table 18.6 Workshop and Stores Buildings

WBS Number	Phase	Building Description	Building Type	Crane
257502	2	Heavy Vehicle Workshop	Steel portal frame with brick fill; dwarf wall and offices including plumbing and electrical fixtures.	2 x 5 tonne
257501	2	Engineering Workshop	Steel portal frame with brick fill; dwarf wall and offices including plumbing and electrical fixtures.	2 x 5 tonne
328504	2	Plant Stores	This is an extension to the Phase 1 plant store.	None
328505	2	Plant Workshop	This is an extension to the Phase 1 plant workshop.	10 tonne
347105	2	Crusher Workshop	Steel portal frame with brick fill; dwarf wall and offices including plumbing and electrical fixtures.	6 tonne
618109	2	Main Store	Steel portal frame with brick fill; dwarf wall and offices including plumbing and electrical fixtures.	None
618118	2	Tyre Store		
618119	2	Cable Yard	Steel portal frame.	None

18.9.3 Change House

The change house building is made up of two buildings, with the first one being part of the Phase 2 FS study and the second one will be constructed in the future. The Phase 2 change house building is a double-storey structural steel and brick in-fill building with civil bases, plinths, and surface bed with all of the necessary sanitary facilities, seating benches, light fittings, extractors, lockers, shelves, and geyser/boilers hot water systems.

The building has been designed to encompass other facilities such as the lamp room, boot wash, laundry room, storerooms, 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 of the employees.

18.9.4 Electrical Buildings

The substation buildings are structural steel buildings with IBR roof sheeting, elevated concrete slabs and filled-in brick work. Table 18.7 lists all substation buildings and their type.

Table 18.7 Substation Buildings

WBS Number	Phase	Building Description	Building Type
252307	2	Minisub No. 1 - Vehicle Workshop	Structural Steel with Filled-In Brick Work
252311	2	Minisub No. 5 - Change House Complex	Structural Steel with Filled-In Brick Work
252320	2	Medium Voltage (MV) Substation – Shaft 6	Structural Steel with Filled-In Brick Work
252321	2	MV Substation - Refrigeration Plant (Phase 2 - 7 MW)	Structural Steel with Filled-In Brick Work
252322	2	MV Substation - Refrigeration Plant (Phase 3 - 14 MW)	Structural Steel with Filled-In Brick Work
252323	2	MV Substation – Shaft 7	Structural Steel with Filled-In Brick Work
252324		MV Substation – Shaft 3 Phase 2 Rock handling	
318301	2	Crushing and Screening Substation	Structural Steel with Filled-In Brick Work
318302	2	Crushing and Screening Motor Control Centre (MCC)	Structural Steel with Filled-In Brick Work
318303	2	Mill Variable Speed Drive Room	Structural Steel with Filled-In Brick Work
318304	2	Milling MCC	Structural Steel with Filled-In Brick Work
318305	2	Flotation MCC	Structural Steel with Filled-In Brick Work
318306	2	Concentrate and Reagent MCC	Structural Steel with Filled-In Brick Work
318307	2	Tailings Disposal MCC	Structural Steel with Filled-In Brick Work
318308	2	Tailings Booster MCC	Structural Steel with Filled-In Brick Work
318309	2	Tailings Filtration MCC	Structural Steel with Filled-In Brick Work
318310	2	Minisub - Plant Change House and Offices	Structural Steel with Filled-In Brick Work
318311	2	Minisub - Crusher Workshop	Structural Steel with Filled-In Brick Work
318312	2	Minisub - Plant Lighting No. 1	Structural Steel with Filled-In Brick Work
318313	2	Minisub - Plant Lighting No. 2	Structural Steel with Filled-In Brick Work
318314	2	Minisub - TSF Lighting	Prefabricated Building
618203	2	Plant MV Substation	Structural Steel with Filled-In Brick Work
618207	2	Generator Substation MV/Low Voltage (LV)	Structural Steel with Filled-In Brick Work
618211	2	Minisub No. 1 - Main Office	Structural Steel with Filled-In Brick Work
618212	2	Minisub No. 2 - Main Store Area	Structural Steel with Filled-In Brick Work
618213	2	Minisub No. 3 - Main Entrance Area	Structural Steel with Filled-In Brick Work

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.

Where practical, firewater reticulation pipework will be buried, comprising of HDPE Class 16 pipework. Above-ground reticulation pipework will be galvanised steel piping. All fittings and flanges will be Class 16. All isolation/section valves will be underwriters laboratories (UL) listed and factory mutual (FM) approved.

Fire hydrants will be fed off the fire water ring main and will be spaced at a maximum of 90 m intervals.

All fire hydrant, hose reel and protection system risers will be steel and protected accordingly against corrosion.

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 fire water ring main.

18.10.1 Fire Detection

All substations and MCCs are equipped with highly sensitive LASER-based smoke detectors with aspirators connected to networks of sampling pipes, providing a very early smoke detection system.

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 a 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 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 .

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 18.8 is a list of systems and typical locations (to be read in conjunction with the fire protection engineering design criteria (EDC)).

Table 18.8 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.
- Design and supply of electrical, control and instrumentation equipment are in accordance with the Platreef and ABB preferred supplier agreement.

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:
  - Mining Equipment fleet voltage: 1000 V AC resistively earthed
  - Underground Infrastructure: 525 V AC resistively earthed
  - Surface LV Distribution: 525 V AC resistively earthed
  - 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. The power factor correction (PFC) design is as per the PFC and harmonic filter system (HFS) Study, document number 1051-ABB-3282-65-Z-0014 conducted by ABB. The PFC being installed as part of the Phase 2 execution project is adequate, and no additional PFC is required for the 4 Mtpa Phase 2 FS.

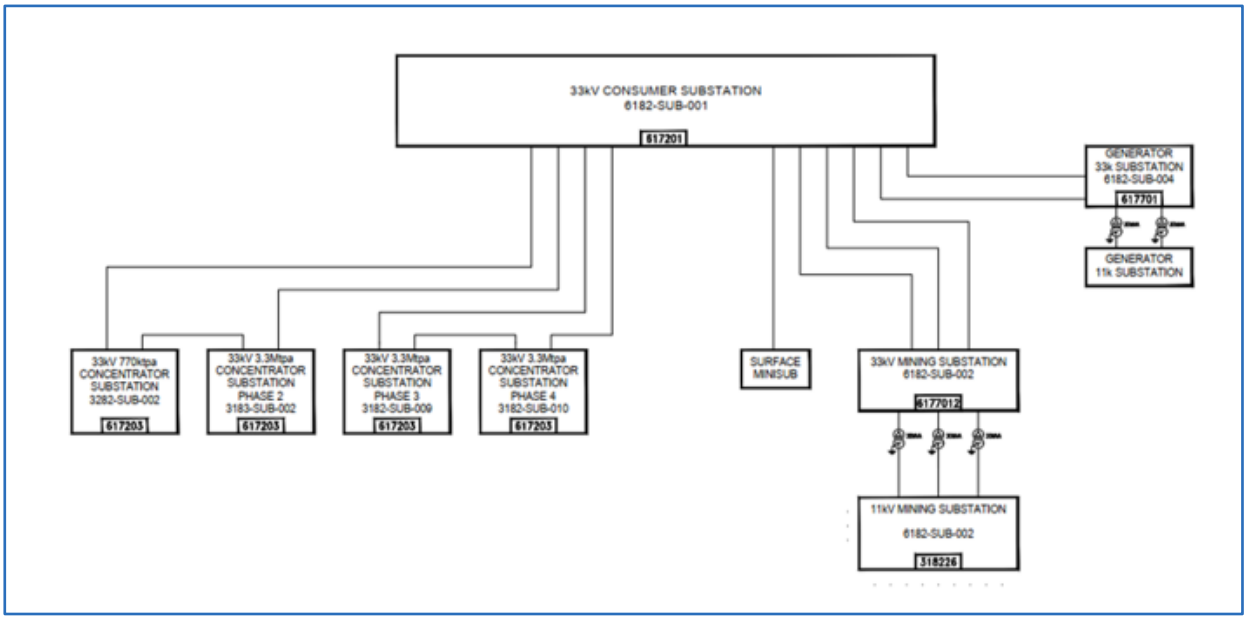
18.11.4 MV Distribution

The MV distribution from the 40 MVA 110 kV/33 kV Eskom transformers and outdoor yard are included within in the general surface infrastructure.

The installed capacity in the Eskom 132/33 kV outdoor substation is four no. 40 MVA 132/33 kV transformers. The 33kV Main Intake substation is only equipped with incomer circuit breakers from two of the Eskom 40MVA transformers with space provision for the future circuit breakers. The additional 33 kV circuit breakers and cable installations are allowed for as part of this study. The design capacity is 120 MVA, with the fourth 40 MVA transfer for redundancy.

Power is distributed from the 33kV Main Intake substation to the 33kV Concentrator substations and the 33kV Mining substation via 33kV cables on racking. The 33 kV Main Intake substation is equipped with a double bus 33 kV indoor switchgear, for maximum load distribution flexibility. The 11kV Mining substation is supplied via three 30 MVA 33/11 kV transformers from the 33 kV Mining substation. Power is distributed to underground substations from the 33kV Mining substation. The electrical design allows for a redundant supply to all substations. Figure 18.28 indicates the planned MV distribution.

Figure 18.28 MV Distribution



In the consumer substation double bus 33 kV indoor switchgear is provided, for maximum load distribution flexibility. From there power is distributed on a 33 kV network to the 33 kV mining, plant, and other substations as illustrated diagrammatically in Figure 18.28. Single bus switchgear is utilised in all other substations.

18.11.5 LV Distribution

Power at 525 V has been derived from a suitable number of 630 kVA, 2,000 kVA, 33 kV/550 V Dyn11 ONAN step-down transformers. These will be connected with the neutral point resistively earthed. These transformers are utilised to power the MCCs.

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).

The Phase 1 control system comprises of an ABB x A800 distributed control system (DCS) and distributed field and MCC networks located at the existing Phase 1 concentrator.

For the Phase 2 FS, allowance has been included for a new DCS to be installed in the new auditorium and control room facility. All new control system equipment in the A6000 areas will be connected to this new control system.

One centralised DCS controller is envisaged to cover the new equipment on surface. This controller and the existing controller will be used to control all surface equipment.

MCC, MV and field inputs/outputs (I/O) equipment fibre networks will be installed on the trunk and secondary routes to cover the surface areas which need to be connected to the new control system.

The design and implementation of the Control System Software will be based on the existing implemented standards and libraries and using the same software versions. No major changes or updates are foreseen to the current DCS software libraries.

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.

18.11.9 Centralised Control Centre

It is recognised that centralised control rooms are becoming an integral feature of the modern mine. This is geared towards improved operational efficiencies with streamlined processes as a key driver in reducing operational costs.

Streamlined operations and planning are best achieved when its control room operators are able to comprehend the entire value chain of the operation and being able to make decisions that improve overall production efficiency.

A centralised control room is a vital ingredient to achieving this comprehension of the mine value chain, from extraction through to shipping. The control room video wall should emulate this view with a video wall that visually represents this value chain. For Platreef, this means housing both the mine and the plant control in one room. The video wall should display key performance indicators for operational performance, and operators, in turn, should be incentivised to meet key performance indicator (KPI) targets. KPI indices and targets should be designed in such a manner that meeting them results in a meeting of organisational goals set at executive level.

This visualisation should include the following:

- Mining fleet machine monitoring.
- Planning and scheduling.
- Supervisory, control and data acquisition (SCADA) for underground, surface and plant process automation, which includes product handling.
- Essential services – fire protection, ventilation-on-demand, water handling.
- Personnel and vehicle tracking.
- Monitoring systems – environmental, geotechnical and 3D scanning.

With co-location of operators from different sections of the mine in the same room, the video wall becomes a source of one version of the truth and information gaps are closed significantly

18.11.9.1 Mining Operations Management Systems

The mining operations management systems (MOMS), also referred to as manufacturing execution systems (MES), required for operational planning and maintenance should be scoped in early stages to ensure that the system architecture caters for MOMS needs. Functions to be explored for these systems include:

- AI and data analytics
- Digital twinning - 3D underground mapping
- Production planning and scheduling
- Production performance
- Production monitoring
- Cost management
- Quality control
- Grade control and stockpile management
- Mass balancing/materials tracking/reagents tracking
- Maintenance and downtime analysis
- Procurement and supply chain
- Asset management
- KPIs and reporting
- Document control
- Short interval controls
- Workflow

Platreef IT/OT teams are instrumental in helping define preferences in how these systems should be deployed to integrate with existing systems.

18.11.9.2 Modern Mining Control and Monitoring Systems

The systems listed below are deemed a normal part of a modern mine. The data from and interfaces to these systems will be centralised in the control centre:

- Mining fleet communication
- Process automation - programmable logic controllers and remote I/O
- Collision avoidance system (logging)
- Personnel and vehicle tracking
- Ventilation-on-demand
- Access control
- Environmental monitoring
- Geotechnical monitoring

- Real-time 3D surveying
- MES/MOMS systems
- Mobile devices – tablets

18.11.9.3 Control Centre

A new auditorium and control room facility has been allowed in the capital estimate in the form of a centralised control centre.

Based on a user requirement specification (URS) derived from stakeholder consultation on the required services and functions needing to be centralised in the control central, a building layout for the identified systems, services and functions has been developed. Refer to Figure 18.29.

Intended to be positioned in a central location, the building will house the control centre, where operators will be able to monitor and control the whole complex from a central location.

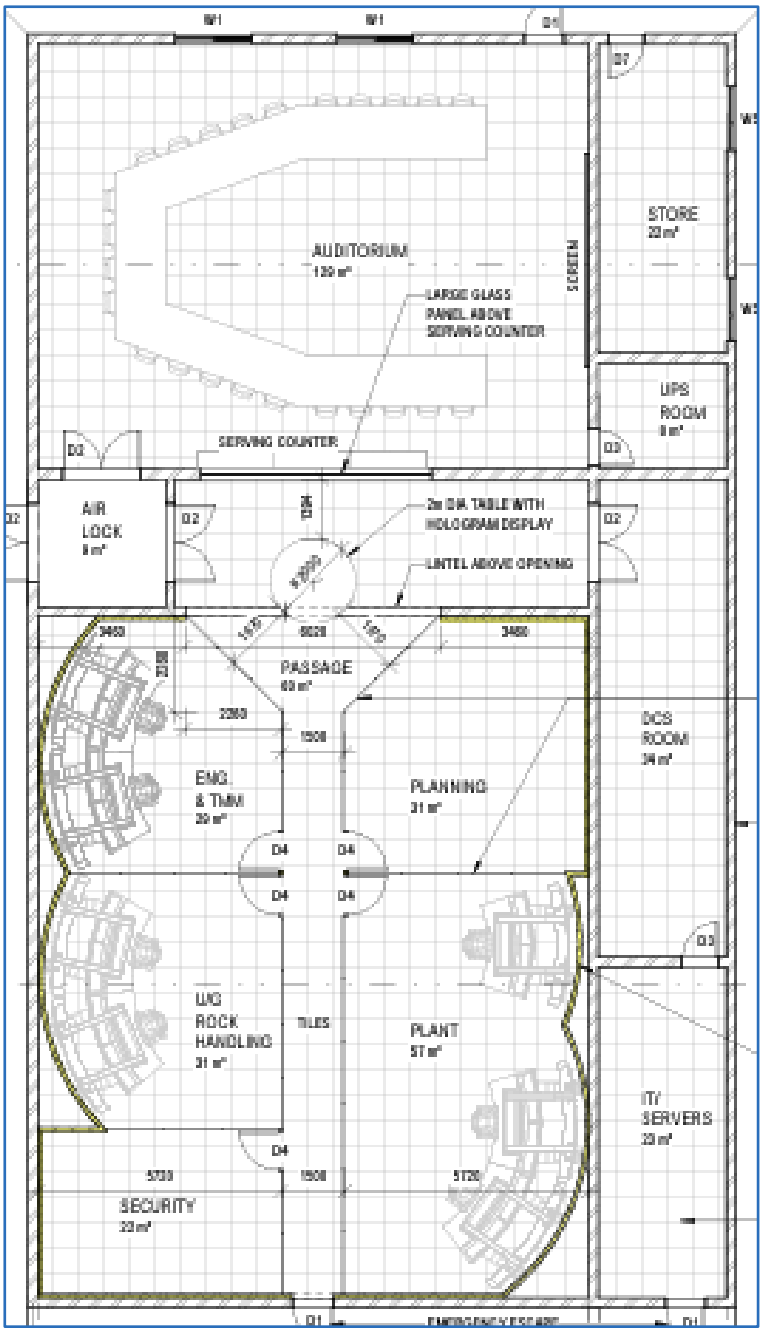
The control centre includes the following areas:

- Auditorium
  - Used for meetings and functions related to the control centre operation, and then general use. It will be fitted with audio and visual equipment and tools to enhance the user experience.
- Plant and underground control rooms
  - Plant and UG control rooms will be fitted with control desks, chairs and video walls to enable operator plant monitoring and control. These two control rooms will be connected via networking equipment to the relevant control systems located in the local DCS and IT server rooms.
- Mining fleet control room
- Planning room
  - Planning room will be used as a general workspace to plan and discuss operations and be close to the control area for relaying pertinent information to operators.
- Security room
  - Security room will be used as a central monitoring space where all operation's security closed-circuit television (CCTV) and communications will be relayed to.
- IT server room
  - The IT server room will house all IT-related equipment and services. Connections will be made from this server room to the main IT server room to establish external IT communication. Fibre and other cable routes will be terminated and routed to and from network cable panels.
- DCS equipment room

- DCS equipment room will house all the control system-related equipment, including DCS controller panels, DCS network equipment panels, network cable panels and videowall equipment panels. Fibre cable will be routed to and from the network cable panels to all the plant, surface and mining areas where control system equipment is located.

- Uninterruptible power supply (UPS) room
  - UPS equipment will be installed in the UPS room to feed power to all the control systems, IT and other equipment in the building. Power to the UPS equipment will be fed from external minisubs or MCCs.

Figure 18.29 Control Centre Layout



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 3
- 4 x 2,500 tonne ROM or waste rock temporary stockpiles
- 1 x temporary 2,500 tonne stockpile
- Permanent waste rock stockpiles
- Rock handling mobile fleet
- 5,200 tonne ROM silos

18.12.1.1 Process Description

The mine will be serviced by three shafts, namely Shaft 1, Shaft 2 and Shaft 3. During steady state operation, Shaft 1 will hoist waste and low grade material and Shaft 3 will be used to hoist ore. Men and material will be hoisted via Shaft 2.

The Shaft 1 winding system initially handles both rock (ROM and waste rock streams) and personnel / materials. The hoisting capacity remains constrained due to this dual purpose. It is noted, however, that the Platreef 4Mtpa Phase 2 FS will dedicate men and material to Shaft 2 allowing shaft 1 to revert to waste and low grade ore hoisting as described above.

ROM and waste will be hoisted to surface through Shaft 1 in batches at 210 tph. 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 (2551-CV-001) via vibrating feeders at a design rate of 300 tph.

The Shaft 1 loadout conveyor (designed at 450 tph but upgraded to 900 tph) 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 (this does not form part of the current FS).

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 3 rock handling system handles both ROM and waste rock streams. Rock is discharged into a bin via one of two openings. A pan feeder below each discharge feeds a conveyor system feeding onto a short conveyor (shaft 3 to shaft 1 loadout conveyor) that directs the feed onto the existing Shaft 1 surface conveyor (2551-CV-001) as described above; alternatively, material can be fed onto a new Shaft 3 conveyor (XXXX-CV-YYY).

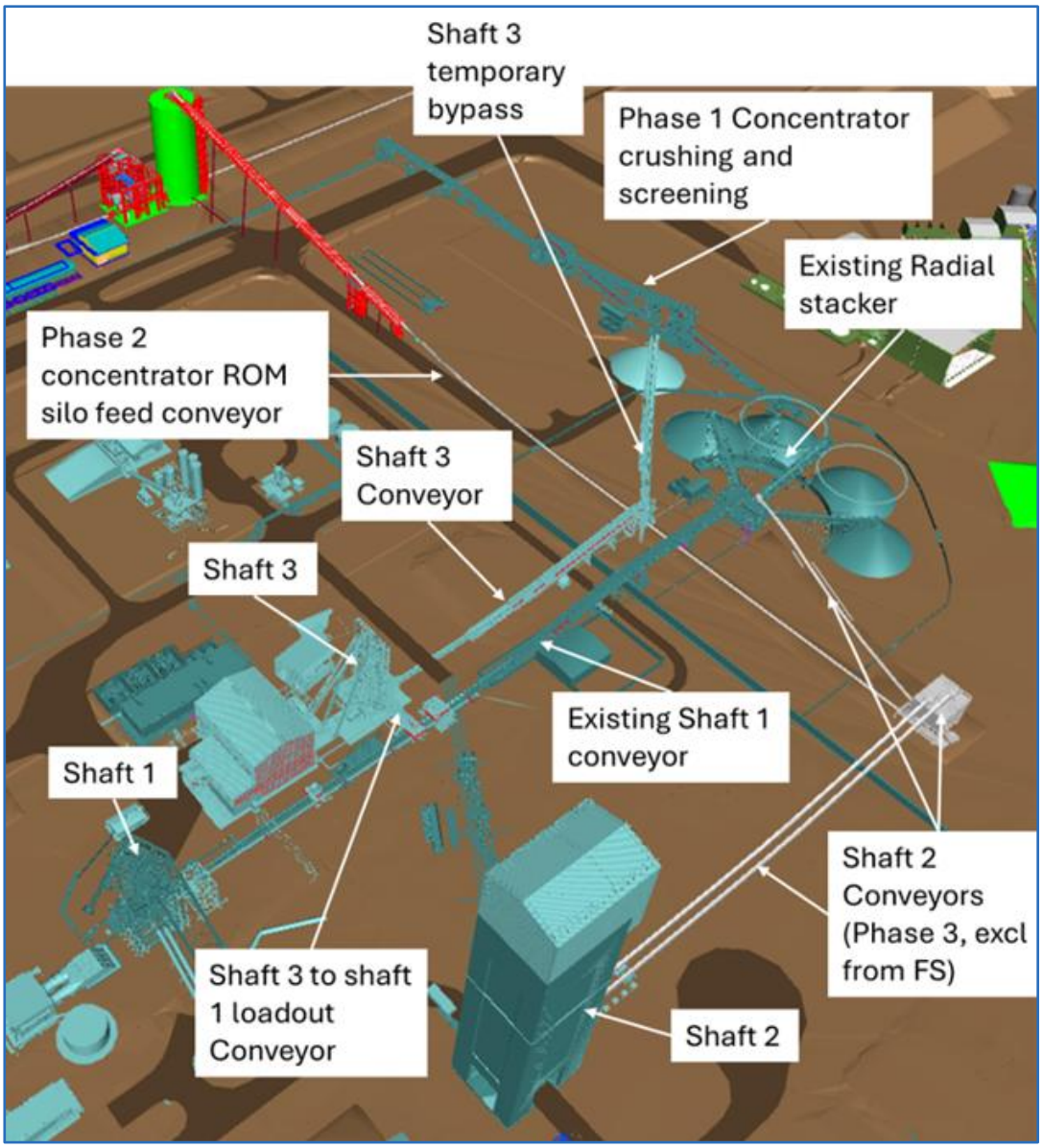
This conveyor feeds a splitter box, allowing ROM feed either to the ROM silo feed conveyor or ROM / waste to the shaft 3 temporary bypass conveyor and its stockpile.

The surface system is designed to be able to separate ore and waste coming from Shaft 1 and/or Shaft 3 at any given time.

The ROM Silo provides storage and buffering capacity, and is 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 18.30 indicates the complete surface rock handling layout for both phases.

Figure 18.30 Surface Rock Handling



DRA, 2024

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 3.3 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 to750 m, 850 m Level and 950 m Level 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.

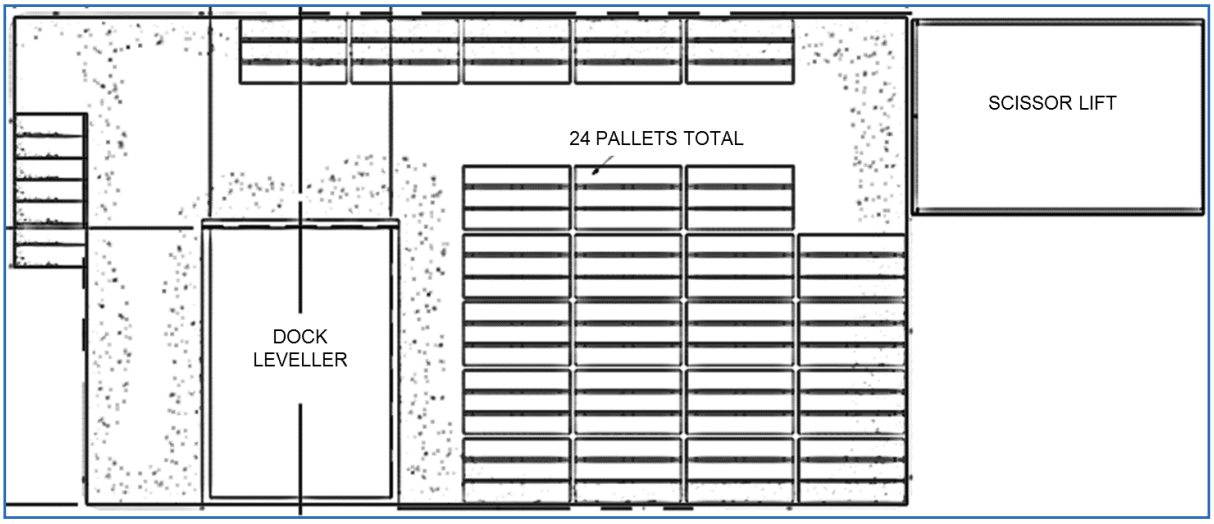
18.12.2.2 Explosives

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.

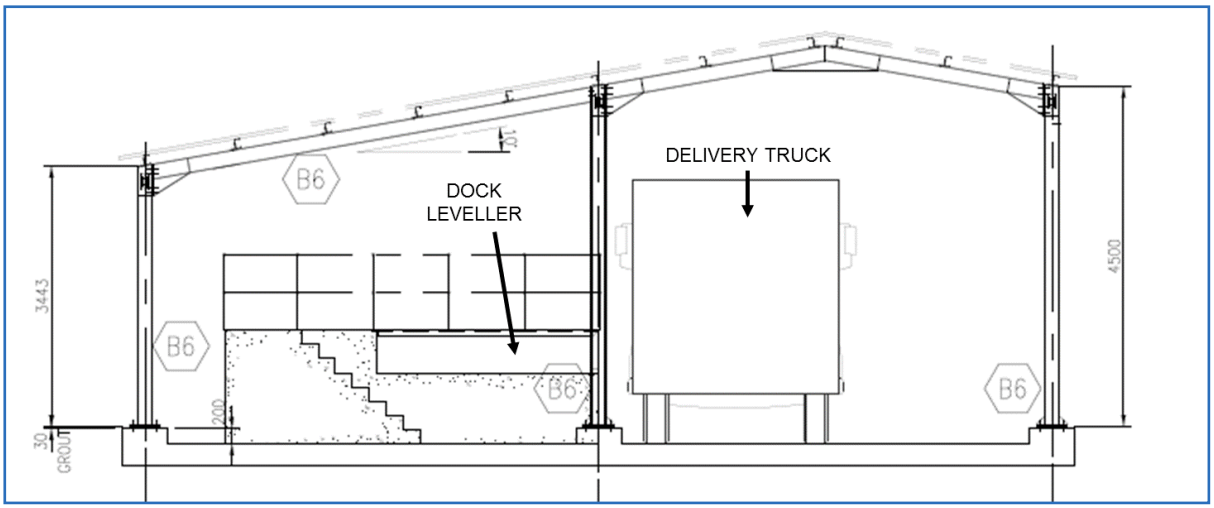
Figure 18.31 to Figure 18.33 show the layout of the offloading area and indicate the dock leveller, the pallet stacking and the scissor lift.

Figure 18.31 Explosives Offloading Facility – Plan View



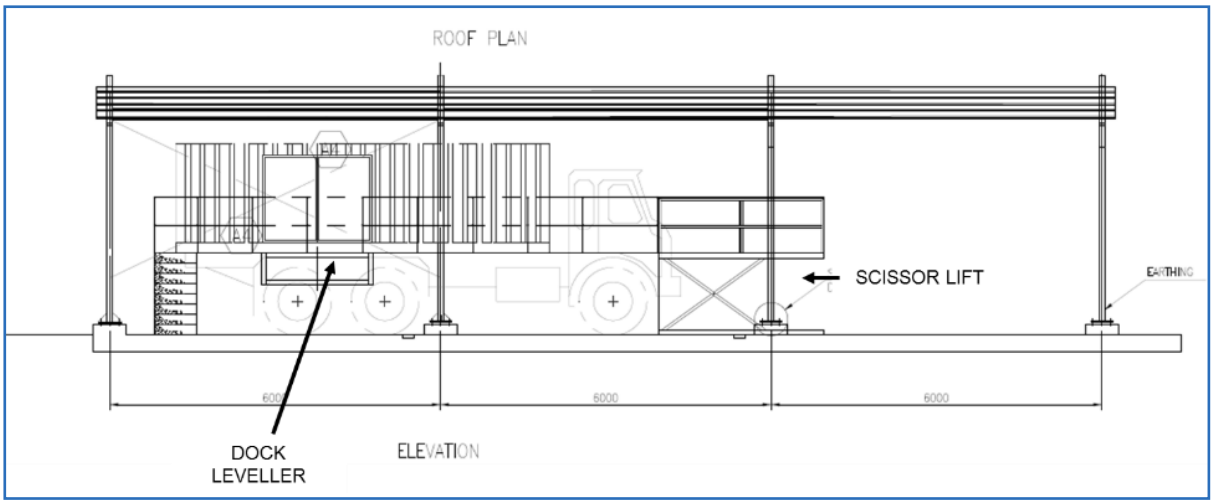
DRA, 2014

Figure 18.32 Explosives Offloading Facility – Front View



DRA, 2014

Figure 18.33 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. Fuel and lubricants will be piped to the three main levels to refuelling stations positioned near the level main workshops.

Figure 18.34 illustrates the plan layout of the surface refuelling and storage depot and indicates the lubricants offloading and dispensing. Figure 18.35 is the same layout, indicating the diesel receiving and dispensing.

Figure 18.34 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Lubes Receiving and Dispensing

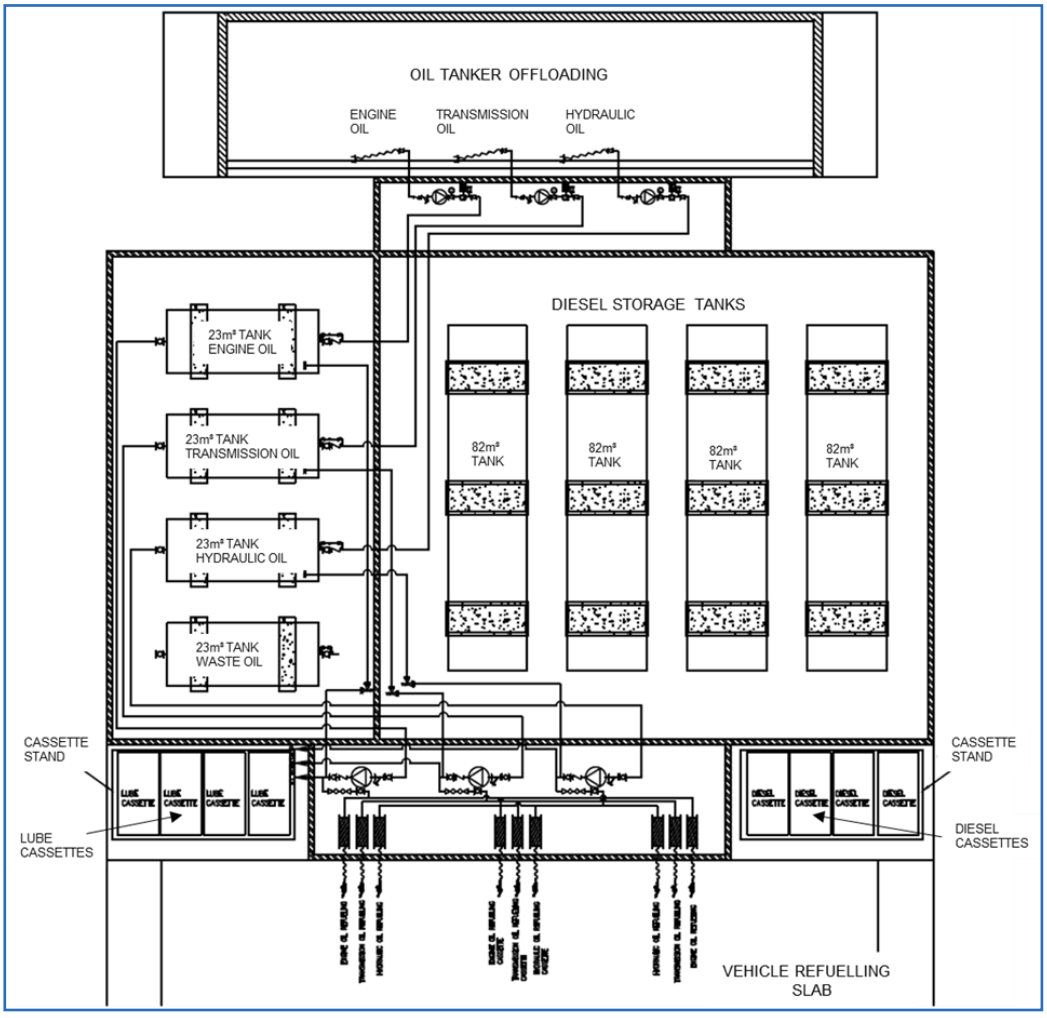

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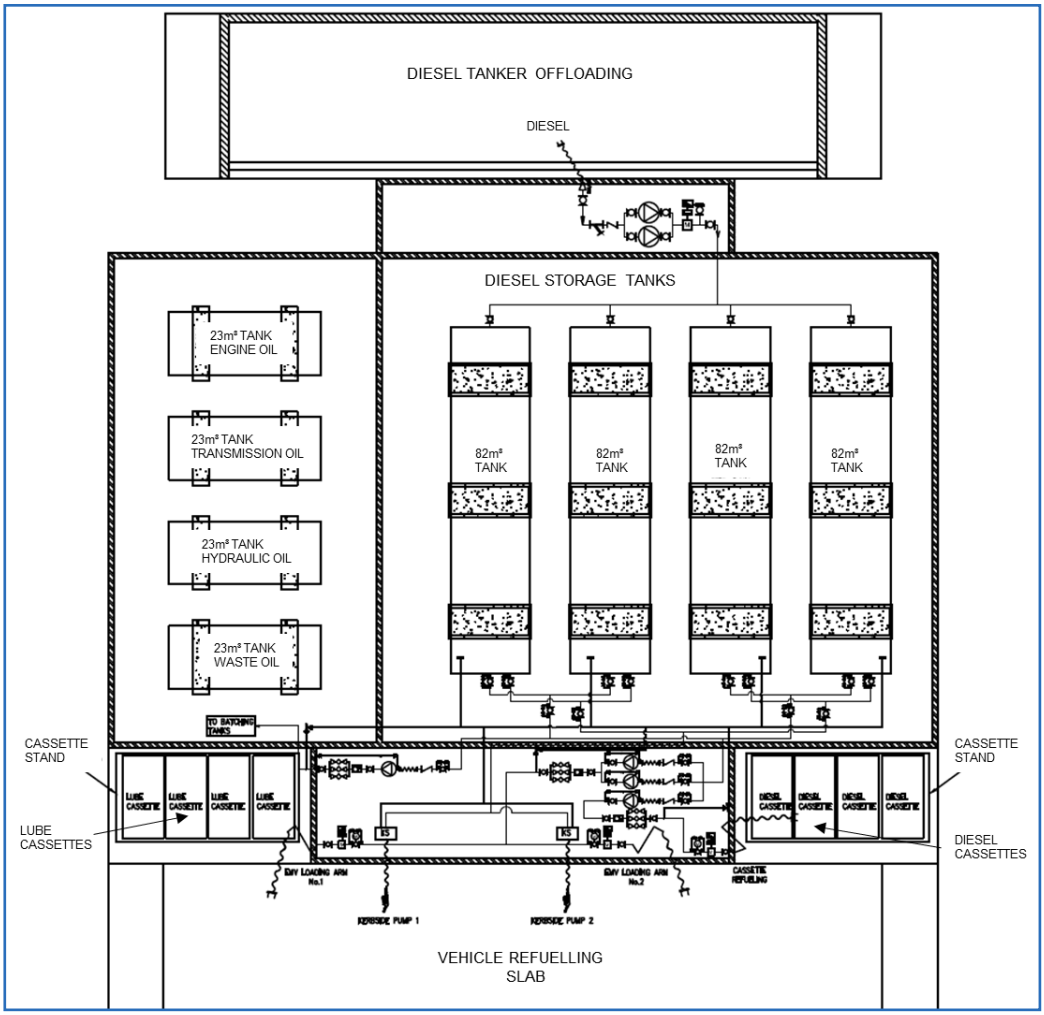
Figure 18.35 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Diesel Receiving and Dispensing



DRA, 2017

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 batched via piping to the underground lube storage tanks.

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:

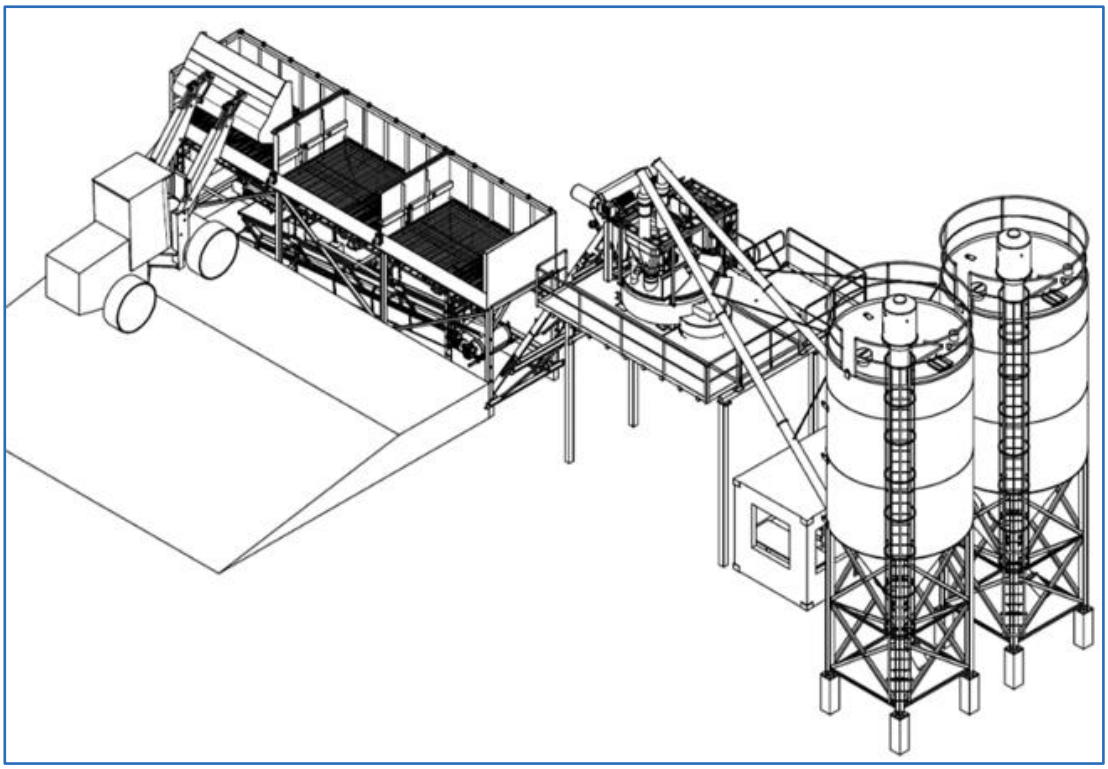
- Engine oil: 1 x 23 m³ tank
- Transmission oil: 1 x 23 m³ tank
- Hydraulic oil: 1 x23 m³ tank
- Waste oil: 1 x 23 m³ tank

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.

Stopes will be filled with waste rock or paste fill. For this purpose, bulkheads will be constructed at the base of the stopes using fibre shotcrete. Underground civil construction material will also be supplied from this plant. This plant is designed to adjust the mix according to the requirements, with flushing facilities to clean out the complete system prior to any changeover. Figure 18.36 shows the layout of the surface shotcrete and concrete batch plant.

Figure 18.36 Surface Shotcrete and Concrete Batch Plant



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The batch plant is equipped with a compulsory planetary mixer L1000, capable of mixing a variety of concrete requirements, including steel and synthetic fibres, although if fibres and/or steel reinforcement is required for underground applications, it will be added underground. The mixer can produce high-quality shotcrete, ready-mix, mortar, and earth fry concrete. Production capacity varies from 10 m³/h up to 40 m³/h depending on the mixing period, materials used and requirements.

The plant includes three aggregate silos filled by a wheel loader. Cement is added by a screw system from the cement silo.

There is an optional additive system for two liquid additives, including tanks, pumps and scale. This plant can be equipped with a dosing system for steel or synthetic fibres. The system will weigh and dose fibres automatically according to the requirements. Fibres are tumbled in the mixer to ensure homogeneous concrete and the best spraying result.

The optional micro silica dosing system enables for specific concrete options. Micro silica is dosed from bulk bags to scale.

Concrete is mixed into a wet sludge and discharged into one of two boreholes feeding the underground receiving areas, located in dedicated excavations on 750 m, 850 m and 950 m Levels.

18.12.5 Vent Raise and Fans

The ventilation system is designed as an exhausting pull system. The main intake shafts (Shaft 1, Shaft 2 and Shaft 5), located at the centre of the mining district, will provide fresh air, while return air will exhaust to surface through three exhaust shafts (Shafts 3, 4 and 6) using a combination of surface and underground fan stations. Refer to Section 16.2.9 for further detail on the overall ventilation system for the Project.

18.12.6 Backfill Plants

18.12.6.1 Surface Installation of Backfill Plants

Ivanplats appointed Paterson & Cooke Consulting Engineers (P&C) to develop a revised Feasibility Study for the implementation of the final high-rate Paste filling plant to accommodate the final mine production profile for the Platreef 4Mtpa Phase 2 FS.

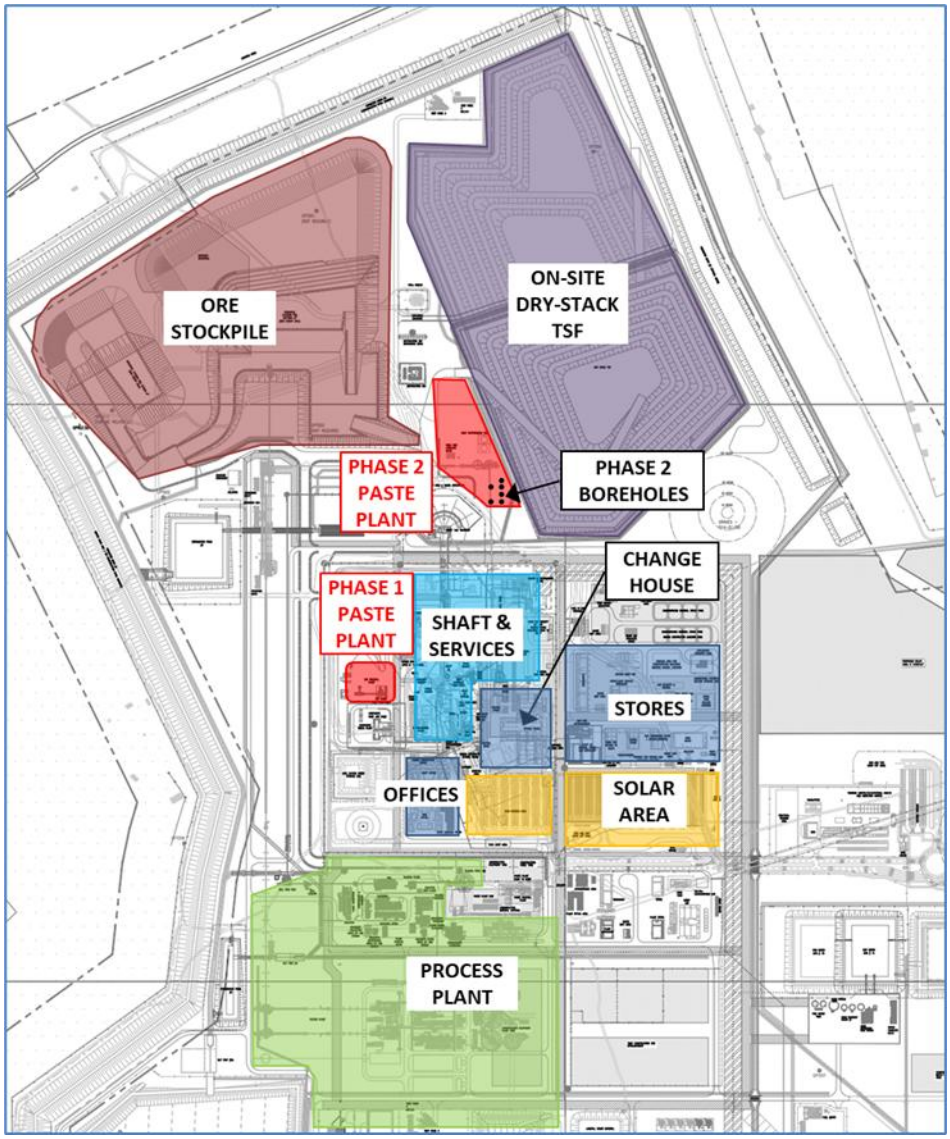
The existing backfill system construction during Phase 1 will be replaced by a Phase 2 backfill system to account for the increase in mining throughput from 2028. The Phase 2 final high-throughput system consists of a two-module pressure filtration and paste plant that receives flotation tailings from the concentrator plant in the form of thickened underflow slurry. The produced cemented paste will be pumped underground via the high-pressure underground distribution system to the mined stopes' deposition points.

Refer to Figure 18.37 for the surface position of the back fill plants in relation to the rest of the mining and process plant surface infrastructure.

The revised production plan proposed will utilise Cemented Paste Fill for all stopes, regardless of the mining method. In certain secondary stopes, the CPF will be supplemented with waste rock. The waste rock will be trucked from underground and dumped into the stopes from the top access drive.

Refer to Section 16 of the technical report for further details on the backfill systems.

Figure 18.37 Site Block Plan with Phase 1 and Phase 2 Backfill Plants



18.12.7 Traffic Flow Study

Simulation Engineering Technologies was engaged to construct a simulation model representing the traffic flow into and out of the mine in 2017. The report was not revisited during this study as the N11 intersection has been constructed, the traffic volumes do not vary materially from those already considered, and the outcomes of the traffic study are incorporated into the design.

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 road infrastructure was modelled to determine if the planned vehicle flow is sufficient to support surface 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 simulation indicated that for the base case, inbound traffic is constrained, resulting in significant queuing up to the intersection and onto the N11. This impacts both mine and N11 traffic, as well as personnel arrival times and vehicle travel times.

Outbound traffic flow is also constrained, resulting in vehicle queuing, especially for employee 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, the most important of which is concentrate collection. The weighbridge also experiences intermittent queuing problems.

The outcome is that the base case scenario, using a two-way stop at the N11 intersection, is a concern and that the main gate entrance operations result in significant queuing. The two-way stop traffic rules constrain traffic flow out of the mine onto the N11.

Employee cars account for 96.5% of vehicle arrivals. For the base case, cars travel on average 15.96 minutes inbound and 5.17 minutes outbound.

There are, on average, 60 vehicles (all types) queuing to enter the mine when a car arrives and a queue of 42 vehicles when a car exits.

It was determined that processing time at the main gate is the biggest concern and causes the bottleneck for inbound vehicles. Vehicles not entering through the main gate are also affected, as there will already be other vehicles queuing when they arrive.

The solution is to reduce the processing time at the main entrance.

18.12.8 Surface Fleet

The operations fleet has been purchased as part of Phase 1. No additional fleet has been allowed for as part of Phase 2.

18.13 Security and Access Control

18.13.1 Control

18.13.1.1 Access Control

This system and infrastructure were installed in Phase 1. All required expansions for the next phase will be integrated into the existing system. In general, the system operates as follows.

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

This system and infrastructure were installed in Phase 1. All required expansions for the next phase will be integrated into the existing system.

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

This system and infrastructure were installed in Phase 1. All required expansions for the next phase will be integrated into the existing system.

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

18.13.2.1 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 2025 FS.

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.

18.13.2.2 Access Control Points

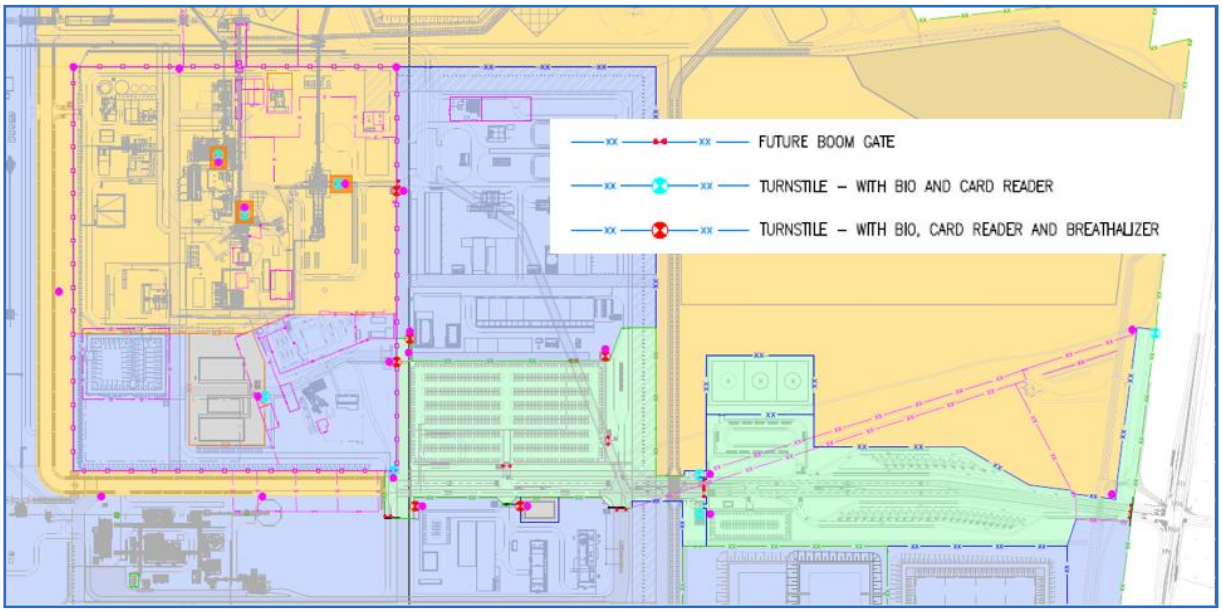
Various access control buildings have been allowed within the mine area as part of the Phase 2 development as listed in Table 18.9 below.

Table 18.9 Access Control Buildings

Phase	Drawing Description	Building Description
2	Shaft Area Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the shaft area.
2	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.
2	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.
2	Main Entrance Pedestrian Gate House	Gate House with turnstiles controlling the entrance of visitors to the mine and employee access to the engagement centre.
2	Main Office Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the main office area.

An excerpt from the Mine Security Layout Plan indicating the permanent access control points for the current Phase 2 Blockplan is shown in Figure 18.38.

Figure 18.38 Permanent Access Control Points



DRA, 2025

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:

- Oil traps and oil storage tanks for oil interception and recovery, and temporary storage of used oil.
- Salvage yard for temporary storage of scrap metal and re-usable mechanical parts.
- Permanent stormwater pond for capture and storage of contaminated water generated on site.
- Sewage treatment plant to treat sewage and grey water produced on site.
- Water filtration plant to treat the bulk make-up water supply from the Masodi 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 DSTF.

- 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 DSTF.

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.

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
- Salvage Yard
- Oil Traps
- Waste Collection Bins
- Sewage Treatment Plant

The following waste facilities have been removed from the Phase 2 Study:

- General Waste Landfill
- Composting Plant
- General Waste Storage and Recycling Area
- Hazardous Waste Storage Area
- Waste Tyre Storage
- Leachate Pond

18.14.3.1 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-usable 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.

18.14.3.2 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.

18.14.3.3 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 18.10 for details.

Table 18.10 Waste Bin Suggested Colour Coding System

Waste Category	Colour of Containers
Printer cartridges	**Red**
Hazardous waste	**Black**
Plastic	**Blue**
Domestic	**Green**
Paper	**White**
Cans	**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:

18.15.1 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.

18.15.2 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.

18.15.3 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.

Additional provision during Phase 2 construction was made for improvement and expansion of construction site establishment and laydown areas. Two additional proposed areas for Phase 2 Construction laydown are:

- Northwestern part of the Dry Stack TSF close to the proposed destruction bay.
- The area west of the Salvage yard at the southern part of the site.

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.

18.15.4 Construction Office

The prefabricated 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.

18.15.5 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.

18.15.6 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.

18.15.7 Construction Vehicles

Construction vehicles have only been allowed for the EPCM project team. Existing project vehicles will be utilised.

18.15.8 Construction SHEQ

Allowance was made for lighting detection equipment, breathalysers, 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.

18.15.9 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.

18.15.10 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.

18.15.11 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 Pd and Rh use over Pt 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 Pd loading in these vehicles has driven automotive demand for Pd to an all-time high of 9.7 million ounces in 2019 based on data from Johnson Matthey. Similarly, higher loading of Rh in automobiles to reduce NOx emissions for real driving emissions (RDE) tests has resulted in significant tightness for the smaller Rh 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 Pt 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 Pt-based catalysts bringing the average Pt requirement for each fuel cell electric vehicle to approximately 10–20 g of Pt. With the typical diesel engine only requiring 5–6 g of Pt, the adoption of fuel cell electric vehicles has the potential to not only replace the loss in diesel demand but drive Pt 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 Pt 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 Pd and Rh prices, influencing the long-term strategic decisions made by producers. However, low global stocks of Pd, in particular those stocks in Russia, and Rh 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 10 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; and 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. Rh), such that these terms can have an impact on the sellers' cash flow although Pt, Pd 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 IDP25 Smelter Terms and Price Assumptions

The prices in the economic analysis for the Platreef 2025 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 $1,200/oz Pt, $1,130/oz Pd, $2,170/oz Au, $5,000/oz Rh, $8.50/lb Ni, and $4.25/lb Cu. For the Platreef 2025 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 IDP25 will produce significantly larger volumes of concentrate once steady-state 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 run-off 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 re-occurrence 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 Permitting Status

Ivanplats conducted an amendment of some of the existing authorisations and licences to facilitate the aspects of Phase 2 of the Platreef Project in 2022. This process required a full ESIA process before commencing with the proposed changes. Subsequently, following the amendment process, in 2023, an integrated EA was granted by the Department of Mineral Resources and Energy (DMRE) and an amended WUL and secondary WUL by the Department of Water and Sanitation (DWS).

- The following changes informed the amendment process as completed:
  - Changes to the infrastructure plan for Phase 1, such as:
  - The inclusion of the Masodi Waste Water Treatment Works (WWTW) pipeline.
  - A change of the existing authorised waste rock footprint to an on-site dry tailing storage facility (DTSF).
  - Repositioning of the waste rock footprint.
  - Establishment of an ore stockpile (run-of-mine).
  - Construction and operation of a 700 ktpa crushing and screening plant.
  - Expansion of stormwater management systems, adding two additional dams to service the DTSF and concentrator plant areas.
  - Expansion of the existing diesel based electricity generating capacity to more than 20 MW.

  - The change of the authorised upstream conventional tailings storage facility (TSF) (Rietfontein KS) to a DTSF (In progress).
  - The backfilling with residue material into the mine's underground voids.
- Changes to the Phase 2 infrastructure such as the inclusion of an additional cut-off drain for the existing concentrator plant, the south-western clean stormwater management drain expanded further down to the Mogalakwena River and the upgrading of internal haul roads.
- These amendments required a full ESIA process prior to the commencement of these activities.
- Additional specialist studies and public participation have been undertaken to inform the applications for these amendments, which were authorised in 2023.
- In an effort to reduce the Mine's carbon footprint and to ensure continued electricity supply for the Platreef operations, Ivanplats investigated alternative energy resources for the operations. Various options were investigated, and the development of a 19 MW solar photovoltaic (PV) facility was selected as the most feasible option. A basic ESIA authorisation process was conducted to attain the environmental license to develop the facility, with the associated electrical infrastructure, in 2022, with an authorisation granted by the DMRE in 2023.

The solar facility development will be undertaken in a phased approach, commencing with a 5 MW supply as Phase 1 and forecast operational in Q1'25.

20.3 Water Use Licence (WUL)

20.3.1 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.

20.3.2 Integrated Water Use Licence (IWUL) for Main Mine

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.

20.3.3 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. However, all waste activities applicable to the project were incorporated into the environmental application and authorised as such in the EA since 2022.

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 (Responsible QP, Bernard Peters and Curtis Smith)
- Area 2000 – Mining (Responsible QP, Bernard Peters and Curtis Smith)
- Area 3000 – Concentrator (Responsible QP, Louise Lintvelt)
- Area 6000 – Infrastructure, Utilities and Ancillaries (Responsible QP, Ryan Males)
- Area 7000 – Site Costs (Responsible QP, Ryan Males)
- Area 8000 – Owner's Costs and General and Administration (Responsible QP, Ryan Males)
- Area 9000 – Escalation and Contingency (QPs as per Above)

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 2025, first concentrate in 2025, ramp up to full production steady state of 4.1 Mtpa in 2030 and finally mine closure requirements in 2059.

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 G5 and G7 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.
- Overland piping and the piping for the 3.3 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, Control and Instrumentation (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 re-skill 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. 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 established optimised work cycle consists of four teams operating on an 11.5-hour, four-day shift schedule, followed by a four-night shift period, and culminating in 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 to achieve a steady production capacity of 0.77 Mtpa during the initial three years, with an anticipated increase to 4.1 Mtpa in 2030. The initial construction and development phase 1 will facilitate a production rate of 0.77 Mtpa. Subsequently, a new concentrator module, with a capacity of 3.3 Mtpa, will be constructed, thereby enhancing the overall production capacity to 4.1 Mtpa in the following phase 2. As the construction and development phases progress, the workforce will be adjusted accordingly until the mining operations commence and the target production of 4.1 Mtpa is achieved. Employment will be structured to ensure that qualified personnel are optimally positioned at the appropriate times to ensure that the mine is operationally prepared.

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

21.4.2 Area 2000 – Mining

The capital costs associated with:

- Shaft 2:
  - Finalising the construction of the headframe.
  - Equipping of shaft and associated infrastructure as required for production.
- Shaft 3:
  - Constructing the headframe.
  - 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 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).
    - Ventilation systems.
    - Electrical, control and instrumentation.
    - Settling and dewatering.
    - Crushing.
    - Underground workshops.
- Backfill System and UG Distribution System Paste Fill Plant:

  - Paste plant.
  - Paste underground distribution system.
- Mining Control System:
  - Establishment of all the control system infrastructure.
- Commission and Spares.
- EPCM.

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, and include the following:

- Labour – All mining related labour, including technical services.
- Power – Bulk Eskom 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.

21.4.3 Area 3000 – Concentrator

The capital cost required for the maintenance of the concentrator plant during Phase 1, which has a capacity of 770 ktpa, as well as for the construction of the concentrator plant in Phase 2, with a capacity of 3.3 Mtpa, includes necessary associated infrastructure. This infrastructure will facilitate the treatment of 770 ktpa at steady state ore production during Phase 1 and 4.1 Mtpa during Phase 2, ensuring that the production of concentrate adheres to the specified volumes and quality standards. 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 as follows:

21.4.4.1 Bulk Power

The bulk power estimate includes provision for additional 33kV circuit breakers and cable installation in the existing consumer substation, the electricity Opex costs as per the Eskom Megaflex tariff and savings due to the lower cost for the solar renewable energy installations.

21.4.4.2 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.

21.4.4.3 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, on quotations provided by the contractors.

21.4.4.4 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 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.

21.4.4.5 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.

21.4.4.6 Spares

Spares were collated into three groups, comprising capital spares, operating spares and commissioning spares. Capital spares are 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 were included as part of the operating cost estimate, and commissioning spares were priced as part of the 3.5% allowance.

21.4.4.7 EPCM

For the Platreef 2025 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:

- Training and development.

21.4.6 Area 8000 – Owner's Costs, Administration and Overheads

The capital costs associated with:

21.4.6.1 Project Services

Project execution services other than EPCM. This includes but not limited to:

- Commissioning
- Project consultant fees (other than EPCM)
- Financing fees
- Logistics and transport

21.4.6.2 Owner's 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

21.4.6.3 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.

21.4.6.4 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 Q4'24 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 net outcome of 6.67% on the items in these areas.

21.5 Cost Basis and Exchange Rates

In the Platreef 2025 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 2025 Exchange Rates

Exchange Rate	Rate
ZAR/USD	18.5
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 1 leads to the first concentrate date and the critical path of Phase 2 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 4.07 Mtpa steady state production target. The key dates are shown in Table 21.2.

Table 21.2 Platreef 2025 FS Key Dates

Activity Name	Start	Finish
Shaft 3 Equipping	Q4'24	Q1'26
Shaft 2 Sinking to –60 m Level	Q4'25	Q1'26
First Concentrator Start Up		Q4'25
Shaft 2 Sinking to –995 m Level	Q1'26	Q1'28
Shaft 2 Equipping Complete		Q3'28
Plant 2 Start Up		Q4'27
Mine Production Steady State (4.07 Mtpa)	Q4'29	

21.7 Cost Estimation Method

The following techniques were used in preparation of the capital and operating cost estimates of the Platreef 2025 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 and 3, 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 2025 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 2025 FS. Costs estimated in ZAR have been converted to US dollars at an exchange rate of 18.50 ZAR/USD. A comparison between the exchange rates used in the Platreef 2025 FS and the 2022 FS is shown in Table 21.3.

Table 21.3 Exchange Rates

Exchange Rates	Forex Amount	
	2025 FS	2022 FS
ZAR	18.5	16
EUR	0.71	0.71
AUD	1.18	1.18
CNY	5.7	5.7
GBP	0.63	0.63
JPY	100	100
NOK	7.00	7.00
SEK	7.00	7.00

21.8 Capital Cost Summary

The total capital cost includes initial (pre-production and expansion) 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 ($M)	Sustaining ($M)	Total ($M)
Mining			
Exploration and Geology	2	–	2
Mining	578	1,364	1,942
Capitalised Operating Costs	75	–	75
Subtotal	**655**	**1,364**	**2,019**
Concentrator and Tailings			
Concentrator	215	30	245
Capitalised Operating Costs	–	–	–
Subtotal	**215**	**30**	**245**
Infrastructure			
Infrastructure	153	74	227
Site Costs	2	–	2
Capitalised Operating Costs	3	–	3
Subtotal	**158**	**74**	**232**
Other Costs			
Owner's Cost	95	34	129
Closure Cost	–	9.5	9.5
Capitalised Operating Costs	20	–	20
Subtotal	**116**	**44**	**159**
Capex Before Contingency	**1,144**	**1,511**	**2,655**
Contingency	99	79	178
Capex After Contingency	**1,243**	**1,589**	**2,832**

1. Remaining capital cost for Phase 1 is $70M.
2. Totals may 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 1–3 Average	Years 4–35 Average	LOM Average
Concentrate Transport	117	0.89	0.90	0.90
Site Operating Costs				
Mining	4,922	62.84	37.53	37.94
Processing and Tailings	1,789	18.43	13.71	13.79
Infrastructure	363	6.28	2.74	2.80
General and Administration	722	26.40	5.23	5.57
Total	**7,796**	**113.95**	**59.22**	**60.09**

Totals may vary due to rounding.

22 ECONOMIC ANALYSIS

22.1 Summary of Financial Results

The key features of the Platreef 2025 FS include:

- Development of a large, mechanised, underground mine is planned at an initial 770 ktpa and expansion to 4.07 Mtpa.
- Planned average annual production rate of 432 koz of platinum, palladium, rhodium and gold (3PE+Au).
- Estimated initial capital requirement of approximately $1,243M.
- After-Tax NPV of $1,393M, at an 8% discount rate.
- After-Tax IRR of 19.77%.
- The Platreef 2025 FS maintains options available to accelerate expansions, to the 10.7 Mtpa scenario, 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 2025 FS are shown in Table 22.1.

Figure 22.1 shows the Development and Production Timeline for the Platreef 2025 IDP.

Figure 22.1 Platreef 2025 IDP Development and Production Timeline

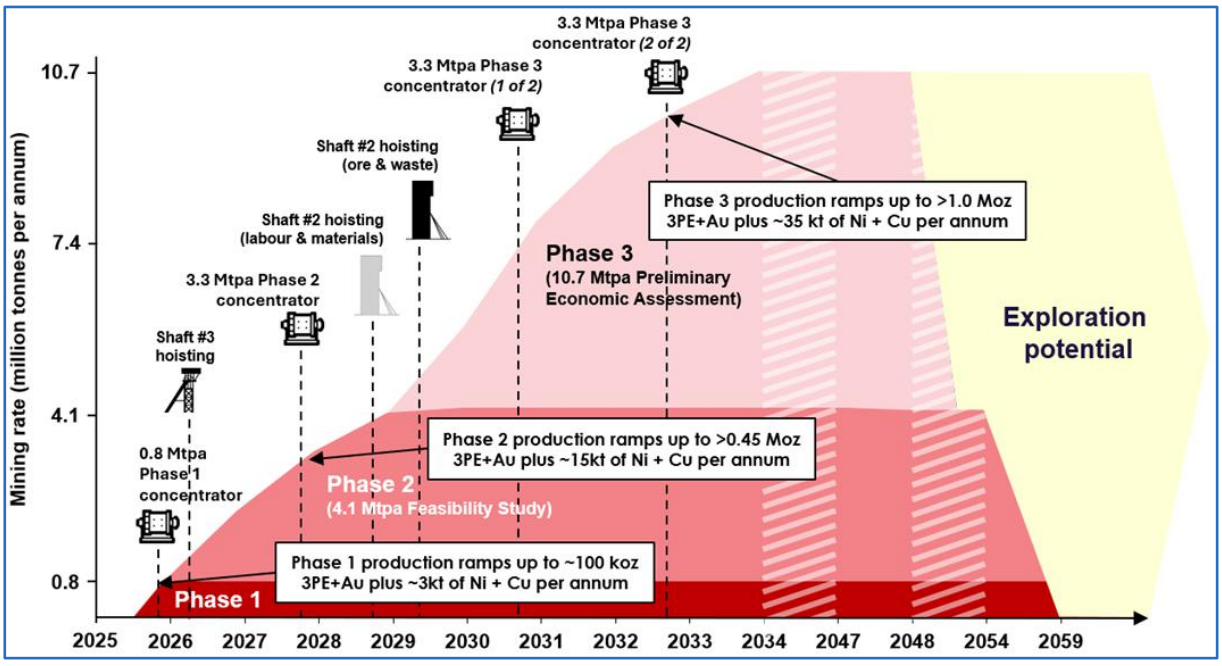


Figure 22.2 Platreef 2025 FS Mining Production

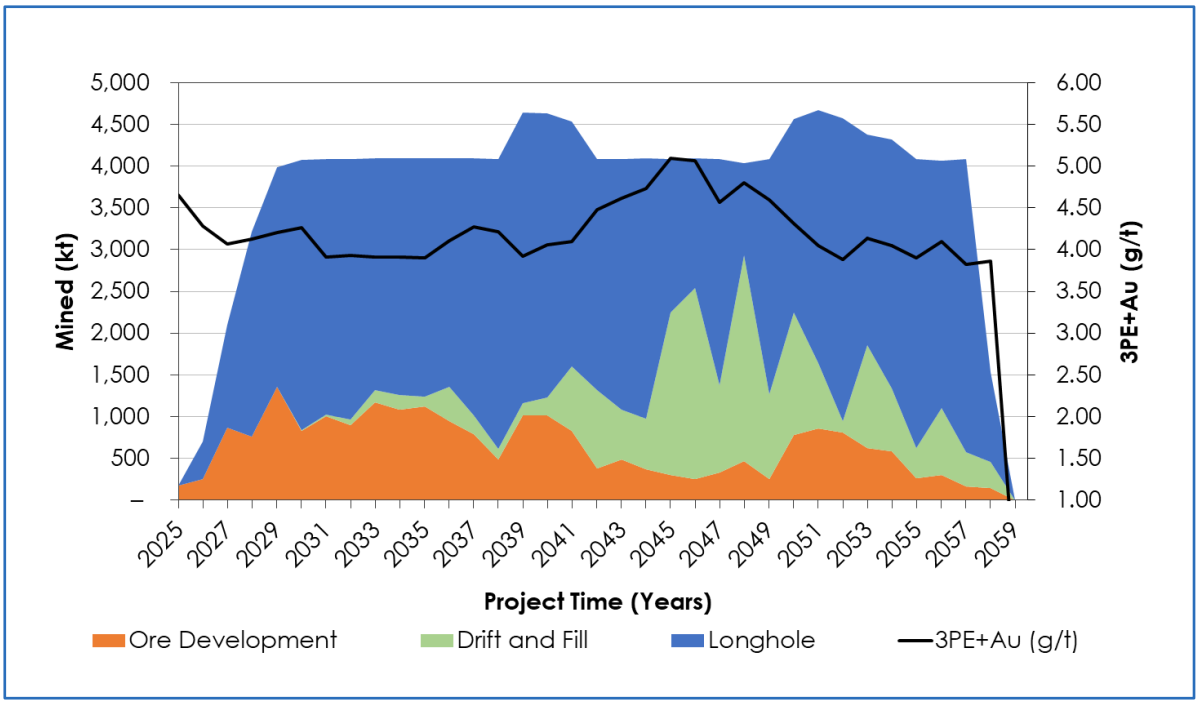


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Figure 22.3 Platreef 2025 FS Cumulative Cash Flow After-Tax

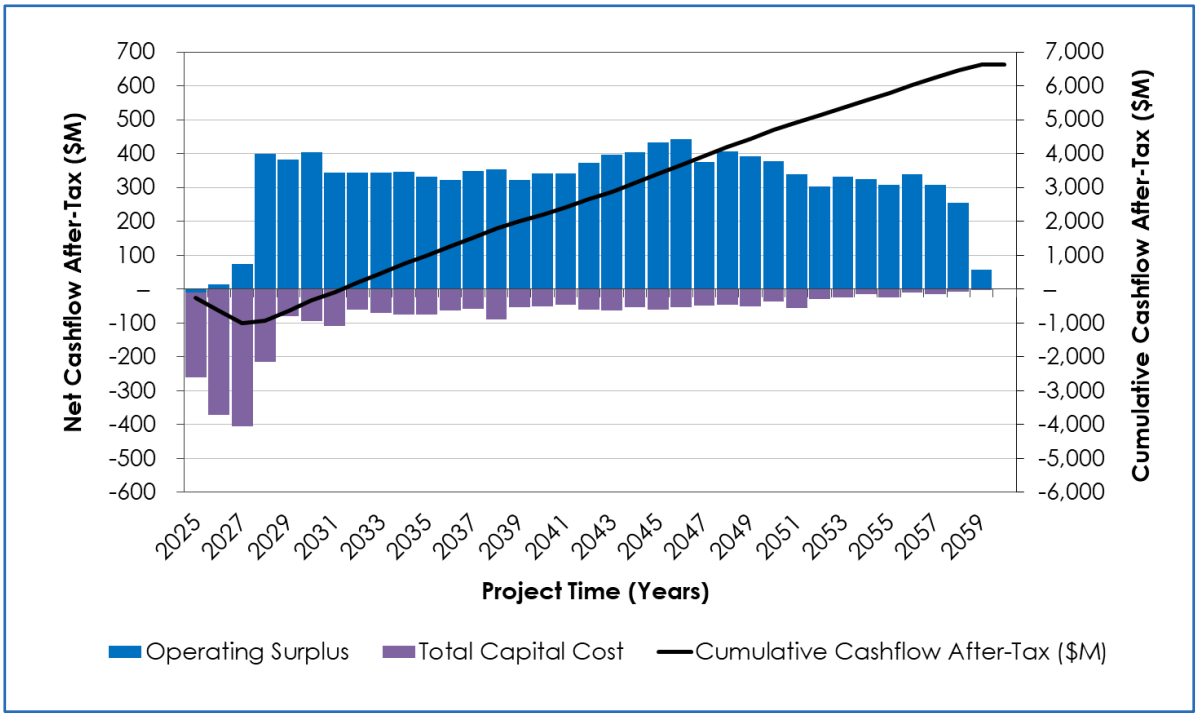

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Table 22.1 Platreef 2025 FS Key Production and Financial Results

Item	Unit	Total
Mined and Processed	Mt	130
Platinum	g/t	1.88
Palladium	g/t	1.93
Gold	g/t	0.29
Rhodium	g/t	0.13
3PE+Au	g/t	4.22
Copper	%	0.16
Nickel	%	0.33
Concentrate Produced	kt	5,523
Platinum	g/t	38.28
Palladium	g/t	39.11
Gold	g/t	5.33
Rhodium	g/t	2.43
3PE+Au	g/t	85.15
Copper	%	3.31
Nickel	%	5.47
Contained Metal		
Platinum	koz	6,797
Palladium	koz	6,944
Gold	koz	947
Rhodium	koz	432
3PE+Au	koz	15,120
Copper	Mlb	403
Nickel	Mlb	666
Key Financial Results		
Life-of-Mine	Years	35
Initial and Expansion	$M	1,243
Sustaining Capital	$M	1,589
Mine-Site Cash Cost	$/oz Rec. 3PE+Au	516
Total Cash Costs After Credits	$/oz Rec. 3PE+Au	599
Site Operating Costs	$/t Milled	60
After-Tax NPV8%	$M	1,393
After-Tax IRR	%	20
Project Payback Period	Years	7.8

1. Initial Capital includes a remaining $70 million for the completion of phase 1.
2. Totals may vary due to rounding.
3. 3PE+Au = platinum, palladium, rhodium and gold.
4. Economic analysis metal price assumptions: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.

22.2 Model Assumptions

22.2.1 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2025. It is presented in 2025 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 2025 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 2025 FS are shown in Table 22.2.

Table 22.2 Platreef 2025 FS Commodity Price Assumptions

Parameter	Unit	2025	2026	2027	Long Term
Platinum	$/oz	1,130	1,200	1,240	1,200
Palladium	$/oz	1,090	1,110	1,090	1,130
Gold	$/oz	2,550	2,480	2,400	2,170
Rhodium	$/oz	4,670	5,000	5,000	5,000
Copper	$/lb	4.60	4.70	4.80	4.25
Nickel	$/lb	8.10	8.30	8.10	8.50

22.2.2 Treatment Charges and Refining Charges

In the Platreef 2025 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 Table 22.3.

Table 22.3 Platreef 2025 FS Refining Charges (% Gross Sales)

Parameter	Refining Charges
Copper	28.8%
Nickel	30.0%
Platinum	16.5%
Palladium	16.5%
Gold	16.5%
Rhodium	17.3%

22.2.3 Concentrate Transport Costs

In the Platreef 2025 FS the average distance of the smelters is 277 km. The transport cost applied to the financial model is $0.07 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 (27%) 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 2025 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	%	27.00
Opening Tax Losses	million ZAR	129
Opening Depreciation	million ZAR	21,483
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 18.50 ZAR/USD. A comparison between the exchange rates used in the Platreef 2025 FS and the 2022 FS is shown in Table 22.5.

Table 22.5 Exchange Rates

Exchange Rates	Forex Amount	
	2025 FS	2022 FS
ZAR	18.5	16.00
EUR	0.71	0.71
AUD	1.18	1.18
CNY	5.70	5.70
GBP	0.63	0.63
JPY	100.00	100.00
NOK	7.00	7.00
SEK	7.00	7.00

22.3 Project Results

The results of the financial analysis show an After-Tax NPV8% of $1,393M. The Platreef 2025 FS exhibits an After-Tax IRR of 19.77% and a payback period of approximately eight years. The estimates of cash flows have been prepared on a real basis as at 1 January 2025 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 2025 FS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	8,338	6,628
	5.00%	3,001	2,450
	8.00%	1,689	1,393
	10.00%	1,150	950
	12.00%	769	632
	15.00%	386	306
	20.00%	24	-11
Internal Rate of Return	–	20.48%	19.77%
Project Payback Period (Years)	–	7.80	7.80

Table 22.7 Platreef 2025 FS Mining Production Statistics

Item	Unit	Total LOM	Years 2025 – 2027 Average	Years 2028 - 2059 Average	LOM Average
Mined and Processed	Mt	130	0.92	3.99	3.71
Platinum	g/t	1.88	1.85	1.88	1.88
Palladium	g/t	1.93	1.87	1.93	1.93
Gold	g/t	0.29	0.30	0.29	0.29
Rhodium	g/t	0.13	0.12	0.13	0.13
3PE+Au	g/t	4.22	4.14	4.23	4.22
Copper	%	0.16	0.18	0.16	0.16
Nickel	%	0.33	0.36	0.33	0.33
Concentrator Recoveries					
Platinum	%	86.88	85.87	86.90	86.88
Palladium	%	86.29	85.19	86.30	86.29
Gold	%	78.33	77.42	78.35	78.33
Rhodium	%	79.78	78.61	79.80	79.78
3PE+Au	%	85.81	84.74	85.82	85.81
Copper	%	87.33	88.96	87.30	87.33
Nickel	%	70.81	73.55	70.76	70.81
Concentrate Produced	kt (dry)	5,523	29	170	158
Platinum	g/t	38.28	37.61	38.29	38.28
Palladium	g/t	39.11	37.76	39.13	39.11
Gold	g/t	5.33	5.55	5.33	5.33
Rhodium	g/t	2.43	2.18	2.44	2.43
3PE+Au	g/t	85.15	83.11	85.18	85.15
Copper	%	3.31	3.84	3.30	3.31
Nickel	%	5.47	6.29	5.45	5.47
Contained Metal					
Platinum	koz	6,797	47	209	194
Palladium	koz	6,944	47	214	198
Gold	koz	947	7	29	27
Rhodium	koz	432	3	13	12
3PE+Au	koz	15,120	103	465	432
Copper	Mlb	403	3	12	12
Nickel	Mlb	666	5	20	19

Concentrator feed and estimated concentrator produced along with grades for the life-of-mine are depicted in Figure 22.4 and Figure 22.5.

Figure 22.4 Platreef 2025 FS Concentrator Production

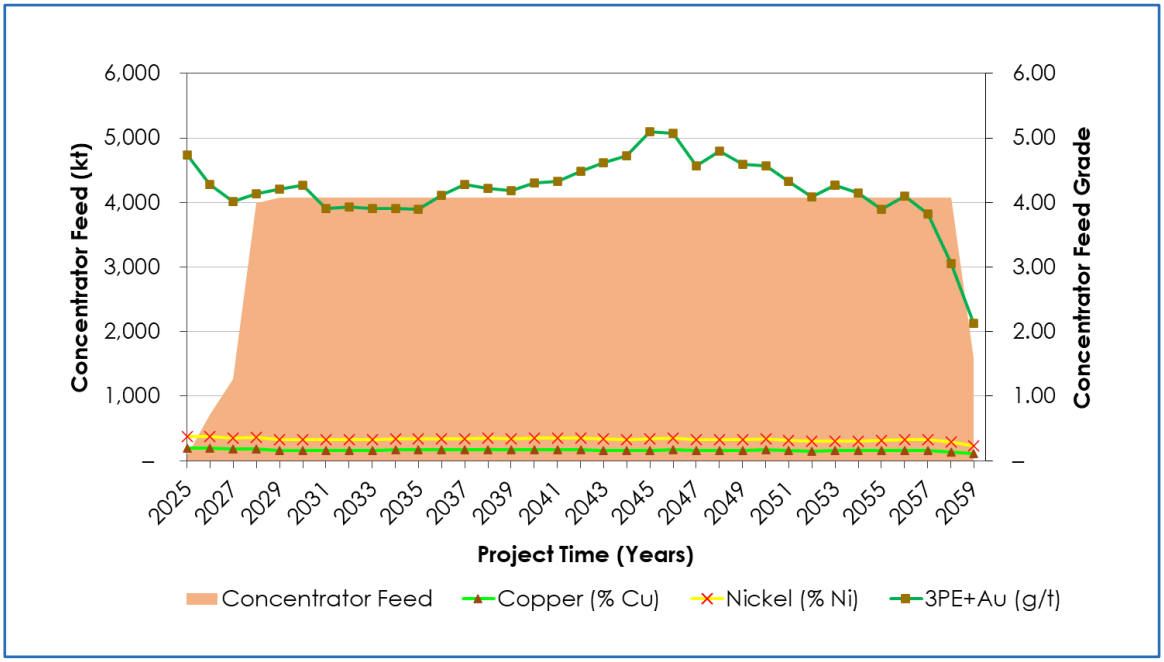

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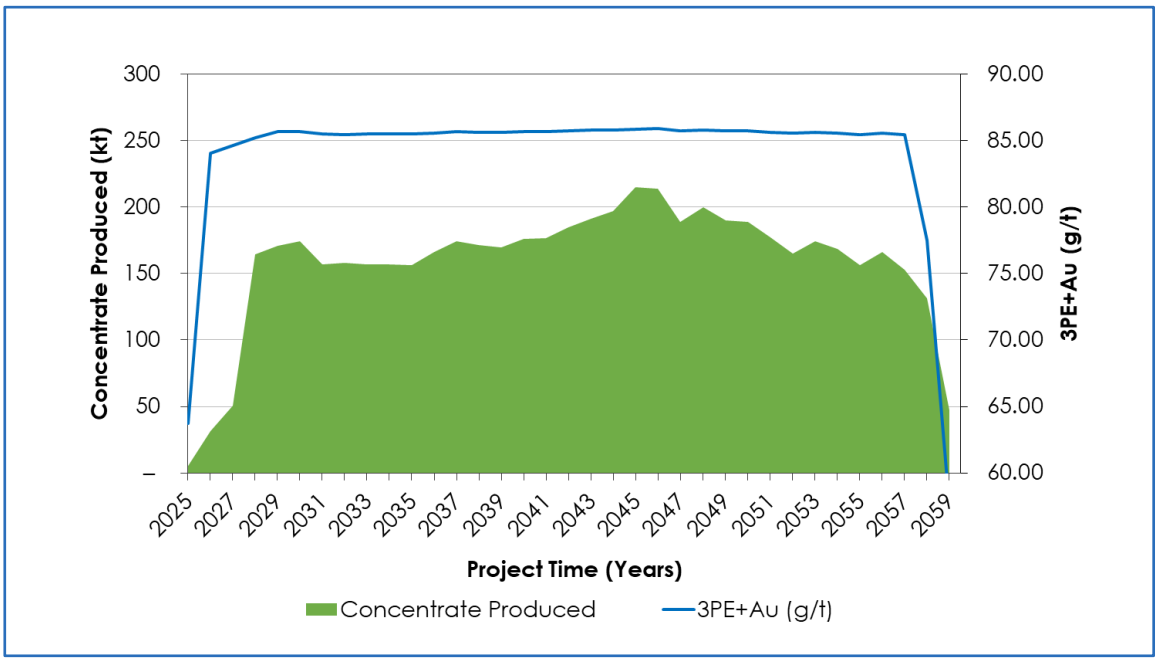
Figure 22.5 Platreef 2025 FS Estimated Concentrate Produced and 3PE+Au Grade



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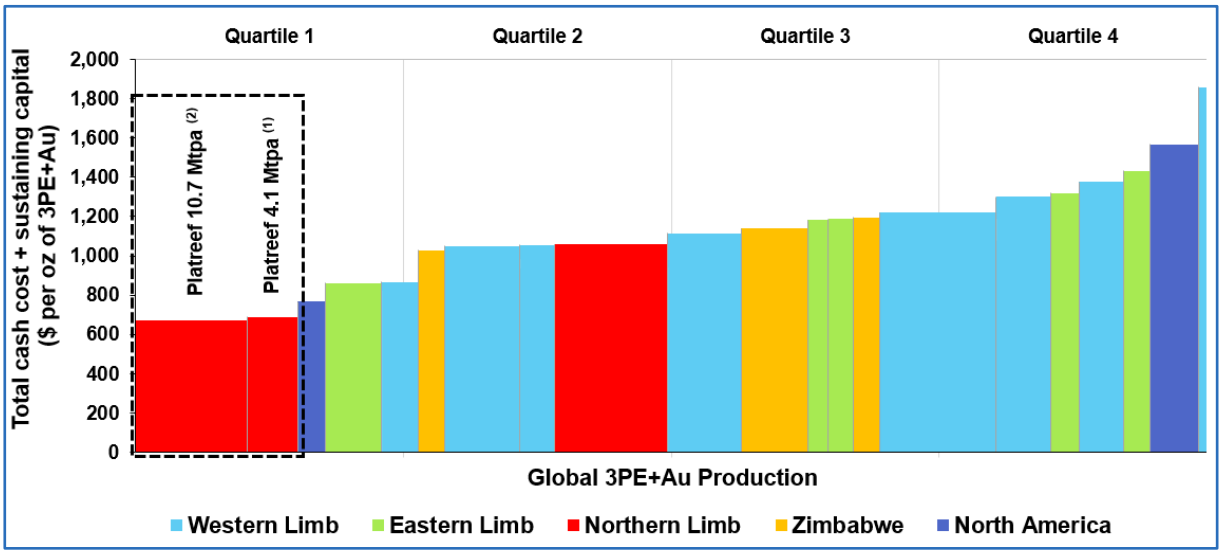
22.4 Capital and Operating Cost Summary

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 2025 FS Cash Costs After Credits

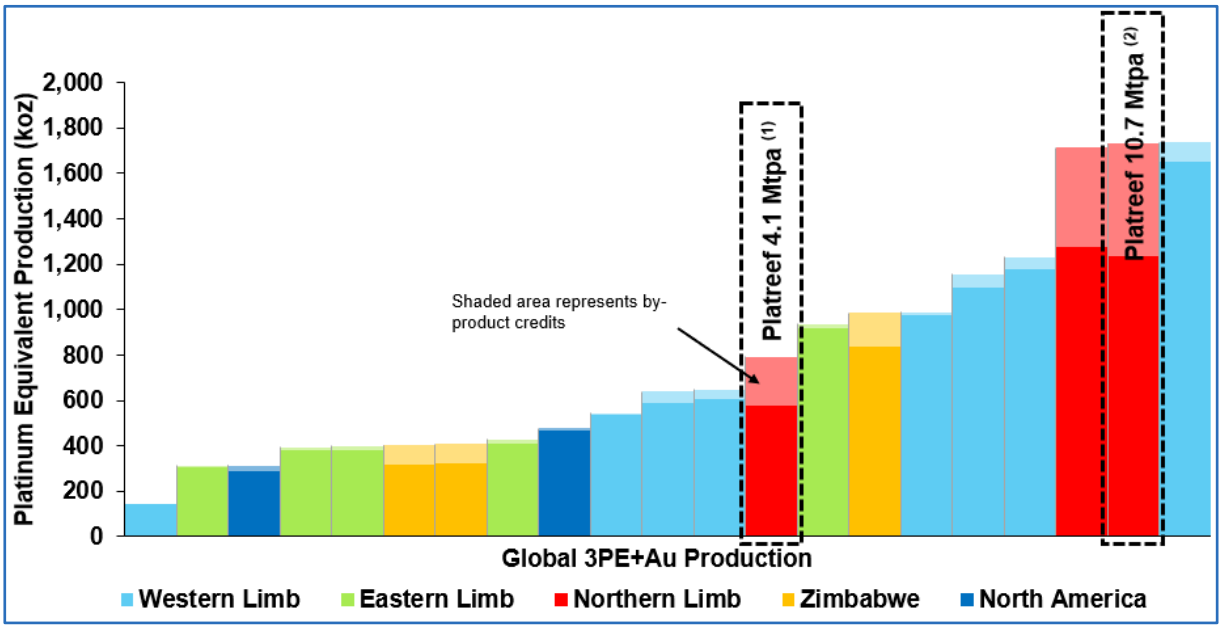
Item	Recovered 3PE+Au ($/oz)		
	Stage 1 2025 - 2027 Average	Stage 2 2028 - 2059 Average	LOM 2025-2059 Average
Mine Site Cash Costs	1,010	508	516
Transport	8	8	8
Treatment & Refining	393	366	367
Au, Pd & Pt Streaming Agreements	156	118	119
Royalties	8	78	77
Total Cash Costs Before Credits	**1,575**	**1,078**	**1,086**
Nickel Credits	424	373	374
Copper Credits	151	113	114
Total Cash Costs After Credits	**1,000**	**592**	**599**
Sustaining Capital Costs	–	108	105
Total Cash Costs After Credits & Sustaining Capital	**1,000**	**700**	**704**

Figure 22.6 Global Primary Producers' Net Total Cash Cost + Sustaining Capital (2024E), US$/3PE+Au oz



Source: SFA (Oxford), Ivanplats. Notes: Cost and production data for the Platreef project is based on the Platreef's 2025 4.1 Mtpa FS and 10.7 Mtpa PEA parameters, applying SFA South African industry average smelting and refining costs. SFA's estimated peer group cost and production data for 2024 is based on H1 2024 figures, extrapolated out to produce an estimate for the full calendar year and 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 2024 average basket prices and exchange rates of 18.78:1 ZAR:USD, US$980/oz platinum, US$1,009/oz palladium, US$4,753/oz rhodium, US$2,300/oz gold, US$17,150/t nickel and US$8,727/t copper. (1) Platreef 4.1 Mtpa between years 4 to 35. (2) Platreef 10.7 Mtpa between years 4 to 29.

Figure 22.7 Ranking of Selected Global Primary PGM Producers, based on 2024E Palladium Equivalent Production



Source: SFA (Oxford), Ivanplats. Notes: Chart only includes primary PGM producers. Cost and production data for the Platreef project is based on the Platreef's 2025 4.1 Mtpa FS and 10.7 Mtpa PEA parameters. Production data for the peer group is provided by SFA (Oxford). Equivalent platinum production has been calculated using average 2024 prices and exchange rates of 18.78:1 ZAR:USD, US$980/oz platinum, US$1,009/oz palladium, US$4,753/oz rhodium, US$2,300/oz gold, US$17,150/t nickel and US$8,727/t copper. (1) Platreef 4.1 Mtpa FS between years 4 to 35, (2) Platreef 10.7 Mtpa PEA between years 4 to 29.

Table 22.9 Platreef 2025 FS Operating Costs and Revenues

Item	LOM Total ($M)	Milled ($/t)		
		Stage 1 2025 - 2027 Average	Stage 2 2028 - 2059 Average	LOM 2025-2059 Average
Gross Sales Revenue	**27,590**	**216.43**	**212.59**	**212.65**
Less: Realisation Costs				
Transport	117	0.89	0.90	0.90
Treatment and Refining	5,544	44.38	42.70	42.73
Au, Pd & Pt Streaming Agreements	1,793	17.56	13.76	13.82
Royalties	1,170	0.86	9.15	9.02
Total Realisation Costs	**8,624**	**63.70**	**66.52**	**66.47**
Net Sales Revenue	**18,965**	**152.74**	**146.07**	**146.18**
Site Operating Costs				
Mining	4,922	62.84	37.53	37.94
Processing and Tailings	1,789	18.43	13.71	13.79
Infrastructure	363	6.28	2.74	2.80
General and Administration	722	26.40	5.23	5.57
Total	**7,796**	**113.95**	**59.22**	**60.09**
Operating Margin	**11,170**	**38.78**	**86.86**	**86.09**
Operating Margin (%)	**40%**	**18%**	**41%**	**40%**

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 2025 FS Total Project Capital Cost

Description	Initial Capital ($M)	Sustaining Capital ($M)	Total Capital Over LOM ($M)
Mining			
Mining	581	1,364	1,944
Capitalised Operating Costs	75	0	75
Subtotal	**655**	**1,364**	**2,019**
Concentrator and Infrastructure	**373**	**104**	**477**
Other Costs			
Owner's Cost	95	34	129
Closure Cost	0	10	10
Capitalised Operating Costs	20	0	20
Subtotal	**116**	**44**	**159**
Capex Before Contingency	**1,144**	**1,511**	**2,655**
Contingency	99	78	177
Capex After Contingency	**1,243**	**1,590**	**2,832**

1. Initial capital reflects the capital costs from 1 January 2025, to achieve initial production from the 770 ktpa Phase 1 concentrator, followed by expansion capital to achieve nameplate Phase 2 production of the 4.1 Mtpa FS.
2. Remaining capital cost for Phase 1 is $70M.
3. 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 1 marks the beginning of production, with the plant operating only during a portion of the year, commencing in Q4'25. The details of mine and process production and the Project cash flow are shown in Table 22.11 to Table 22.16. Year 1 is 2025.

Figure 22.8 Platreef 2025 FS Cumulative Cash Flow After-Tax

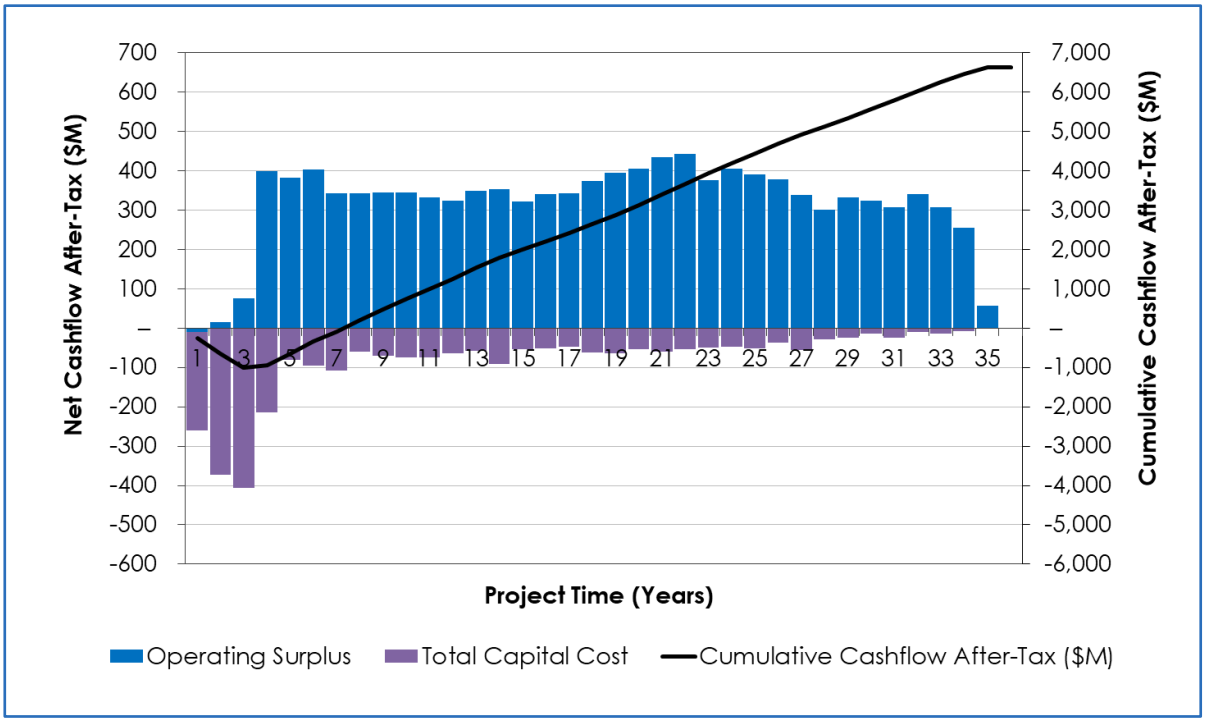


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Table 22.11 Platreef 2025 FS Mine Production

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
Total Ore Mined	kt	129,742	172	704	2,104	3,218	20,330	42,443	60,770
NSR	$/t	141.11	167.06	156.89	141.28	143.88	135.50	140.05	143.33
Platinum	g/t	1.88	2.12	1.92	1.80	1.83	1.76	1.82	1.96
Palladium	g/t	1.93	2.06	1.93	1.85	1.89	1.90	1.93	1.94
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.27	0.28	0.30
Rhodium	g/t	0.13	0.13	0.12	0.11	0.12	0.12	0.13	0.14
Copper	%	0.16	0.19	0.19	0.18	0.18	0.16	0.16	0.16
Nickel	%	0.33	0.37	0.38	0.35	0.36	0.33	0.34	0.32
Sulfur	%	0.80	0.95	1.02	0.93	0.91	0.83	0.82	0.77
3PE+Au	g/t	4.22	4.65	4.28	4.07	4.13	4.04	4.15	4.34
Ore Development	kt	21,921	172	250	866	764	5,255	8,143	6,471
NSR	$/t	110.32	167.06	139.85	121.53	105.12	100.88	111.54	108.18
Platinum	g/t	1.64	2.12	1.81	1.65	1.54	1.43	1.66	1.72
Palladium	g/t	1.73	2.06	1.78	1.72	1.66	1.56	1.76	1.75
Gold	g/t	0.26	0.33	0.31	0.27	0.24	0.23	0.25	0.26
Rhodium	g/t	0.11	0.13	0.11	0.11	0.10	0.10	0.12	0.12
Copper	%	0.15	0.19	0.19	0.17	0.16	0.15	0.15	0.15
Nickel	%	0.31	0.37	0.37	0.33	0.31	0.30	0.32	0.31
Sulfur	%	0.78	0.95	0.97	0.89	0.79	0.74	0.78	0.75
3PE+Au	g/t	3.74	4.65	4.01	3.75	3.55	3.31	3.79	3.85
Longhole	kt	88,242	–	454	1,238	2,455	14,832	30,592	38,672

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
NSR	$/t	141.00	–	166.29	155.10	155.94	147.59	142.43	136.62
Platinum	g/t	1.85	–	1.98	1.91	1.92	1.87	1.80	1.88
Palladium	g/t	1.92	–	2.01	1.93	1.96	2.01	1.93	1.89
Gold	g/t	0.28	–	0.30	0.32	0.31	0.29	0.28	0.28
Rhodium	g/t	0.13	–	0.13	0.12	0.12	0.13	0.13	0.13
Copper	%	0.16	–	0.20	0.19	0.18	0.17	0.16	0.15
Nickel	%	0.32	–	0.39	0.37	0.37	0.34	0.34	0.31
Sulfur	%	0.79	–	1.04	0.96	0.95	0.85	0.81	0.73
3PE+Au	g/t	4.18	–	4.43	4.29	4.31	4.29	4.14	4.18
Drift and Fill	kt	19,580	–	–	–	–	244	3,708	15,628
NSR	$/t	176.10	–	–	–	–	285.99	182.97	174.49
Platinum	g/t	2.26	–	–	–	–	3.41	2.28	2.25
Palladium	g/t	2.20	–	–	–	–	3.72	2.32	2.17
Gold	g/t	0.37	–	–	–	–	0.53	0.33	0.38
Rhodium	g/t	0.15	–	–	–	–	0.22	0.16	0.15
Copper	%	0.18	–	–	–	–	0.29	0.19	0.18
Nickel	%	0.36	–	–	–	–	0.63	0.37	0.36
Sulfur	%	0.90	–	–	–	–	1.31	0.93	0.89
3PE+Au	g/t	4.97	–	–	–	–	7.88	5.09	4.95

Note: NSR is reported for BDT30. BDT30 metal prices were used in the Mineral Reserve estimate are as follows: $1,150/oz platinum, $1,200/oz palladium, $1,700/oz gold, $4,500/oz rhodium, $8.80/lb nickel and $3.80/lb copper. Metal-price assumptions used for the Platreef 2025 FS economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.

Table 22.12 Platreef 2025 FS Process Production Plant Feed

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
Concentrator Feed	kt	129,742	84	715	1,265	3,988	20,350	40,700	62,641
Platinum	g/t	1.88	2.16	1.93	1.79	1.84	1.76	1.85	1.93
Palladium	g/t	1.93	2.11	1.93	1.82	1.89	1.90	1.97	1.92
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.27	0.28	0.30
Rhodium	g/t	0.13	0.14	0.12	0.11	0.12	0.12	0.13	0.13
Copper	%	0.16	0.19	0.19	0.18	0.18	0.16	0.17	0.16
Nickel	%	0.33	0.38	0.38	0.35	0.36	0.33	0.34	0.32
Sulfur	%	0.80	0.97	1.00	0.93	0.92	0.83	0.83	0.77
3PE+Au	g/t	4.22	4.74	4.28	4.02	4.14	4.04	4.23	4.29

Note: NSR is reported for BDT30. BDT30 metal prices were used in the Mineral Reserve estimate are as follows: $1,150/oz platinum, $1,200/oz palladium, $1,700/oz gold, $4,500/oz rhodium, $8.80/lb nickel and $3.80/lb copper. Metal-price assumptions used for the Platreef 2025 FS economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.

Table 22.13 Platreef 2025 FS Concentrator Recoveries

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
Concentrate Produced	kt	5,523	5	31	50	164	817	1,724	2,731
Concentrator Recoveries									
Platinum	%	86.88	88.88	86.79	85.07	85.96	85.98	86.69	87.07
Palladium	%	86.29	88.65	86.24	84.30	85.34	85.23	86.12	86.45
Gold	%	78.33	78.80	78.16	76.90	77.65	77.78	78.25	78.47
Rhodium	%	79.78	82.88	79.76	77.56	78.66	78.50	79.52	79.94
Copper	%	87.33	88.48	90.20	88.23	88.85	87.44	87.95	86.58
Nickel	%	70.81	74.03	75.37	72.42	73.39	70.67	72.04	69.53
Sulfur	%	70.40	72.72	75.11	72.84	73.05	71.00	70.99	69.34
3PE+Au	%	85.81	87.89	85.73	83.90	84.88	84.85	85.64	85.97
Mass Pull	%	4.26	6.54	4.37	3.98	4.12	4.01	4.24	4.36

Table 22.14 Platreef 2025 FS Concentrate Production

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
Concentrate Produced	kt	5,523	5	31	50	164	817	1,724	2,731
Concentrator Grades									
Platinum	g/t	38.28	29.39	38.25	38.11	38.28	37.67	37.95	38.69
Palladium	g/t	39.11	28.55	38.08	38.56	39.15	40.27	40.03	38.22
Gold	g/t	5.33	4.01	5.45	5.79	5.55	5.27	5.20	5.41
Rhodium	g/t	2.43	1.75	2.23	2.19	2.23	2.34	2.44	2.47
Copper	%	3.31	2.58	3.95	3.92	3.85	3.54	3.47	3.09
Nickel	%	5.47	4.29	6.47	6.39	6.35	5.78	5.80	5.08
Sulfur	%	13.28	10.75	17.19	16.95	16.23	14.62	13.84	12.24
3PE+Au	g/t	85.15	63.70	84.01	84.65	85.21	85.55	85.62	84.79

Table 22.15 Platreef 2025 FS Metal Production

Item	Year Number	Total	1	2	3	4	5	10	20
	Year To						9	19	LOM
Contained Metal									
Platinum	koz	6,797	5	38	62	202	989	2,103	3,397
Palladium	koz	6,944	5	38	62	207	1,057	2,218	3,356
Gold	koz	947	1	5	9	29	138	288	475
Rhodium	koz	432	0.3	2	4	12	62	135	217
Copper	klb	402,911	310	2,719	4,354	13,934	63,775	131,804	186,015
Nickel	klb	665,535	517	4,462	7,097	23,022	104,092	220,306	306,041
Sulfur	klb	1,617,135	1,295	11,848	18,833	58,811	263,128	525,972	737,249
3PE+Au	koz	15,120	11	84	137	450	2,246	4,745	7,446

Table 22.16 Platreef 2025 FS Cash Flow

Cash Flow Statement ($M)		Year										
Year Number	**Total**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**20**
Year To											**19**	**LOM**
Gross Revenue	27,590	20	163	263	854	859	869	803	808	806	8,767	13,378
Realisation Costs	8,624	6	48	78	244	240	243	229	233	228	2,827	4,249
Net Sales Revenue	**18,965**	**14**	**115**	**186**	**609**	**619**	**626**	**574**	**576**	**578**	**5,939**	**9,129**
Less: Site Operating Costs												
Mining	4,922	13	57	60	123	147	132	141	142	144	1,579	2,384
Concentrator	1,789	2	15	21	55	56	56	56	56	56	558	860
Infrastructure	363	1	5	6	10	10	11	11	12	11	114	171
General and Administration	722	7	23	24	23	23	23	23	23	23	211	317
Subtotal	**7,796**	**24**	**100**	**111**	**210**	**237**	**222**	**231**	**233**	**233**	**2,462**	**3,732**
Operating Surplus/(Deficit)	**11,170**	**-10**	**15**	**75**	**399**	**383**	**403**	**343**	**343**	**344**	**3,477**	**5,397**
Capital Costs												
Initial Capital Expenditure	1,243	251	372	405	215	–	–	–	–	–	–	–
Sustaining Capital	1,589	–	–	–	–	81	95	108	60	71	638	536
Working Capital	–	3	27	22	127	-0	4	-16	1	-1	31	-197
Subtotal	2,832	254	398	427	342	81	98	92	62	70	669	339
Net Cashflow Before Tax	**8,338**	**-263**	**-383**	**-352**	**57**	**302**	**305**	**251**	**282**	**274**	**2,808**	**5,058**
Depreciation	-2,832	-251	-372	-405	-215	-81	-95	-108	-60	-71	-638	-536
Income Tax Expense	1,710	–	–	–	–	–	–	–	–	–	397	1,312
Net Cashflow After-Tax	**6,628**	**-263**	**-383**	**-352**	**57**	**302**	**305**	**251**	**282**	**274**	**2,411**	**3,745**

22.6 Price Sensitivity Analysis

A price sensitivity analysis on the long-term metal prices and the impact on NPV8% were performed on the financial model with results 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**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	106	300	477	867	922	1,084	1,246
4.50	280	458	629	1,010	1,065	1,228	1,389
6.00	439	611	777	1,155	1,209	1,371	1,530
8.50	690	855	1,017	1,393	1,447	1,606	1,764
10.00	837	998	1,161	1,535	1,588	1,746	1,905
12.50	1,078	1,240	1,398	1,769	1,822	1,980	2,138
15.00	1,315	1,474	1,632	2,002	2,054	2,214	2,372

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**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	-92%	-78%	-66%	-38%	-34%	-22%	-11%
4.50	-80%	-67%	-55%	-28%	-24%	-12%	-0%
6.00	-69%	-56%	-44%	-17%	-13%	-2%	10%
8.50	-50%	-39%	-27%	–	4%	15%	27%
10.00	-40%	-28%	-17%	10%	14%	25%	37%
12.50	-23%	-11%	0%	27%	31%	42%	53%
15.00	-6%	6%	17%	44%	47%	59%	70%

Figure 22.9 Platinum and Nickel Price Sensitivity

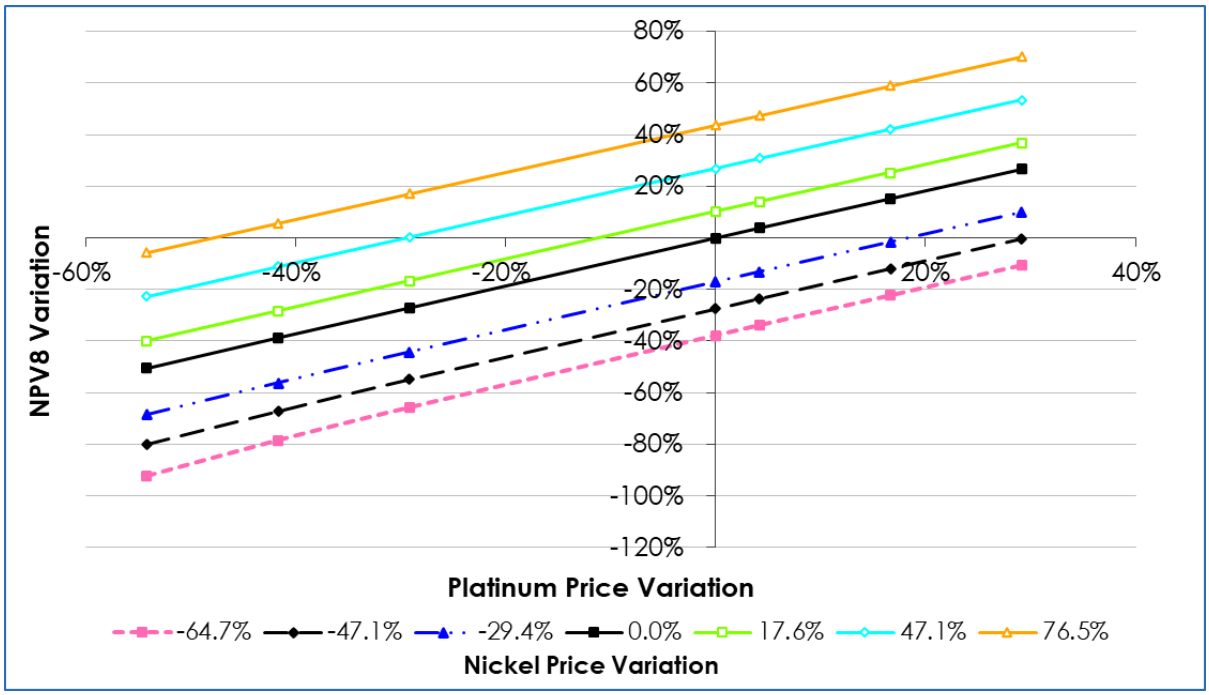

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Figure 22.10 Nickel and Platinum Price Sensitivity

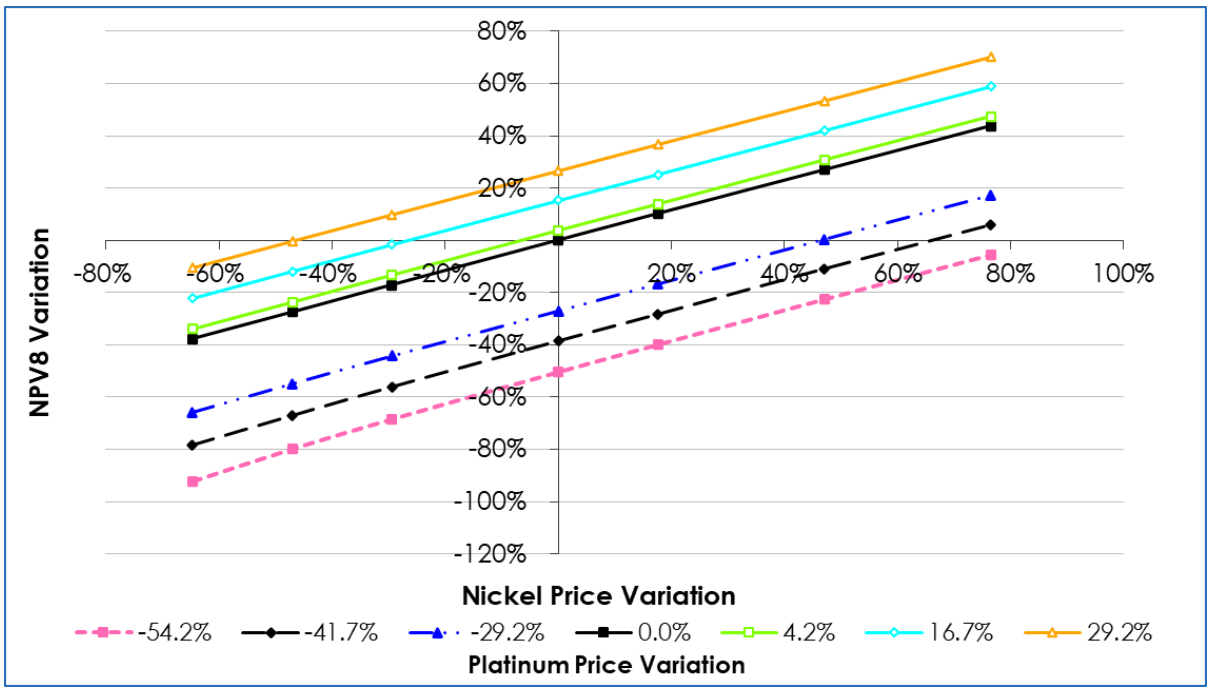

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Table 22.19 After-Tax NPV8% Platinum and Palladium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Palladium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
800	293	471	642	1,023	1,079	1,241	1,400
1,000	539	706	871	1,250	1,303	1,461	1,621
1,130	690	855	1,017	1,393	1,447	1,606	1,764
1,450	1,053	1,215	1,375	1,746	1,800	1,957	2,115
1,700	1,334	1,493	1,651	2,021	2,073	2,233	2,390
2,000	1,664	1,824	1,981	2,352	2,403	2,562	2,719
2,500	2,215	2,372	2,530	2,895	2,948	3,106	3,264

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**	**1,200**	**1,250**	**1,400**	**1,550**
800	-79%	-66%	-54%	-27%	-23%	-11%	0%
1,000	-61%	-49%	-38%	-10%	-6%	5%	16%
1,130	-50%	-39%	-27%	–	4%	15%	27%
1,450	-24%	-13%	-1%	25%	29%	40%	52%
1,700	-4%	7%	19%	45%	49%	60%	72%
2,000	19%	31%	42%	69%	72%	84%	95%
2,500	59%	70%	82%	108%	112%	123%	134%

Figure 22.11 Platinum and Palladium Price Sensitivity

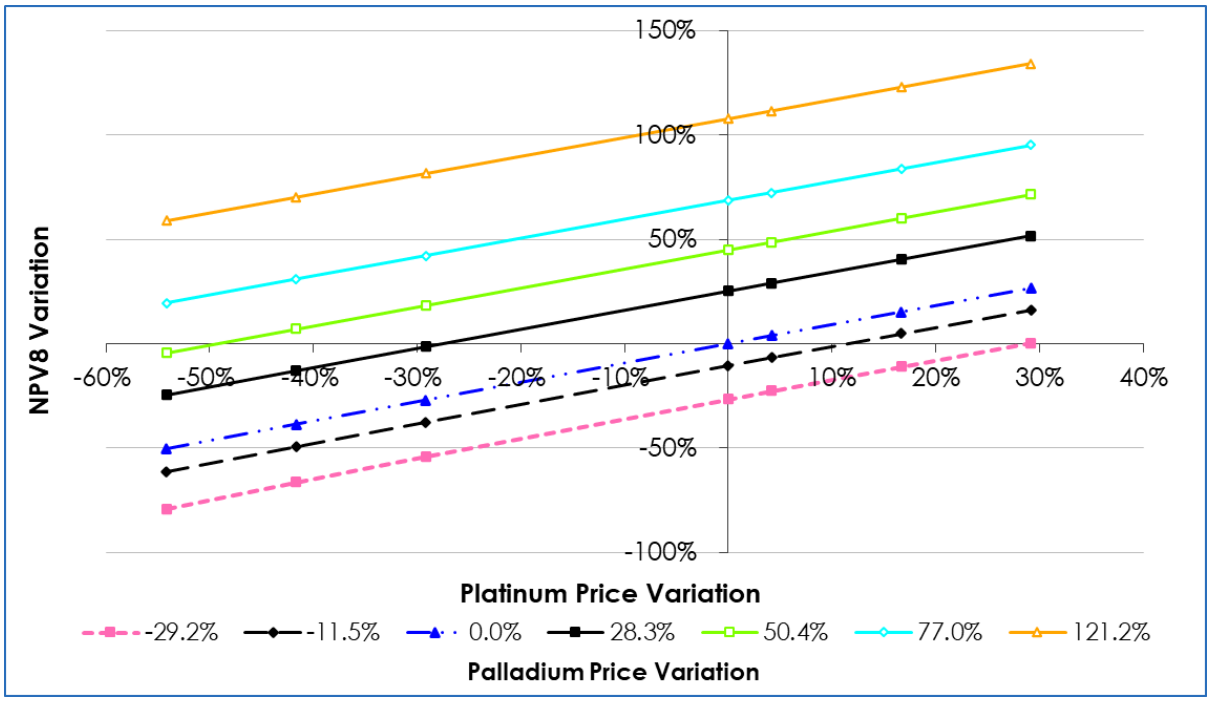

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Figure 22.12 Palladium and Platinum Price Sensitivity

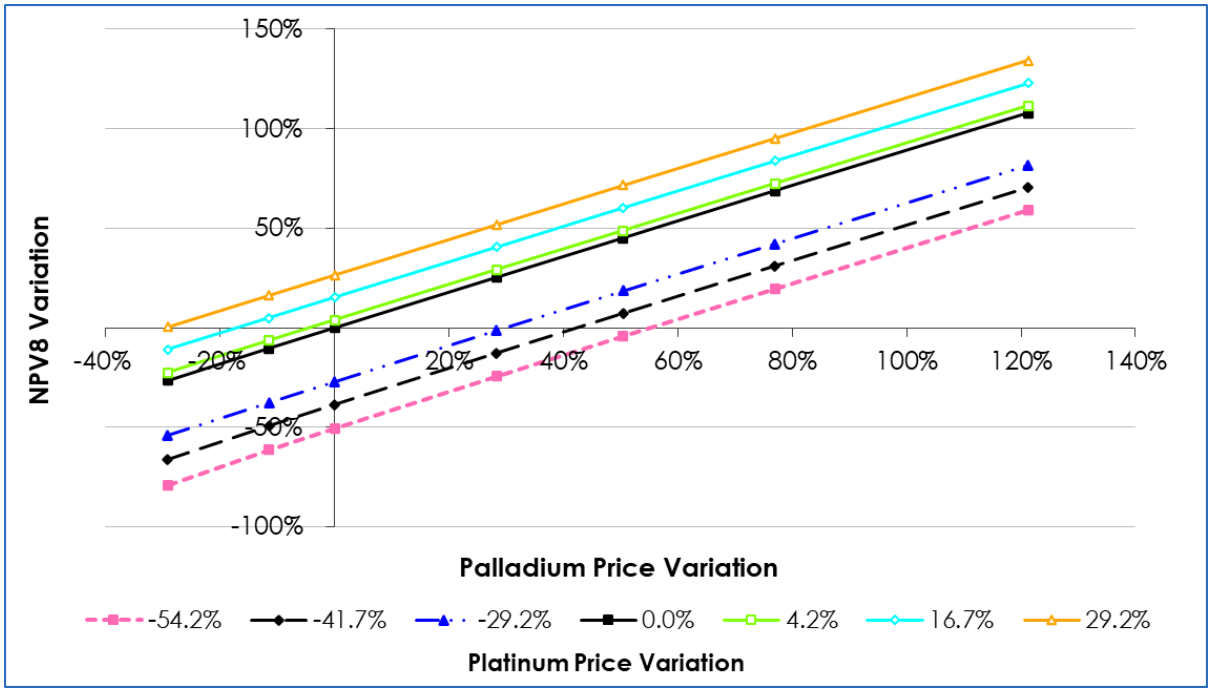

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Table 22.21 After-Tax NPV8% Platinum and Gold Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Gold Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,600	688	854	1,016	1,393	1,446	1,605	1,763
1,800	689	854	1,016	1,393	1,446	1,605	1,763
2,000	690	855	1,016	1,393	1,447	1,605	1,764
2,170	690	855	1,017	1,393	1,447	1,606	1,764
2,500	691	856	1,017	1,394	1,448	1,606	1,765
2,700	692	857	1,017	1,394	1,448	1,607	1,765
2,900	693	857	1,018	1,395	1,449	1,607	1,766

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**	**1,200**	**1,250**	**1,400**	**1,550**
1,600	-51%	-39%	-27%	-0%	4%	15%	27%
1,800	-51%	-39%	-27%	-0%	4%	15%	27%
2,000	-51%	-39%	-27%	-0%	4%	15%	27%
2,170	-50%	-39%	-27%	–	4%	15%	27%
2,500	-50%	-39%	-27%	0%	4%	15%	27%
2,700	-50%	-39%	-27%	0%	4%	15%	27%
2,900	-50%	-38%	-27%	0%	4%	15%	27%

Figure 22.13 Platinum and Gold Price Sensitivity

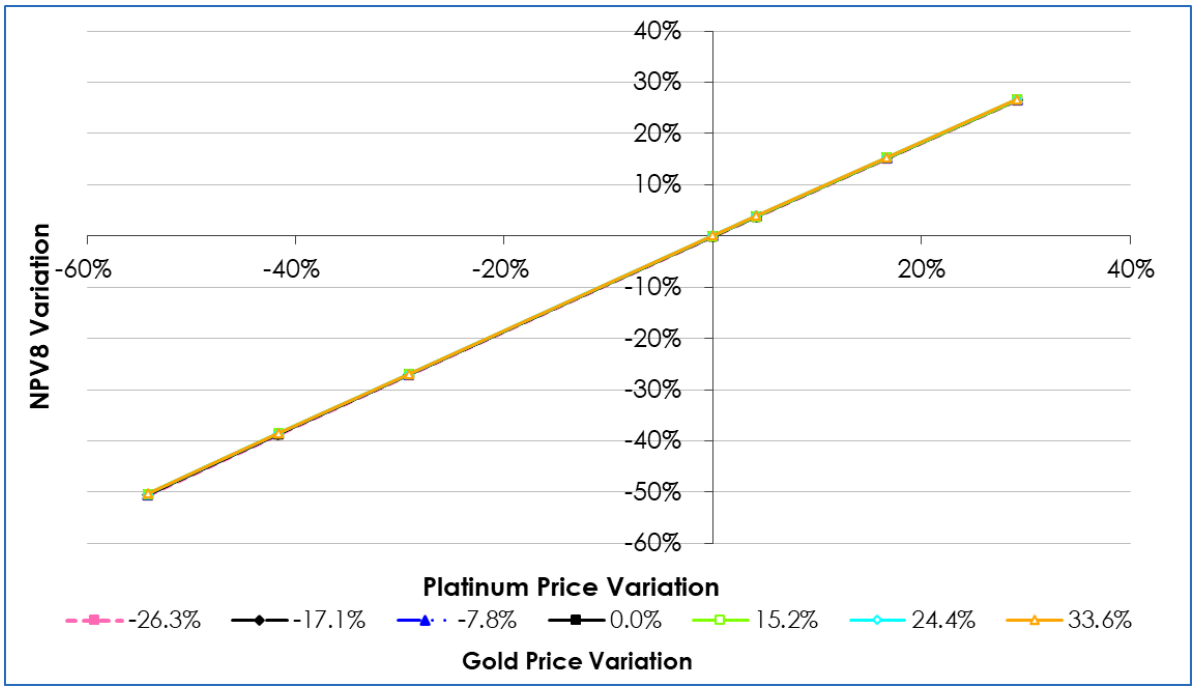

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Figure 22.14 Gold and Platinum Price Sensitivity

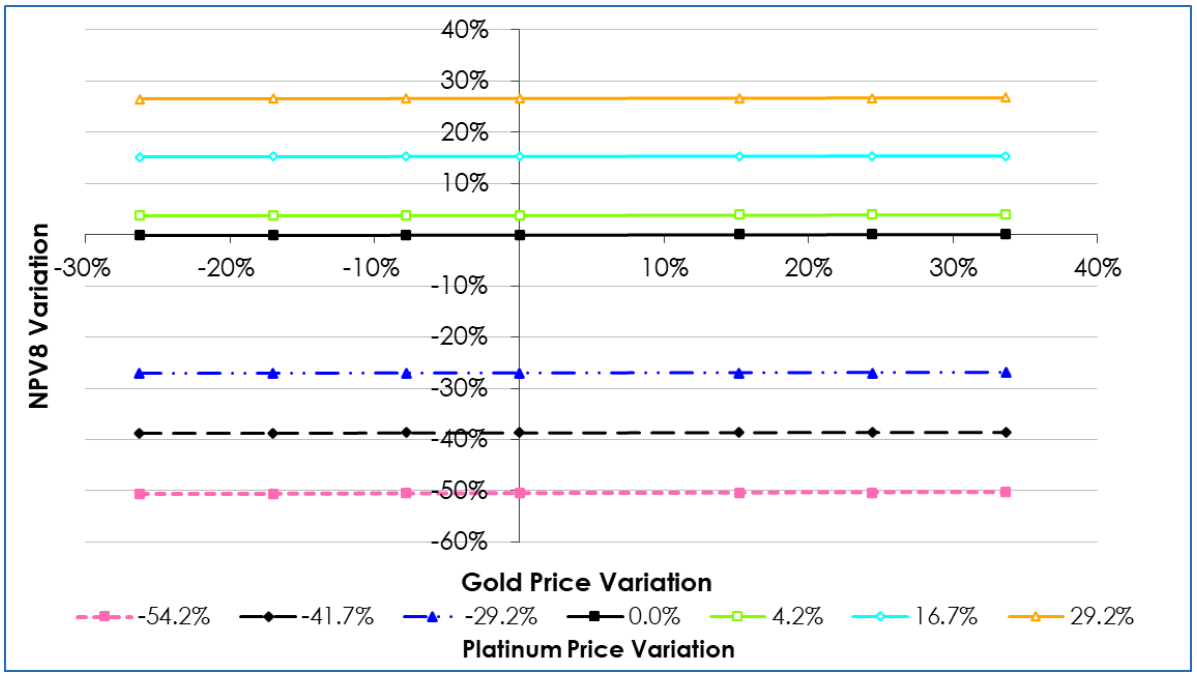

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Table 22.23 After-Tax NPV8% Platinum and Rhodium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Rhodium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,000	395	568	736	1,116	1,169	1,331	1,490
2,000	471	642	806	1,185	1,240	1,400	1,560
3,500	582	750	913	1,291	1,344	1,503	1,661
5,000	690	855	1,017	1,393	1,447	1,606	1,764
10,000	1,042	1,205	1,366	1,736	1,790	1,948	2,106
15,000	1,391	1,551	1,709	2,078	2,131	2,291	2,448
20,000	1,733	1,894	2,052	2,421	2,473	2,630	2,789

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**	**1,200**	**1,250**	**1,400**	**1,550**
1,000	-72%	-59%	-47%	-20%	-16%	-4%	7%
2,000	-66%	-54%	-42%	-15%	-11%	0%	12%
3,500	-58%	-46%	-34%	-7%	-4%	8%	19%
5,000	-50%	-39%	-27%	–	4%	15%	27%
10,000	-25%	-14%	-2%	25%	28%	40%	51%
15,000	-0%	11%	23%	49%	53%	64%	76%
20,000	24%	36%	47%	74%	78%	89%	100%

Figure 22.15 Platinum and Rhodium Price Sensitivity

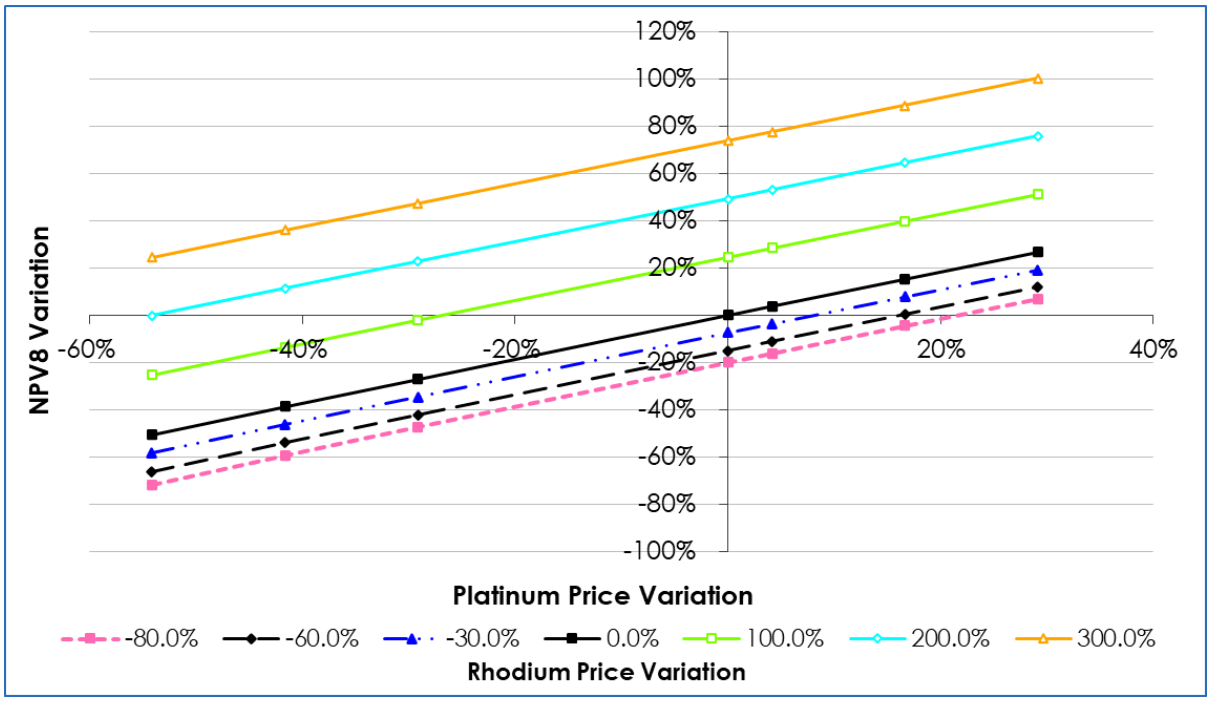

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Figure 22.16 Rhodium and Platinum Price Sensitivity

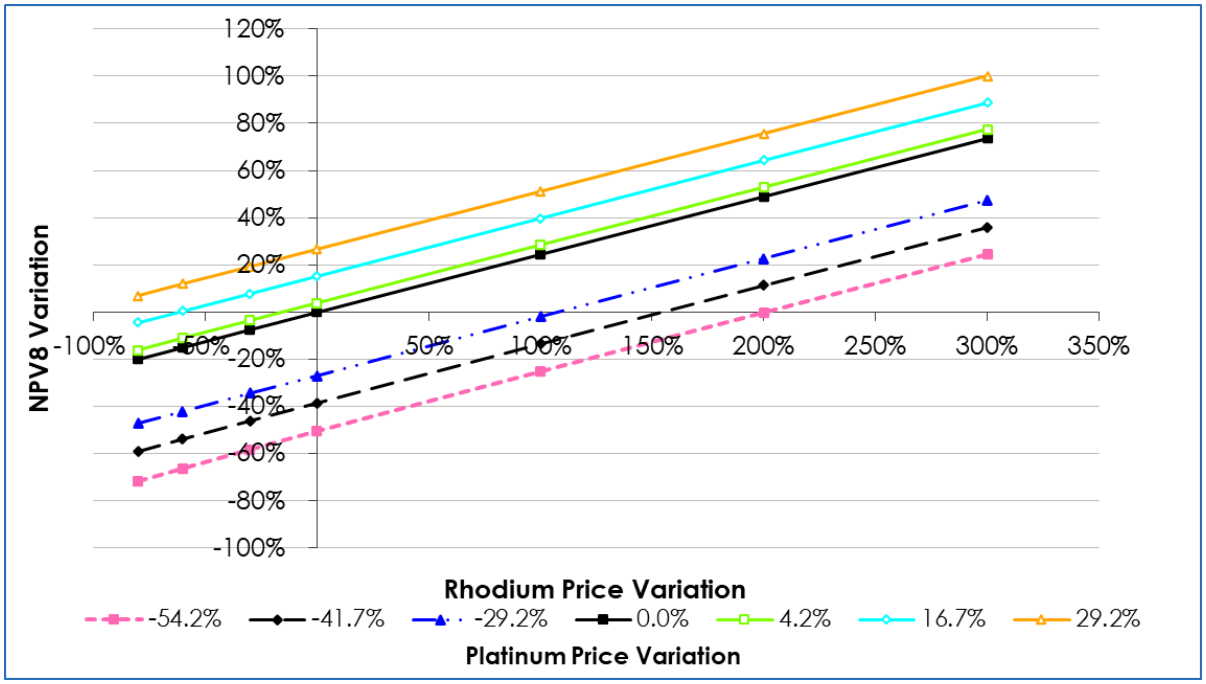

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Table 22.25 After-Tax NPV8% Platinum and Copper Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Copper Price - $/lb	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	615	781	943	1,322	1,374	1,534	1,692
3.50	646	810	971	1,350	1,403	1,564	1,720
4.00	675	840	1,001	1,379	1,432	1,592	1,749
4.25	690	855	1,017	1,393	1,447	1,606	1,764
4.50	705	870	1,032	1,408	1,460	1,620	1,779
5.00	736	900	1,062	1,437	1,490	1,648	1,808
5.50	766	929	1,092	1,465	1,519	1,678	1,837

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**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	-56%	-44%	-32%	-5%	-1%	10%	21%
3.50	-54%	-42%	-30%	-3%	1%	12%	23%
4.00	-52%	-40%	-28%	-1%	3%	14%	26%
4.25	-50%	-39%	-27%	–	4%	15%	27%
4.50	-49%	-38%	-26%	1%	5%	16%	28%
5.00	-47%	-35%	-24%	3%	7%	18%	30%
5.50	-45%	-33%	-22%	5%	9%	20%	32%

Figure 22.17 Platinum and Copper Price Sensitivity

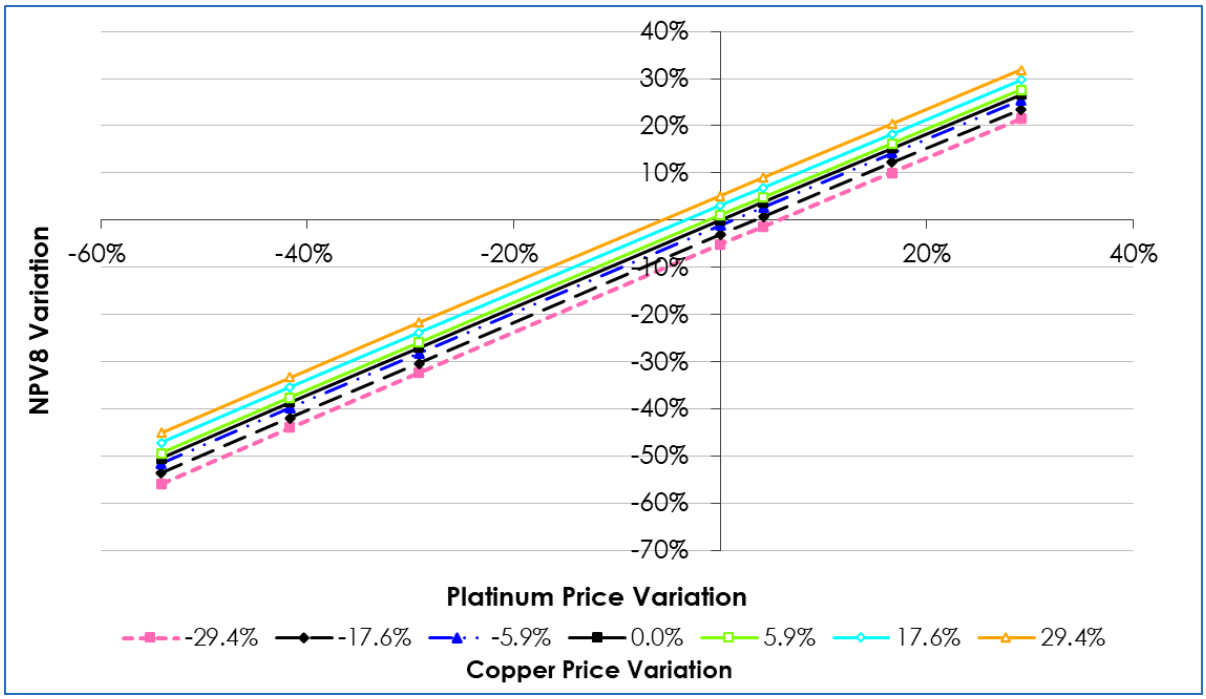

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Figure 22.18 Copper and Platinum Price Sensitivity

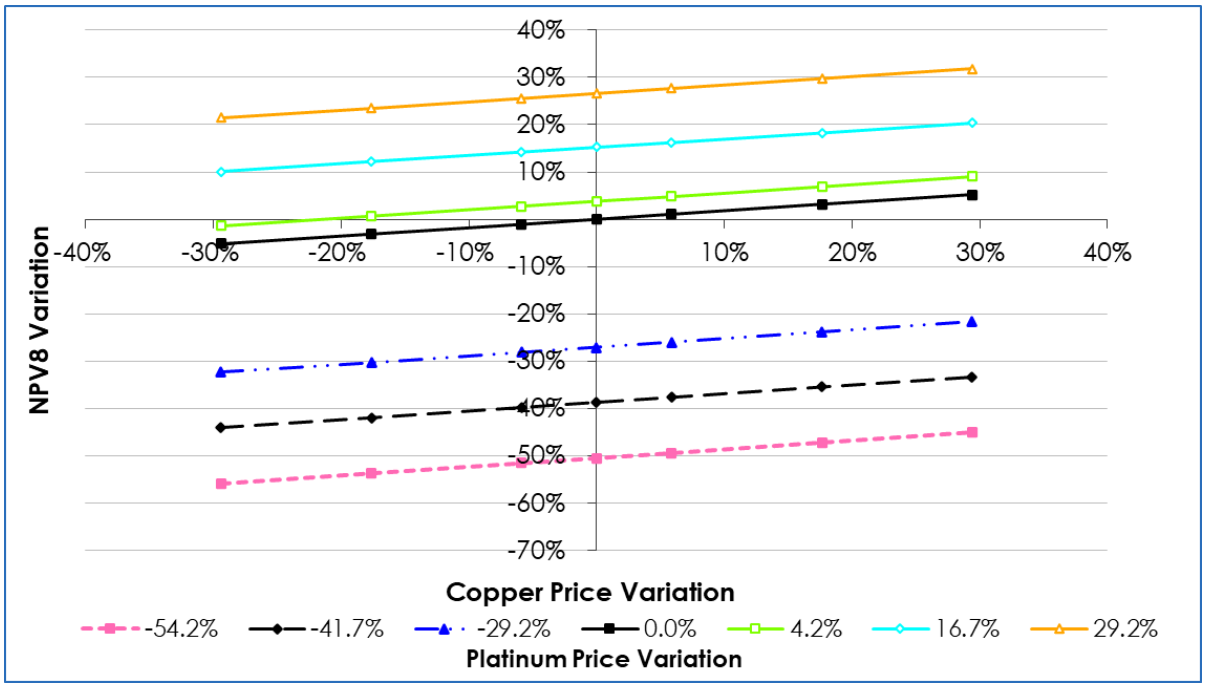

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Table 22.27 After-Tax NPV8% Nickel and Copper Price Sensitivity

After-Tax NPV8% ($M)	Nickel Price ($/lb)						
Copper Price - $/lb	**3.00**	**4.50**	**6.00**	**8.50**	**10.00**	**12.50**	**15.00**
3.00	793	938	1,082	1,322	1,462	1,695	1,930
3.50	822	967	1,112	1,350	1,491	1,725	1,959
4.00	852	995	1,141	1,379	1,520	1,754	1,988
4.25	867	1,010	1,155	1,393	1,535	1,769	2,002
4.50	882	1,025	1,170	1,408	1,550	1,783	2,017
5.00	912	1,055	1,200	1,437	1,578	1,812	2,044
5.50	942	1,086	1,230	1,465	1,606	1,842	2,073

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**	**8.50**	**10.00**	**12.50**	**15.00**
3.00	-43%	-33%	-22%	-5%	5%	22%	39%
3.50	-41%	-31%	-20%	-3%	7%	24%	41%
4.00	-39%	-29%	-18%	-1%	9%	26%	43%
4.25	-38%	-28%	-17%	–	10%	27%	44%
4.50	-37%	-26%	-16%	1%	11%	28%	45%
5.00	-35%	-24%	-14%	3%	13%	30%	47%
5.50	-32%	-22%	-12%	5%	15%	32%	49%

Figure 22.19 Nickel and Copper Price Sensitivity

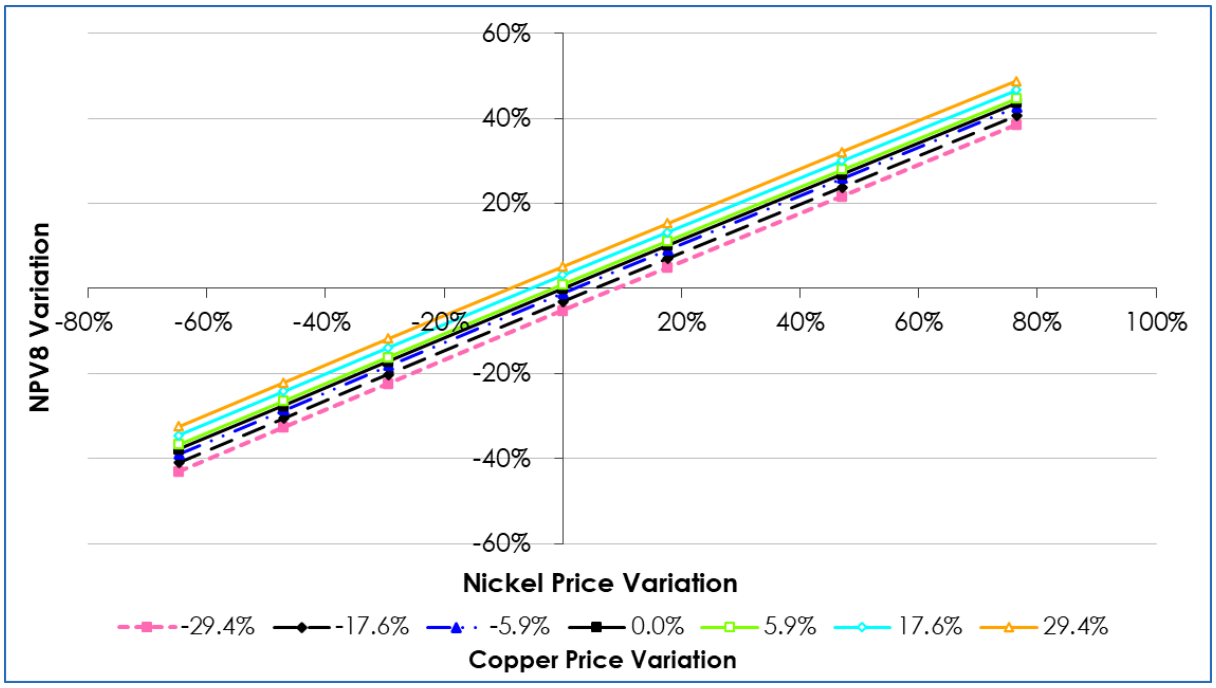

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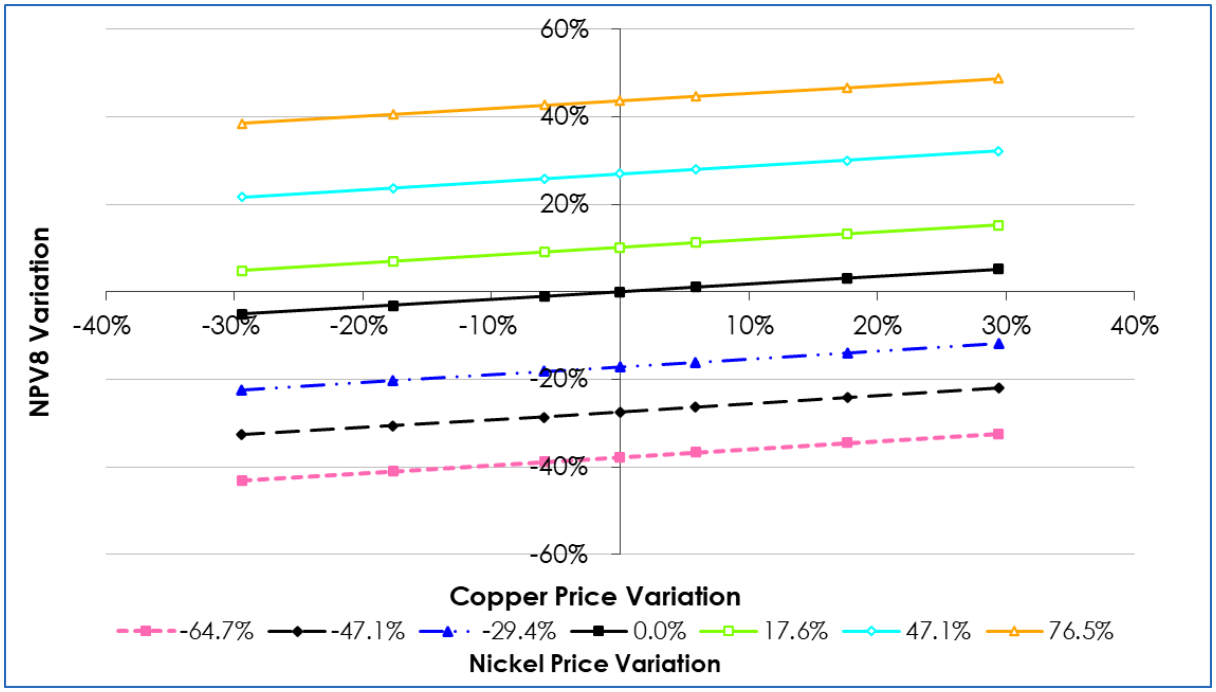
Figure 22.20 Copper and Nickel Price Sensitivity



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22.7 Cost Sensitivity Analysis

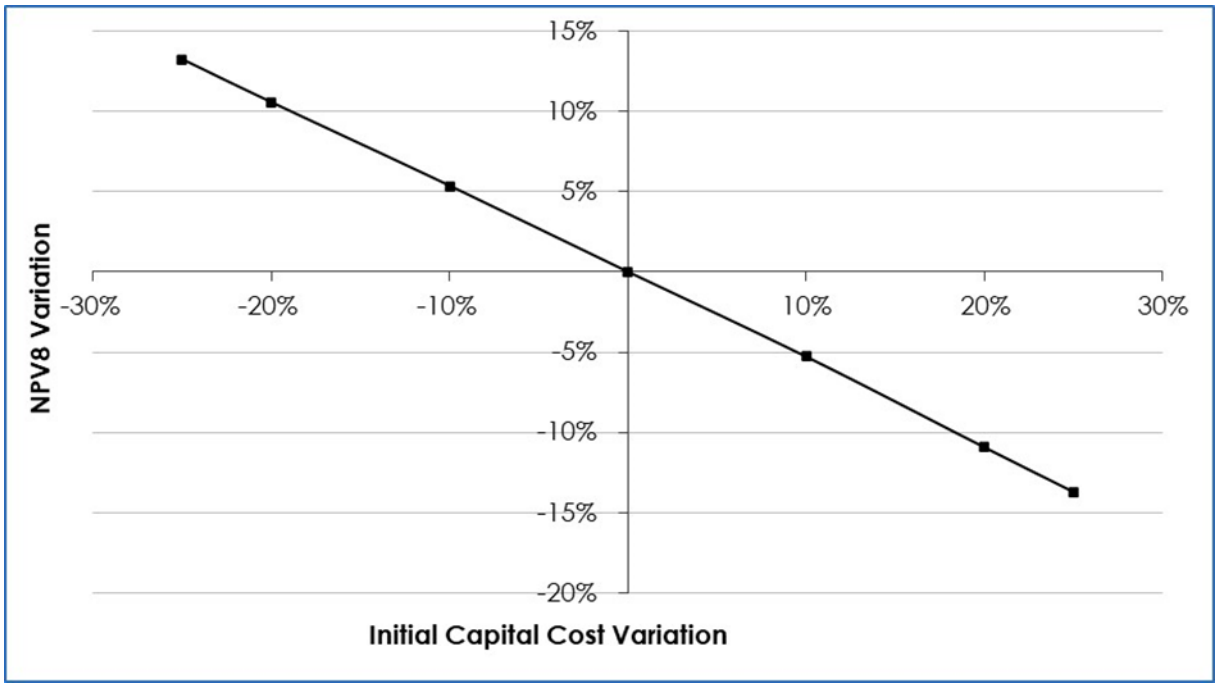
Cost sensitivity analyses of the After-Tax NPV8% were examined for the initial capital, expansion capital, sustaining capital, and operating costs.

The initial capital cost sensitivity analysis shows the After-Tax NPV8% of $577M at a 25% decrease in the initial capital cost and the After-Tax NPV8% of $1,203M at a 25% increase in the pre-production capital cost. The results of the initial 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% ($M)	1,577	1,540	1,468	1,393	1,393	1,241	1,203
Difference ($M)	184	147	75	–	-74	-152	-191
Difference (%)	13%	11%	5%	0%	-5%	-11%	-14%

Figure 22.21 Initial Capital Cost Sensitivity



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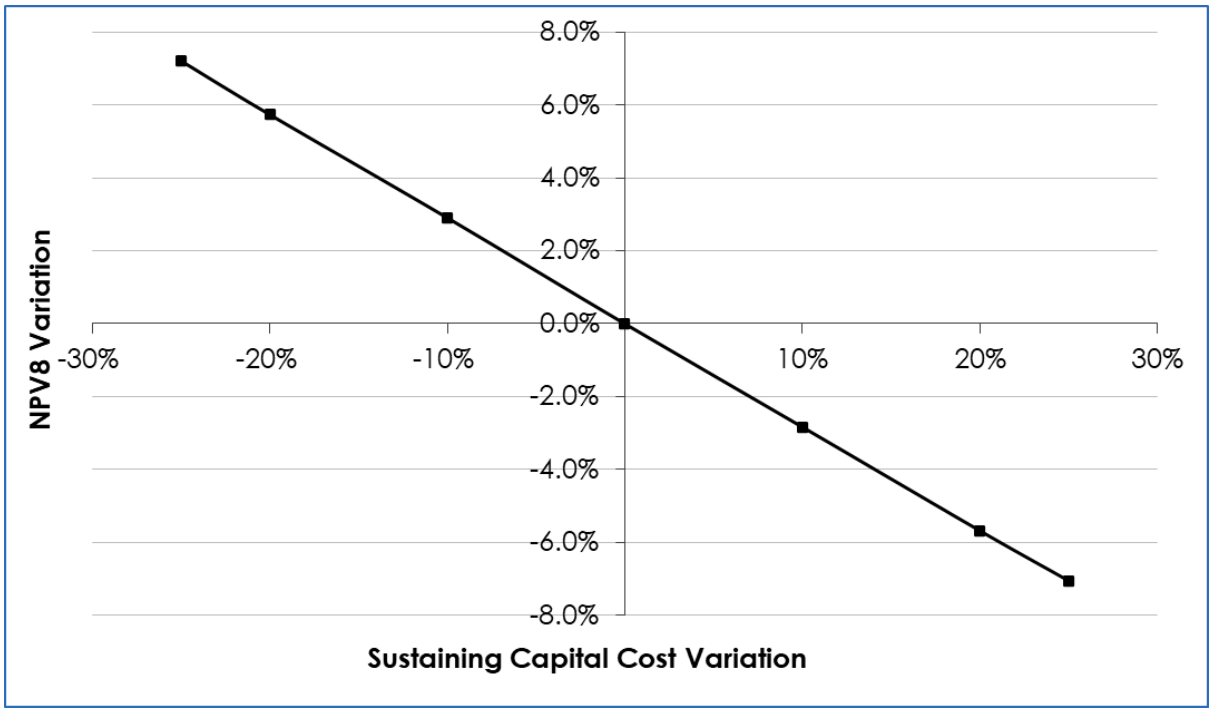
The sustaining capital cost sensitivity analysis shows the After-Tax NPV8% of $1,494M at a 25% decrease in the sustaining capital cost and the After-Tax NPV8% of $1,295M at a 25% increase in the sustaining capital cost. The results of the sustaining capital cost sensitivity analysis are shown in Table 22.30 and Figure 22.22.

Table 22.30 Sustaining Capital Cost Sensitivity

Sustaining Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	1,494	1,473	1,434	1,393	1,354	1,314	1,295
Difference ($M)	100	80	40	–	-39	-79	-98
Difference (%)	7.2%	5.7%	2.9%	0.0%	-2.8%	-5.7%	-7.0%

Figure 22.22 Sustaining Capital Cost Sensitivity



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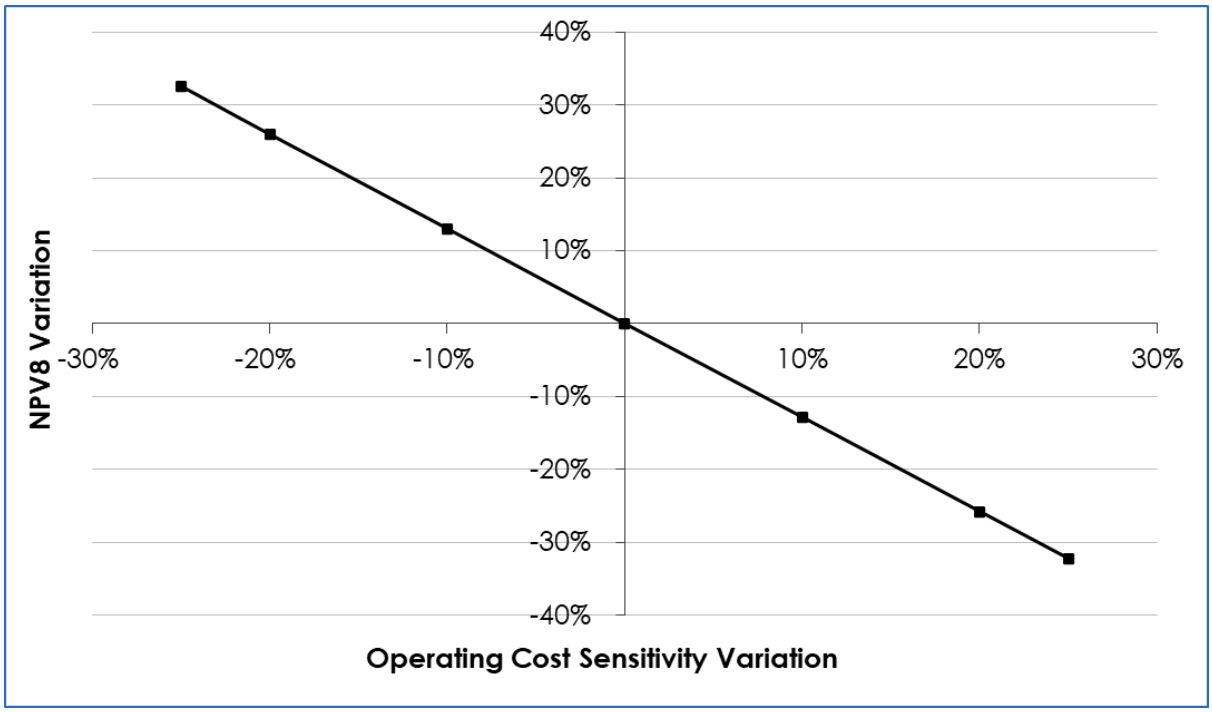
The operating cost sensitivity analysis shows the After-Tax NPV8% of $1,847 at a 25% decrease in the operating cost and the After-Tax NPV8% of $944M at a 25% increase in the operating cost. The results of the operating cost sensitivity analysis are shown in Table 22.31 and Figure 22.23.

Table 22.31 Operating Cost Sensitivity

Sustaining Capital Cost	–25%	–20%	–10%	0%	10%	20%	25%
After-Tax NPV8% ($M)	1,847	1,755	1,575	1,393	1,215	1,034	944
Difference ($M)	454	362	182	–	-179	-359	-449
Difference (%)	33%	26%	13%	0%	-13%	-26%	-32%

Figure 22.23 Operating Cost Sensitivity



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22.8 Platreef 2025 FS Comparison to Platreef 2022 FS

A comparison of the Platreef 2025 FS and Platreef 2022 FS financial models was carried out. The Platreef 2025 FS estimates of cash flows were prepared on a real basis as at 1 January 2025 to calculate NPV and the Platreef 2022 FS estimates cash flows were prepared on a real basis as at 1 January 2022. The After-Tax financial results in 2025 FS and 2022 FS are shown in Table 22.32.

Table 22.32 After-Tax Financial Results Comparison

	Discount Rate	2025 FS	2022 FS
Net Present Value ($M)	Undiscounted	6,628	8,543
	5.0%	2,450	3,098
	8.0%	1,393	1,690
	10.0%	950	1,104
	12.0%	632	692
	15.0%	306	283
	20.0%	-11	-83
Internal Rate of Return	–	19.77%	18.48%
Project Payback Period (Years)	–	7.8	7.9

The 2025 FS and 2022 FS metal prices are based on a review of consensus price forecasts from a financial institutions and similar studies. A comparison of the long-term base and precious metal prices in 2025 FS and 2022 FS is presented in Table 22.33.

Table 22.33 Long Term Metal Prices Comparison

Parameter	Unit	2025 FS	2022 FS
Platinum	$/oz	1,200	1,100
Palladium	$/oz	1,130	1,450
Gold	$/oz	2,170	1,600
Rhodium	$/oz	5,000	5,000
Copper	$/lb	4.25	3.50
Nickel	$/lb	8.50	8.00

The key production summary comparison is shown in Table 22.34. A comparison of the Total Capital Costs in 2025 FS and 2022 FS are shown in Table 22.35.

Table 22.34 Production Summary Comparison

Item	Unit	2025 FS	2022 FS
Mined and Processed	Mt	130	125
Platinum	g/t	1.88	1.94
Palladium	g/t	1.93	1.99
Gold	g/t	0.29	0.30
Rhodium	g/t	0.13	0.13
3PE+Au	g/t	4.22	4.37
Copper	% Cu	0.16	0.16
Nickel	% Ni	0.33	0.34
Concentrator Recoveries			
Platinum	%	86.88	87.20
Palladium	%	86.29	86.80
Gold	%	78.33	78.50
Rhodium	%	79.78	80.30
Copper	%	87.33	87.70
Nickel	%	70.81	71.60
Concentrate Produced	kt (dry)	5,523	5,545
Platinum	g/t	38.28	38.21
Palladium	g/t	39.11	39.00
Gold	g/t	5.33	5.32
Rhodium	g/t	2.43	2.43
3PE+Au	g/t	85.15	84.97
Copper	% Cu	3.31	3.26
Nickel	% Ni	5.47	5.44
Recovered Metal			
Platinum	koz	6,797	6,813
Palladium	koz	6,944	6,954
Gold	koz	947	948
Rhodium	koz	432	433
3PE+Au	koz	15,120	15,149
Copper	Mlb	403	399
Nickel	Mlb	666	665

Table 22.35 Total Capital Costs Comparison

Item	2025 FS ($M)	2022 FS ($M)
Mining		
Mining	1,944	1,816
Capitalised Operating Costs	75	–
Subtotal	**2,019**	**1,816**
Concentrator and Tailings		
Concentrator and Tailings	245	349
Subtotal	**245**	**349**
Infrastructure		
Infrastructure	227	363
Site Costs	2	7
Capitalised Operating Costs	3	–
Subtotal	**232**	**371**
Other Costs		
Owner's Cost	129	222
Closure Cost	9.5	11.0
Capitalised Operating Costs	20	
Subtotal	**159**	**233**
Capex Before Contingency	**2,655**	**2,768**
Contingency	177	134
Capex After Contingency	**2,832**	**2,902**

23 ADJACENT PROPERTIES

This Section not used.

24 OTHER RELEVANT DATA AND INFORMATION

The 10.7 Mtpa Platreef 2025 PEA considers the Phase 3 expansion once the major Shaft #2 is available for hoisting in 2029. This expansion further doubles the annualized platinum, palladium, rhodium, and gold production to over 1 million ounces of 3PE+Au, plus approximately 22,000 tonnes of nickel and 13,000 tonnes of copper. The 10.7 Mtpa PEA generates an NPV8% of $3.2 billion, and an IRR of 25%, at consensus long-term metal prices.

The Platreef 2025 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The 10.7 Mtpa Platreef 2025 PEA includes a further phase of expansion, Phase 3, to a total processing capacity of 10.7 Mtpa, following the completion of two additional 3.3 Mtpa concentrator modules in 2030 and 2032.

Construction of the Shaft #2 concrete headframe is more than 90% complete with construction completion targeted for Q4'25. The remaining headframe construction work is the installation of the material and production winders, which will be used initially for 4.1 Mtpa Platreef 2025 FS operations and then for 10.7 Mtpa Platreef 2025 PEA.

LOM total cash costs for the 10.7 Mtpa Platreef 2025 PEA are expected to be $511 per oz of 3PE+Au, net of by-products, benefitting from significant economies of scale. Including sustaining capital, total cash costs are expected to be $641 per ounce of 3PE+Au, net of by-products.

Annualized production in the 10.7 Mtpa Platreef 2025 PEA, once fully ramped up, is expected to be between 1.0 and 1.2 million oz of 3PE+Au, plus approximately 22,000 tonnes of nickel and 13,000 tonnes of copper.

The 10.7 Mtpa Platreef 2025 PEA uses Shaft #2 and Shaft #3 for hoisting ore and waste with a combined total capacity of over 12 Mtpa. The incremental expansion capital cost for the 10.7 Mtpa PEA is estimated at $803 million, leveraging the significant surface and underground infrastructure already constructed during Phase 2. The 10.7 Mtpa PEA delivers an NPV8% of $3.2 billion and an IRR of 25%, based on long-term consensus prices over a mine life of 29 years.

Key steps involved in preparing the Platreef 2025 PEA are as follows:

- Phase 1 concentrator 770 ktpa was completed in June 2024 and Shaft 1 commissioned for hoisting labour and material in 2022. Therefore, the first feed of ore is expected to commence in Q4'25.
- Phase 2 new 3.3 Mtpa concentrator module built alongside the Phase 1 module, increasing the processing capacity to 4.1 Mtpa.
- Shaft 3 will provide an additional 4 Mtpa of hoisting capacity (ore and waste rock).

- Shaft 2 will support Phase 1 and Phase 2 by commencing first with hoisting labour and materials.
- Phase 3 consists of a further two 3.3 Mtpa concentrator modules, built sequentially and located alongside the Phase 1 and Phase 2 modules, which increase the total Phase 3 concentrator capacity to 10.7 Mtpa.

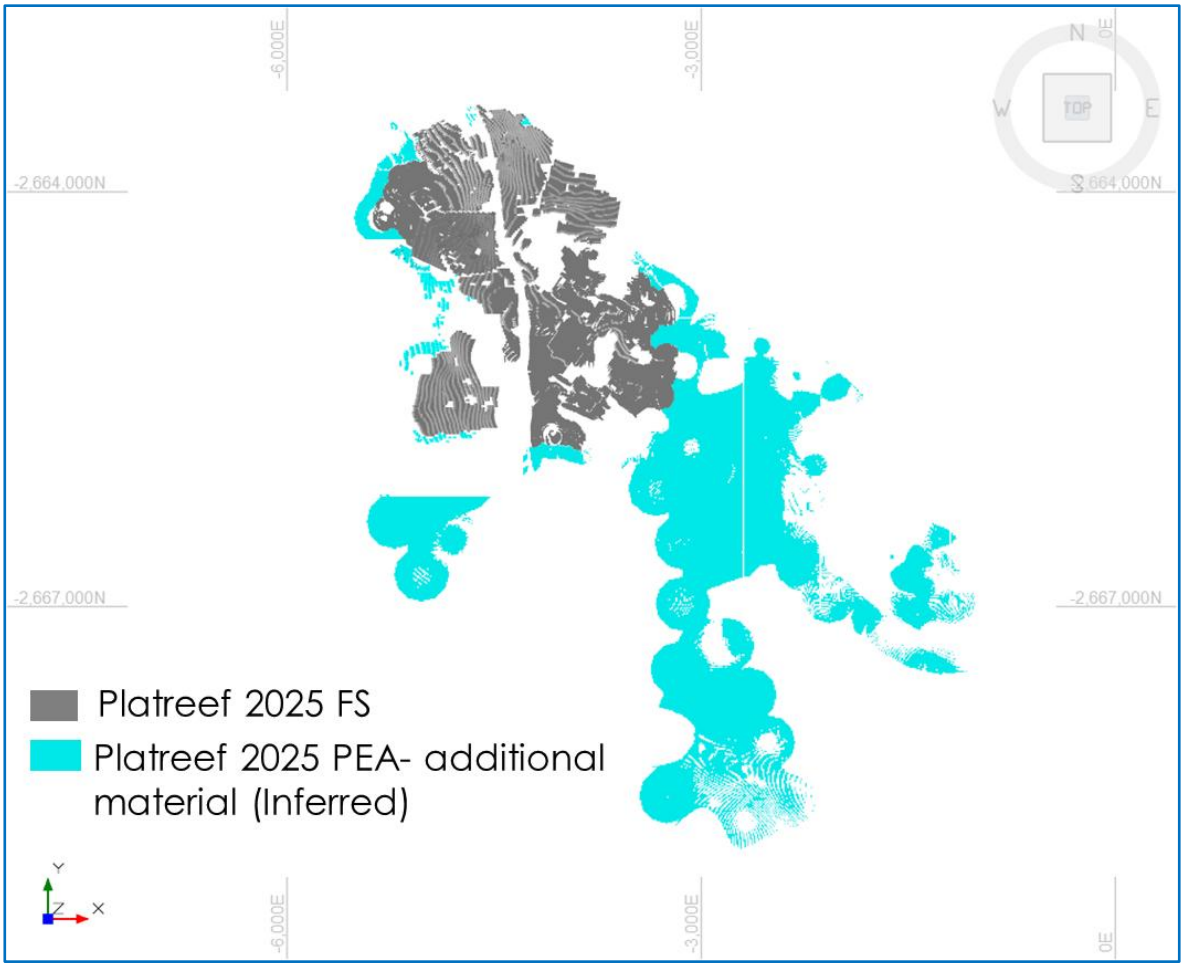
The key dates for the Platreef 2025 PEA are summarised in Table 24.1.

Table 24.1 Platreef 2025 PEA Key Dates

Activity Name	Start	Finish
Shaft 3 Equipping	Q4'24	Q1'26
Shaft 2 Sinking to –60 m Level	Q4'25	Q1'26
Ventilation Shaft (Shaft 4)		Q3'25
Plant 1 Start Up		Q4'25
Ventilation Shaft (Shaft 5)	Q4'25	Q1'27
Shaft 2 Sinking to –995 m Level	Q1'26	Q1'28
Shaft 2 Equipping Complete		Q3'28
Shaft 2 Expansion (First rock winders)		2029
Plant 2 Start Up		Q4'27
Plant 3 Start Up		Q2'30
Plant 4 Start Up		Q4'32
Mine Production Steady State (10.67 Mtpa)	2033	

Figure 24.1 shows the plan of the areas included in the Platreef 2025 FS mine design and the additional material included in the Platreef 2025 PEA mine design.

Figure 24.1 Platreef 2025 PEA Design



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The designs for the 770 ktpa and 3.3 Mtpa concentrators are based on the Platreef 2025 FS process flow sheet. Additionally, the construction of two more 3.3 Mtpa concentrators is being considered to reach a total throughput of 10.67 Mtpa. The ROM will be treated in a modular multi-stage crushing and screening circuit followed by ball milling and flotation.

The optimised flotation circuit flow sheet as developed during testing was used as the basis for the flotation circuit design with the inclusion of Jameson flotation cells for the final concentrate cleaning stages to reduce capital costs while still catering for the required final concentrate 3E+Au grade of 85.5 g/t.

Water recovery will be achieved by thickening and filtration of both the concentrate and tailings streams prior to each being dispatched to the smelter and Dry Stack Tailings (DST) area respectively.

The production of the Platreef 2025 PEA by Resource category is shown in Figure 24.3

Figure 24.2 Platreef 2025 PEA Phased Development

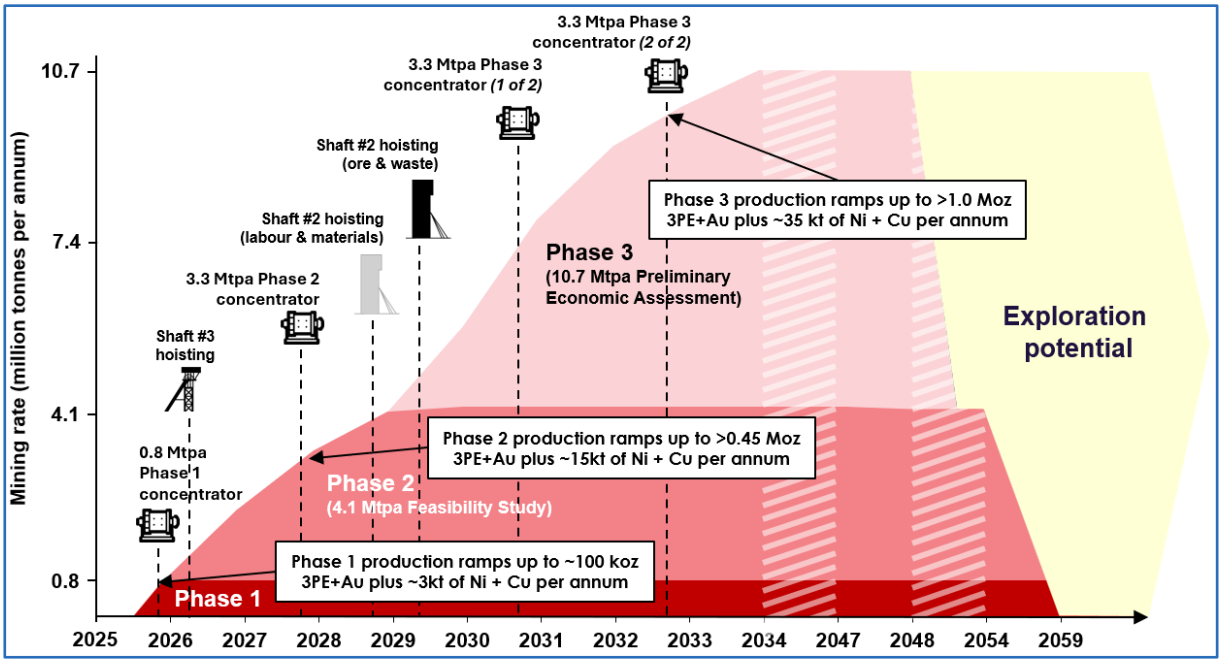

Figure 24.3 Platreef 2025 PEA by Resource Category

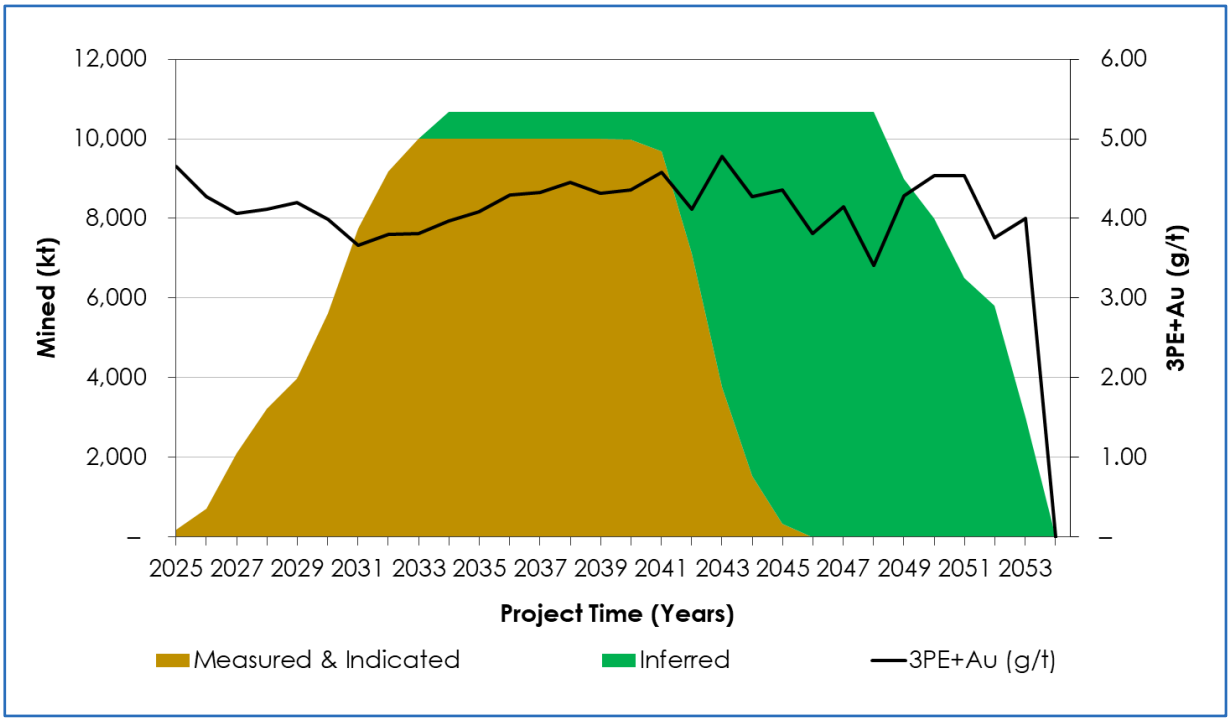

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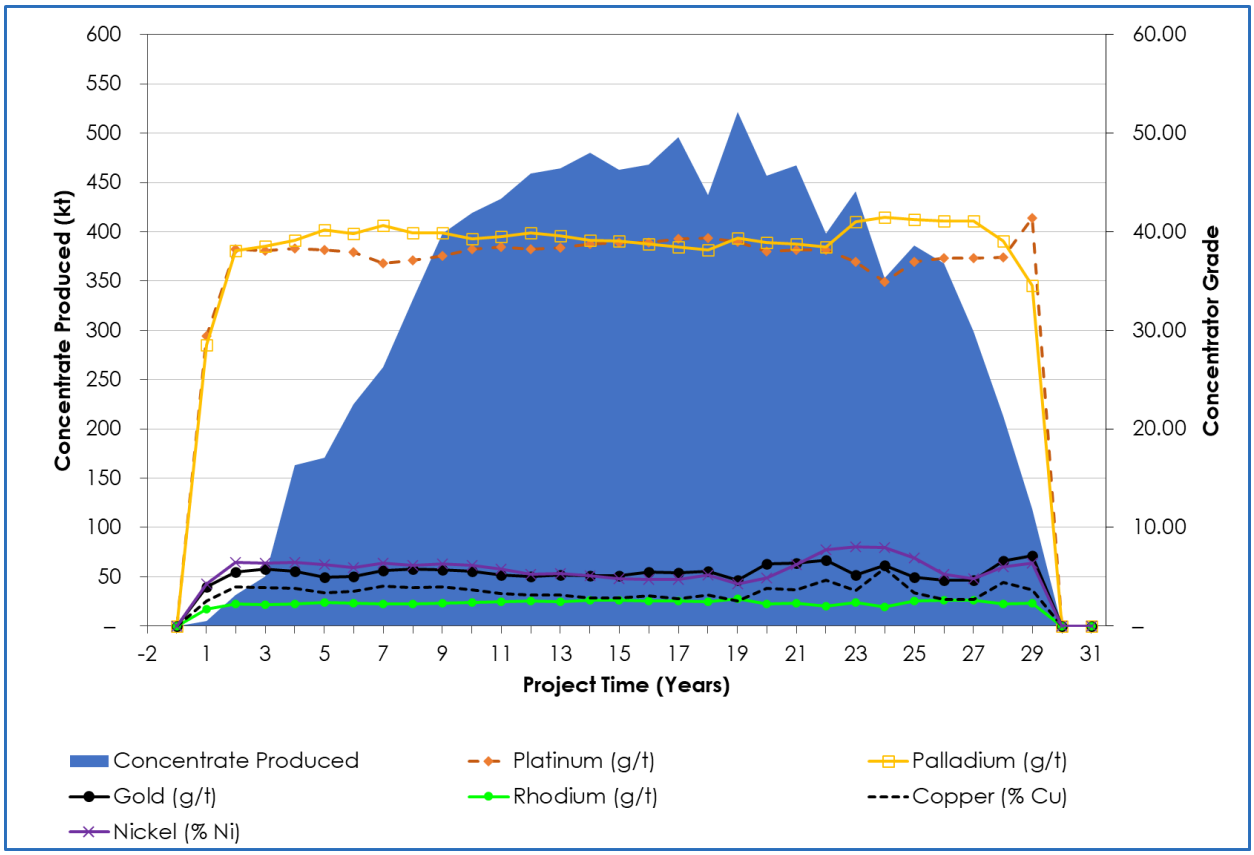
Figure 24.4 Platreef 2025 FS Concentrate Produced



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24.1 Platreef 2025 PEA Cost Estimation

The 10.7 Mtpa Platreef 2025 PEA includes a further phase of expansion, Phase 3, to a total processing capacity of 10.7 Mtpa, following the completion of two additional 3.3 Mtpa concentrator modules in 2030 and 2032. LOM total cash costs for the 10.7 Mtpa Platreef 2025 PEA are expected to be $511 per oz, leveraging the significant surface and underground infrastructure already constructed during Phase 2.

24.2 Platreef 2025 PEA Financial Results Summary

The key features of the Platreef 2025 PEA include:

- The incremental expansion capital cost for the 10.7 Mtpa Platreef 2025 PEA is estimated at $803 million.
- After-Tax Net Present Value (NPV) of $3,195M, at an 8% discount rate.
- After-Tax Internal Rate of Return (IRR) of 25.03%.

The PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The key production and financial results including Net Present Value at 8% Discount Rate (NPV8%) and Internal Rate of Return (IRR) of the Platreef 2025 FS and Platreef 2025 PEA are shown in Table 24.2. The Platreef 2025 PEA describes a change in production rate for the Project that will require separate capital costs and infrastructure. The Platreef 2025 PEA is preliminary in nature and there is no certainty that the results will be realised.

Table 24.2 Platreef 2025 FS & Platreef 2025 PEA Key Production and Financial Results

Item	Unit	2025 FS	2025 PEA
Mined and Processed	Mt	130	235
Platinum	g/t	1.88	1.83
Palladium	g/t	1.93	1.92
Gold	g/t	0.29	0.29
Rhodium	g/t	0.13	0.13
3PE+Au	g/t	4.22	4.17
Copper	%	0.16	0.16
Nickel	%	0.33	0.34
Concentrate Produced	kt	5,523	9,780
Platinum	g/t	38.28	38.08
Palladium	g/t	39.11	39.52
Gold	g/t	5.33	5.48
Rhodium	g/t	2.43	2.43
3PE+Au	g/t	85.15	85.50
Copper	%	3.31	3.46
Nickel	%	5.47	5.79
Recovered Metal			
Platinum	koz	6,797	11,974
Palladium	koz	6,944	12,426
Gold	koz	947	1,722
Rhodium	koz	432	763
3PE+Au	koz	15,120	26,885
Copper	Mlb	403	746
Nickel	Mlb	666	1,248
Key Financial Results			
Life-of-Mine	Years	35	29
Pre-Production	$M	1,243	1,243
Total cash cost after credits	$/oz Rec. 3PE+Au	599	511
All-in cash cost after credits	$/oz Rec. 3PE+Au	704	641
Site Operating Costs	$/t Milled	60	56
After-Tax NPV8%	$M	1,393	3,195
After-Tax IRR	%	19.77	25.03
Project Payback Period	Years	7.8	8.7

1. 3PE+Au = platinum, palladium, rhodium and gold.
2. Includes the impact of the gold, platinum and palladium streaming agreements
3. All-in cash costs include sustaining capital costs.
4. Long-term metal price assumptions for economic analysis are as follows: $1,200/oz platinum, $1,130/oz palladium, $2,170/oz gold, $5,000/oz rhodium, $8.50/lb nickel and $4.25/lb copper.
5. 4.1 Mtpa FS Initial Capital includes $70 million for the completion of Phase 1.

The PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

24.2.1 Platreef 2025 PEA Results

The results of the financial analysis show an After-Tax NPV8% of $3,195M. The Platreef 2025 PEA exhibits an After-Tax IRR of 25.03% and a payback period of 8.7 years. The estimates of cash flows have been prepared on a real basis as at 1 January 2025 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 24.3. The mining production statistics are shown in Table 24.4.

Table 24.3 Platreef 2025 PEA Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	16,682	12,733
	5.00%	6,865	5,263
	8.00%	4,171	3,195
	10.00%	3,015	2,301
	12.00%	2,182	1,652
	15.00%	1,329	983
	20.00%	518	339
IRR	–	26.82%	25.03%
Project Payback (Years)	–	8.7	8.7

Concentrator feed and estimated concentrator production along with grades for the LOM are depicted in Figure 24.5 and Figure 24.6.

Table 24.4 Platreef 2025 PEA Mining Production Statistics

Item	Unit	Total LOM	Years 1-3 Average	Years 7–30 Average	LOM Average
Nameplate Throughput	Mt	235	0.77	10.67	10.67
Platinum	g/t	1.83	1.85	1.83	1.83
Palladium	g/t	1.92	1.87	1.92	1.92
Gold	g/t	0.29	0.30	0.29	0.29
Rhodium	g/t	0.13	0.12	0.13	0.13
3PE+Au	g/t	4.17	4.14	4.17	4.17
Copper	%	0.16	0.18	0.16	0.16
Nickel	%	0.34	0.36	0.34	0.34
Concentrator Recoveries					
Platinum	%	86.49	85.87	86.50	86.49
Palladium	%	85.83	85.19	85.84	85.83
Gold	%	78.06	77.42	78.06	78.06
Rhodium	%	79.31	78.61	79.32	79.31
3PE+Au	%	85.38	84.74	85.39	85.38
Copper	%	87.68	88.96	87.67	87.68
Nickel	%	71.66	73.55	71.64	71.66
Concentrate Produced	kt(dry)	9,780.45	29.04	372.82	337.26
Platinum	g/t	38.08	37.61	38.08	38.08
Palladium	g/t	39.52	37.76	39.53	39.52
Gold	g/t	5.48	5.55	5.48	5.48
Rhodium	g/t	2.43	2.18	2.43	2.43
3PE+Au	g/t	85.50	83.11	85.52	85.50
Copper	%	3.46	3.84	3.45	3.46
Nickel	%	5.79	6.29	5.78	5.79
Recovered Metal					
Platinum	koz	11,974	47	456	413
Palladium	koz	12,426	47	474	428
Gold	koz	1,722	7	66	59
Rhodium	koz	763	3	29	26
3PE+Au	koz	26,885	103	1,025	927
Copper	Mlb	746	3	28	26
Nickel	Mlb	1,248	5	48	43

1. 3PE+Au is the sum of the grades for and production of platinum, palladium, rhodium, and gold.
2. Duration of Year 1 is 0.25 years.

Figure 24.5 Platreef 2025 PEA Concentrator Production

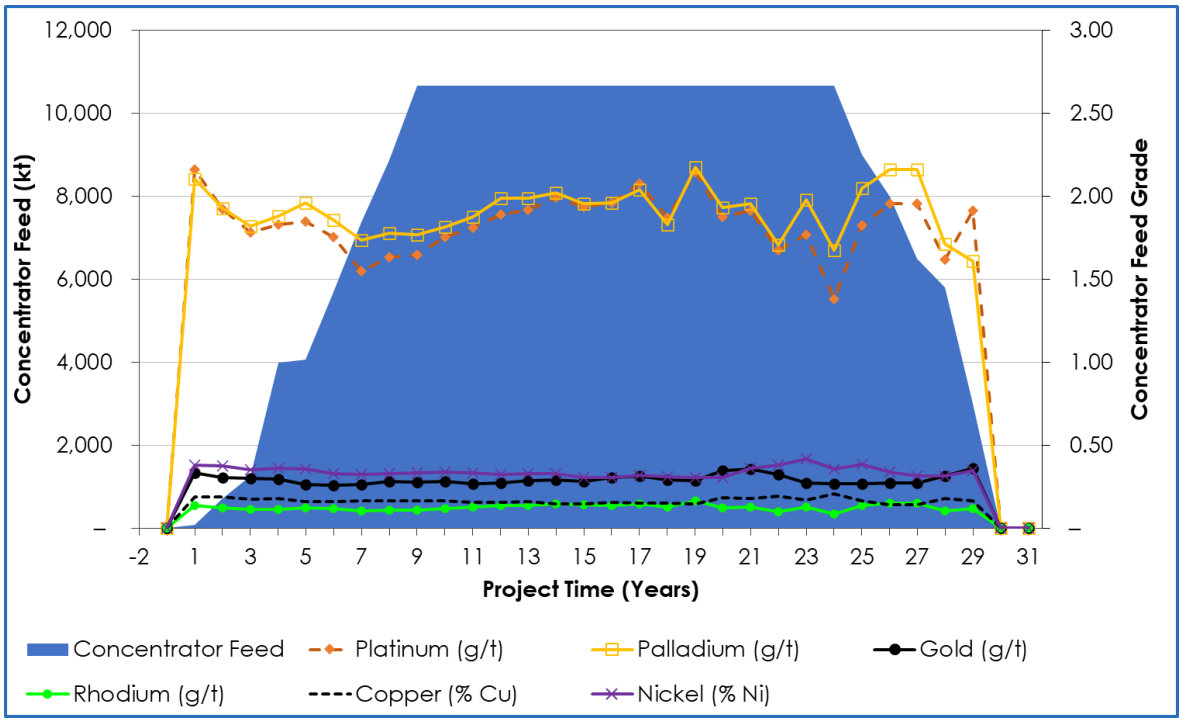

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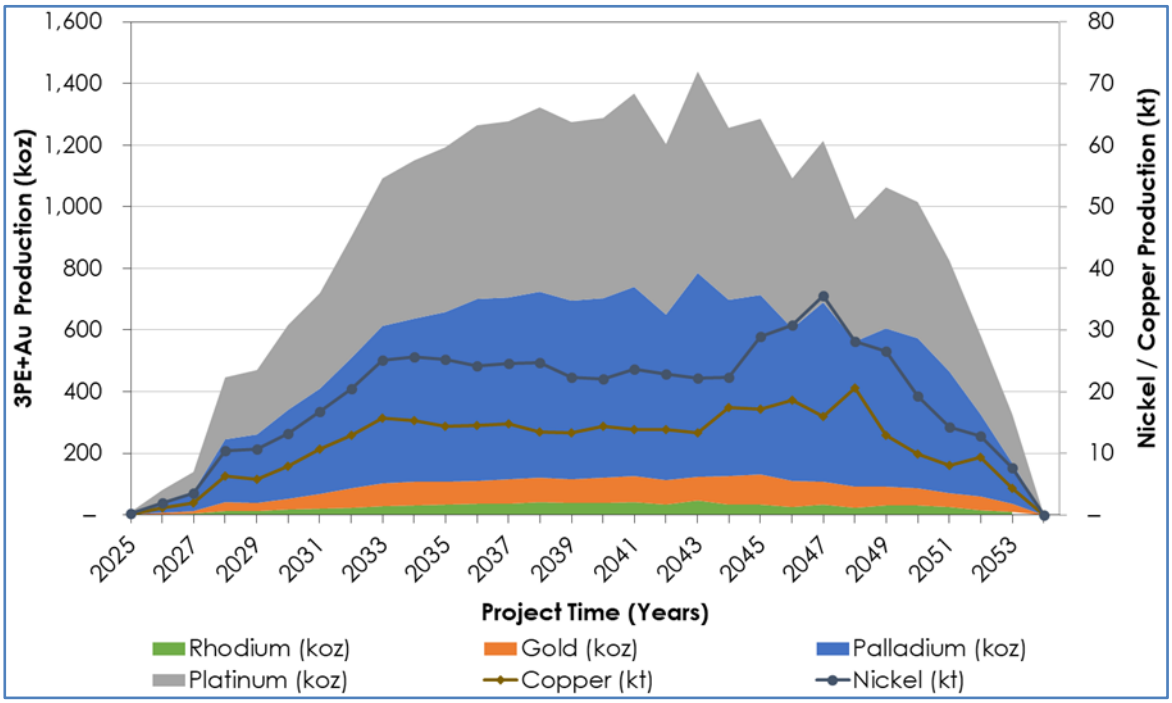
Figure 24.6 Platreef 2025 PEA Estimated Contained 3PE+Au and Nickel and Copper Metal



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24.2.2 Capital and Operating Cost Summary

Capital investment for Platreef 2025 PEA is summarised in Table 24.5.

Table 24.5 Platreef 2025 PEA Capital Investment Summary

Item	Initial 4.1 Mtpa FS ($M)	Expansion 10.7 Mtpa PEA ($M)	Sustaining Capital ($M)	Total LOM ($M)
Mining				
Mining	581	233	2,845	3,658
Capitalised Operating Costs	75	0	0	75
Subtotal	**655**	**233**	**2,845**	**3,733**
Concentrator				
Concentrator & Infrastructure	**373**	**397**	**292**	**1,062**
Other Costs				
Owner's Cost	95	58	120	273
Closure Cost	-	-	35	35
Capitalised Operating Costs	20	-	-	20
Subtotal	**116**	**58**	**154**	**328**
Capex Before Contingency	**1,144**	**688**	**3,292**	**5,119**
Contingency	99	115	213	427
Capex After Contingency	**1,243**	**803**	**3,505**	**5,550**

Initial capital reflects the capital costs from 1 January 2025 to achieve initial production of 4.1 Mtpa, followed by expansion capital to achieve full production of 10.7 Mtpa. Total PEA is the sum of initial, expansion, and sustaining capital.

Revenues and Operating Costs for the Platreef 2025 PEA are shown in Table 24.6.

Table 24.6 Platreef 2025 PEA Cash Costs

Item	$ per ounce of 3PE+Au		
	Years 1 - 3	Years 4 - 29	Life-of-Mine Average
Mine site	1,010	485	489
Transport	8	8	8
Treatment & Refining	393	374	374
Au, Pd & Pt Streaming Agreements	156	67	68
Government Royalties	8	85	84
Total cash costs before credits	**1,575**	**1,018**	**1,023**
Nickel credits	424	394	394
Copper credits	151	118	118
Total cash costs after credits	**1,000**	**506**	**511**
Sustaining capital costs (1)	–	131	130
All-in cash costs after credits (2)	**1,000**	**638**	**641**

1. Phase 1 operating costs include allowance for sustaining capital costs.
2. All-in cash costs include sustaining capital costs.

A summary of the financial results comparing with Platreef 2025 FS are shown in Table 24.7.

Table 24.7 Platreef 2025 FS and PEA Financial Results

	Discount Rate	2025 FS	2020 PEA
Net Present Value After-Tax ($M)	Undiscounted	6,628	12,733
	5.00%	2,450	5,263
	8.00%	1,393	3,195
	10.00%	950	2,301
	12.00%	632	1,652
	15.00%	306	983
	20.00%	-11	339
IRR	–	19.77%	25.03%
Project Payback (Years)	–	7.8	8.7

The cash flow After-Tax is depicted in Figure 24.7. Year 1 is 2025. The details of mine and process production and the Project cash flow are shown in Table 24.8 to Table 24.13.

Figure 24.7 Platreef 2025 PEA Cumulative Cash Flow After-Tax

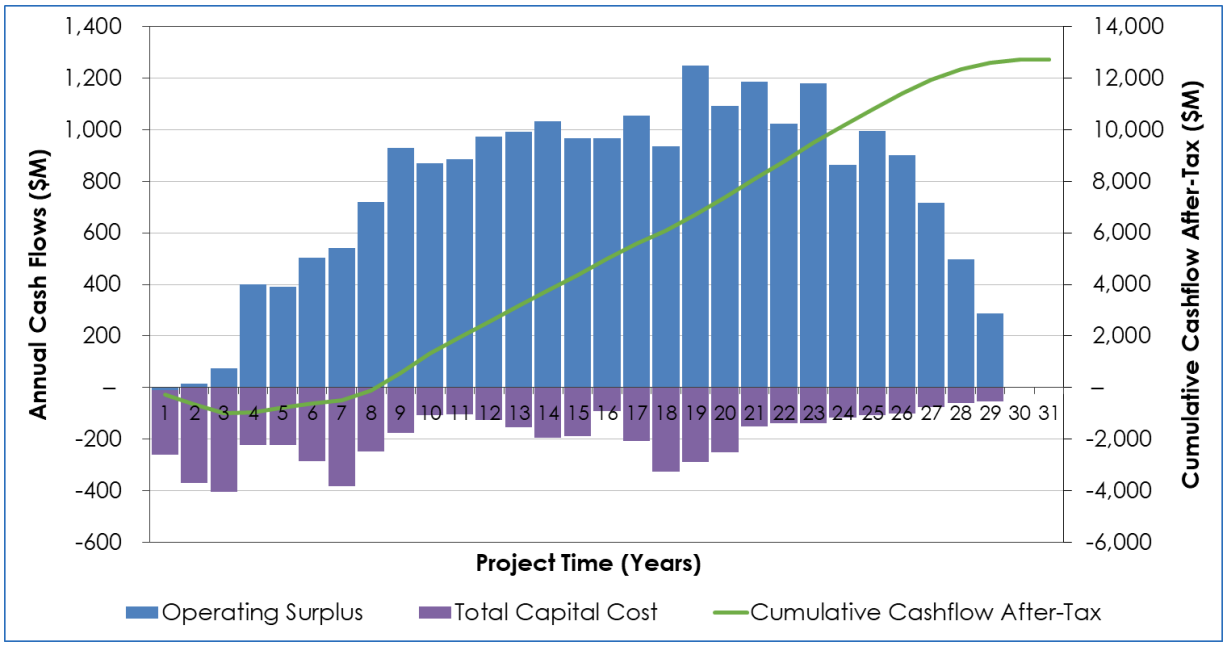

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Table 24.8 Platreef 2025 PEA Mine Production

Item	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Total ROM	kt	235,077	172	704	2,104	3,220	3,983	5,626	7,739	9,179	9,999	106,700	85,650
NSR	$/t	144	167	157	141	144	140	130	121	128	128	143	151
Platinum	g/t	1.83	2.12	1.92	1.80	1.82	1.85	1.75	1.55	1.63	1.65	1.94	1.77
Palladium	g/t	1.92	2.06	1.93	1.85	1.88	1.96	1.86	1.74	1.78	1.77	1.96	1.91
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.26	0.26	0.26	0.28	0.28	0.29	0.30
Rhodium	g/t	0.13	0.13	0.12	0.11	0.12	0.13	0.12	0.11	0.11	0.11	0.14	0.12
Copper	%	0.16	0.19	0.19	0.18	0.18	0.16	0.16	0.17	0.17	0.17	0.15	0.18
Nickel	%	0.34	0.37	0.38	0.35	0.35	0.33	0.32	0.33	0.33	0.34	0.32	0.36
Sulfur	%	0.83	0.95	1.02	0.93	0.91	0.81	0.79	0.87	0.83	0.84	0.76	0.91
3PE+Au	g/t	4.17	4.65	4.28	4.07	4.11	4.20	3.99	3.66	3.80	3.81	4.33	4.10

Table 24.9 Platreef 2025 PEA Process Production Plant Feed

Item	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrator Feed	kt	235,075	84	715	1,265	3,988	4,070	5,687	7,370	8,879	10,669	106,698	85,650
Platinum	g/t	1.83	2.16	1.93	1.79	1.83	1.85	1.76	1.55	1.63	1.65	1.94	1.77
Palladium	g/t	1.92	2.11	1.93	1.82	1.88	1.96	1.86	1.74	1.78	1.77	1.96	1.91
Gold	g/t	0.29	0.33	0.30	0.30	0.29	0.26	0.26	0.26	0.28	0.28	0.29	0.30
Rhodium	g/t	0.13	0.14	0.12	0.11	0.12	0.13	0.12	0.11	0.11	0.11	0.14	0.12
Copper	%	0.16	0.19	0.19	0.18	0.18	0.16	0.16	0.17	0.17	0.17	0.15	0.18
Nickel	%	0.34	0.38	0.38	0.35	0.36	0.36	0.33	0.32	0.33	0.33	0.32	0.36
Sulfur	%	0.69	0.97	1.00	0.93	0.64	0.54	0.52	0.48	0.46	0.43	0.59	0.89
3PE+Au	g/t	4.17	4.74	4.28	4.02	4.12	4.20	3.99	3.66	3.80	3.81	4.33	4.10

Table 24.10 Platreef 2025 PEA Concentrator Recoveries

Item	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrate Produced	kt	9,780	5	31	50	163	171	225	262	331	398	4,642	3,502
Concentrator Recoveries													
Platinum	%	86.49	88.88	86.79	85.07	85.81	86.56	85.27	84.45	84.67	85.04	87.04	86.18
Palladium	%	85.83	88.65	86.24	84.30	85.20	86.00	84.50	83.20	83.65	84.02	86.55	85.45
Gold	%	78.06	78.80	78.16	76.90	77.58	78.19	77.17	76.74	76.80	77.15	78.49	77.92
Rhodium	%	79.31	82.88	79.76	77.56	78.48	79.35	77.72	76.21	76.74	77.12	80.00	78.75
Copper	%	87.68	88.48	90.20	88.23	88.75	87.47	86.79	87.73	87.53	88.00	86.30	88.87
Nickel	%	71.66	74.03	75.37	72.42	73.73	73.67	70.45	70.10	70.27	71.04	69.64	73.67
Sulfur	%	68.13	72.72	75.11	72.84	65.06	61.76	60.33	59.58	58.07	57.23	62.95	72.13
3PE+Au	%	85.38	87.89	85.73	83.90	84.74	85.56	84.17	83.06	83.38	83.76	86.02	85.01
Mass Pull	%	–	6.54	4.37	3.98	4.10	4.20	3.95	3.56	3.73	3.73	4.35	4.09

Table 24.11 Platreef 2025 PEA Concentrate Grade

Item	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Concentrate Produced	kt	9,780	5	31	50	163	171	225	262	331	398	4,642	3,502
Concentrator Grade													
Platinum	g/t	38.08	29.39	38.25	38.11	38.28	38.15	37.91	36.78	37.11	37.53	38.77	37.43
Palladium	g/t	39.52	28.55	38.08	38.56	39.11	40.19	39.78	40.63	39.89	39.87	39.12	39.90
Gold	g/t	5.48	4.01	5.45	5.79	5.58	4.92	5.05	5.66	5.79	5.73	5.22	5.80
Rhodium	g/t	2.43	1.75	2.23	2.19	2.22	2.38	2.33	2.28	2.26	2.29	2.55	2.33
Copper	%	3.46	2.58	3.95	3.92	3.85	3.39	3.54	4.07	3.91	3.95	3.04	3.84
Nickel	%	5.79	4.29	6.47	6.39	6.49	6.25	5.90	6.37	6.18	6.31	5.11	6.46
Sulfur	%	11.23	10.75	17.19	16.95	10.15	7.95	7.87	8.11	7.11	6.57	8.88	15.79
3PE+Au	g/t	85.50	63.70	84.01	84.65	85.19	85.65	85.06	85.34	85.05	85.41	85.65	85.46

Table 24.12 Platreef 2025 PEA Metal Production

Item	Year Number	Total	1	2	3	4	5	6	7	8	9	10	20
	Year To											19	LOM
Contained Metal													
Platinum	koz	11,974	5	38	62	201	210	274	310	395	480	5,786	4,214
Palladium	koz	12,426	5	38	62	205	221	287	343	424	510	5,837	4,492
Gold	koz	1,722	1	5	9	29	27	36	48	62	73	779	653
Rhodium	koz	763	0	2	4	12	13	17	19	24	29	380	262
Copper	klb	745,624	310	2,719	4,354	13,873	12,784	17,549	23,534	28,497	34,656	311,061	296,286
Nickel	klb	1,247,596	517	4,462	7,097	23,379	23,533	29,222	36,823	45,101	55,430	523,234	498,797
Sulfur	klb	2,421,550	1,295	11,848	18,833	36,568	29,955	39,010	46,911	51,845	57,692	908,920	1,218,674
3PE+Au	koz	26,885	11	84	137	447	470	615	720	904	1,093	12,782	9,621

Table 24.13 Platreef 2025 PEA Cash Flow

Cash Flow Statement ($M)	Total	Year										
Year Number		**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**20**
Year To											**19**	**LOM**
Gross Revenue	49,738	20	163	263	853	879	1,140	1,372	1,711	2,077	22,900	18,358
Realisation Costs	14,355	6	48	78	245	246	320	394	494	600	7,084	4,840
Net Sales Revenue	**35,383**	**14**	**115**	**186**	**608**	**633**	**819**	**978**	**1,217**	**1,477**	**15,816**	**13,518**
Less: Site Operating Costs												
Mining	8,888	13	57	60	122	152	200	295	334	362	4,033	3,260
Concentrator	2,872	2	15	21	55	56	74	94	109	128	1,281	1,037
Infrastructure	642	1	5	6	10	10	15	21	26	27	291	230
General & Administration	748	7	23	24	23	23	25	27	29	30	284	252
Subtotal	**13,151**	**24**	**100**	**111**	**210**	**241**	**315**	**437**	**497**	**547**	**5,889**	**4,779**
Operating Surplus/(Deficit)	**22,232**	**-10**	**15**	**75**	**399**	**392**	**505**	**541**	**720**	**930**	**9,927**	**8,740**
Capital Costs												
Initial Capital Expenditure	1,243	251	372	405	215	–	–	–	–	–	–	–
Expansion Capital Expenditure	803	–	–	–	9	143	191	273	187	–	–	–
Sustaining Capital Expenditure	3,505	–	–	–	–	81	95	108	60	175	1,792	1,193
Working Capital	–	3	27	22	127	4	54	43	73	80	94	-525
Subtotal	**5,550**	**254**	**398**	**427**	**351**	**228**	**340**	**424**	**320**	**255**	**1,886**	**668**
Net Cashflow Before Tax	**16,682**	**-263**	**-383**	**-352**	**48**	**164**	**165**	**117**	**400**	**675**	**8,042**	**8,072**
Depreciation	-5,550	-251	-372	-405	-224	-224	-286	-382	-247	-175	-1,792	-1,193
Income Tax Expense	3,949	–	–	–	–	–	–	–	–	–	1,911	2,038
Net Cashflow After-Tax	**12,733**	**-263**	**-383**	**-352**	**48**	**164**	**165**	**117**	**400**	**675**	**6,130**	**6,034**

24.3 Sensitivity Analysis

A price sensitivity analysis on the long-term metal prices analysis and the impact on NPV8% were performed on the financial model and the results are shown in Table 24.14 to Table 24.25 and Figure 24.8 to Figure 24.19. Palladium, platinum and nickel are the major contributors to Project revenue.

Table 24.14 After-Tax NPV8% Platinum and Nickel Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Nickel Price - $/lb	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	863	1,178	1,488	2,197	2,301	2,599	2,904
4.50	1,143	1,455	1,763	2,467	2,566	2,871	3,181
6.00	1,423	1,731	2,030	2,736	2,838	3,148	3,448
8.50	1,877	2,183	2,481	3,195	3,294	3,599	3,908
10.00	2,151	2,449	2,752	3,463	3,566	3,875	4,185
12.50	2,597	2,904	3,208	3,923	4,026	4,336	4,642
15.00	3,055	3,354	3,661	4,383	4,486	4,790	5,099

Table 24.15 After-Tax NPV8% Sensitivity to Platinum and Nickel Prices

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Nickel Price - $/lb	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	-73%	-63%	-53%	-31%	-28%	-19%	-9%
4.50	-64%	-54%	-45%	-23%	-20%	-10%	-0%
6.00	-55%	-46%	-36%	-14%	-11%	-1%	8%
8.50	-41%	-32%	-22%	–	3%	13%	22%
10.00	-33%	-23%	-14%	8%	12%	21%	31%
12.50	-19%	-9%	0%	23%	26%	36%	45%
15.00	-4%	5%	15%	37%	40%	50%	60%

Figure 24.8 Platinum and Nickel Price Sensitivity

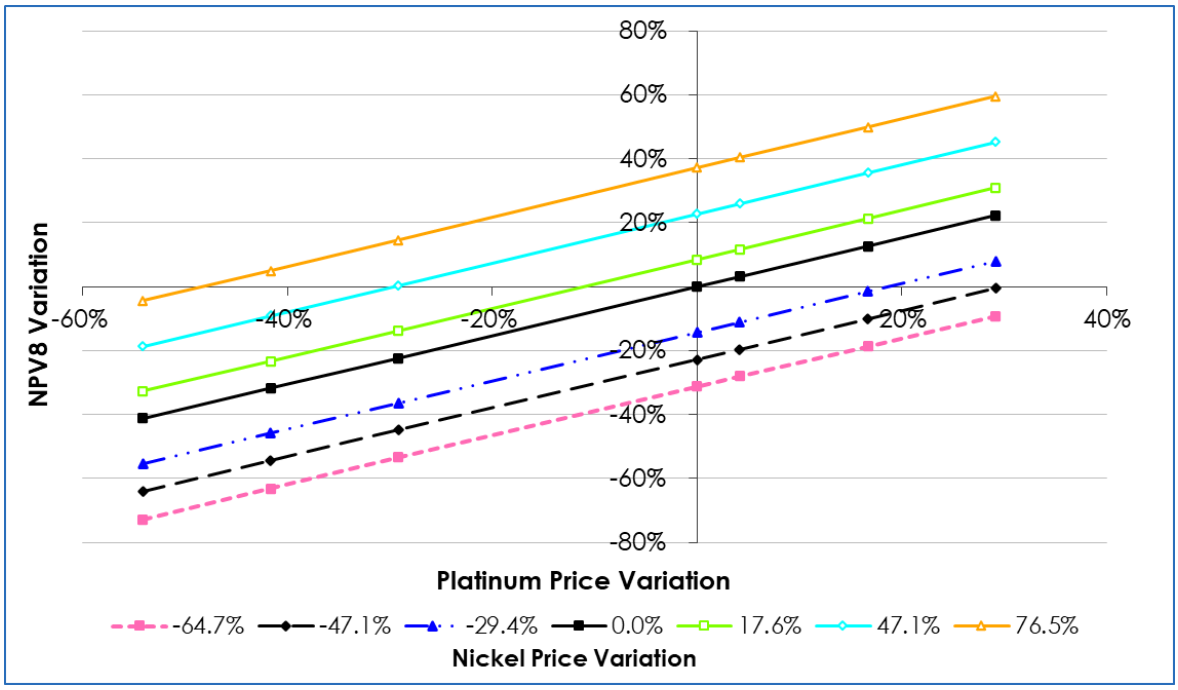

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Figure 24.9 Nickel and Platinum Price Sensitivity

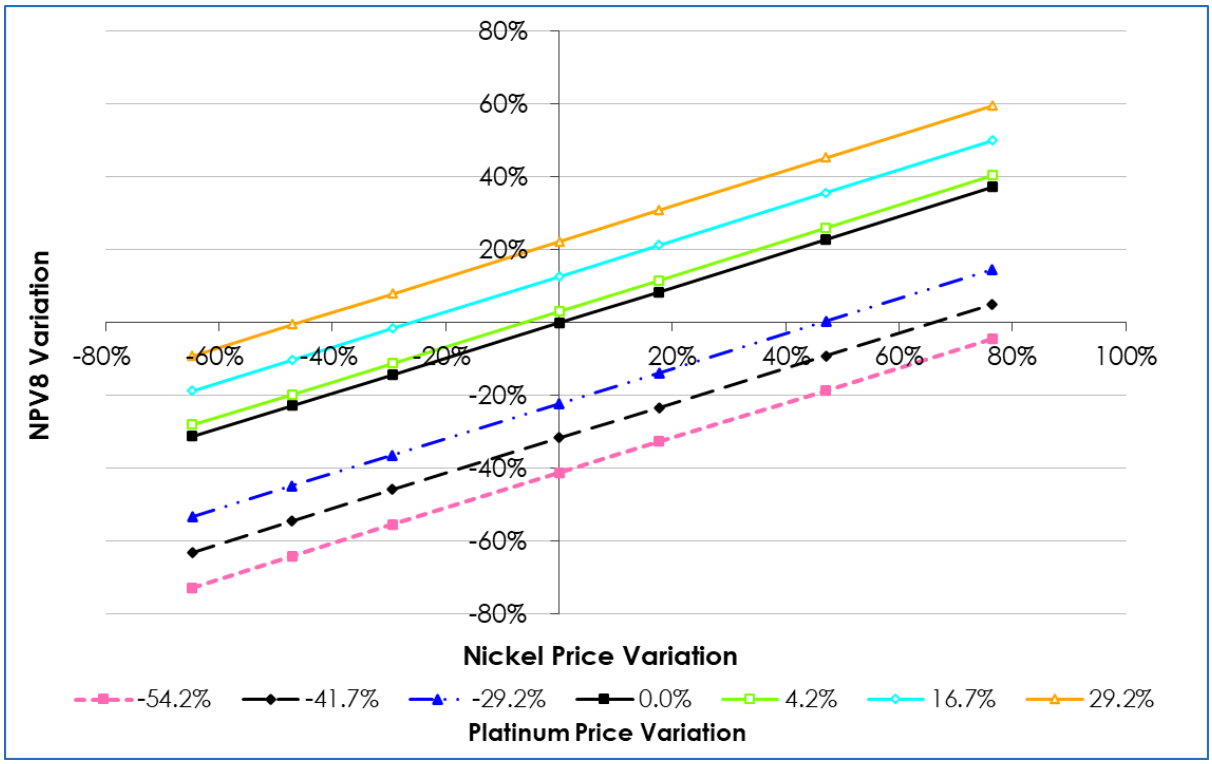

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Table 24.16 After-Tax NPV8% Platinum and Palladium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Palladium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
800	1,176	1,487	1,792	2,496	2,597	2,903	3,211
1,000	1,606	1,908	2,214	2,919	3,022	3,325	3,629
1,130	1,877	2,183	2,481	3,195	3,294	3,599	3,908
1,450	2,546	2,851	3,159	3,871	3,974	4,284	4,593
1,700	3,076	3,377	3,683	4,407	4,510	4,815	5,122
2,000	3,706	4,017	4,326	5,042	5,145	5,455	5,764
2,500	4,773	5,081	5,390	6,112	6,212	6,516	6,825

Table 24.17 After-Tax NPV8% Sensitivity to Platinum and Palladium Prices

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Palladium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
800	-63%	-53%	-44%	-22%	-19%	-9%	0%
1,000	-50%	-40%	-31%	-9%	-5%	4%	14%
1,130	-41%	-32%	-22%	–	3%	13%	22%
1,450	-20%	-11%	-1%	21%	24%	34%	44%
1,700	-4%	6%	15%	38%	41%	51%	60%
2,000	16%	26%	35%	58%	61%	71%	80%
2,500	49%	59%	69%	91%	94%	104%	114%

Figure 24.10 Platinum and Palladium Price Sensitivity

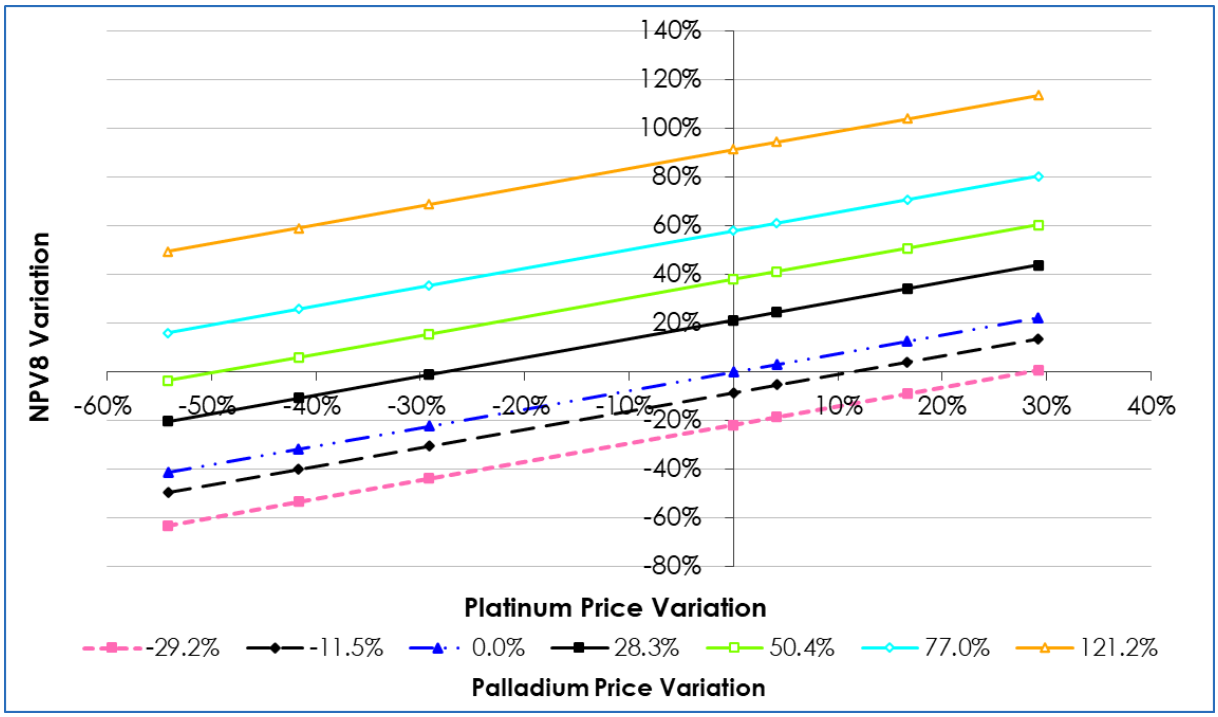

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Figure 24.11 Palladium and Platinum Price Sensitivity

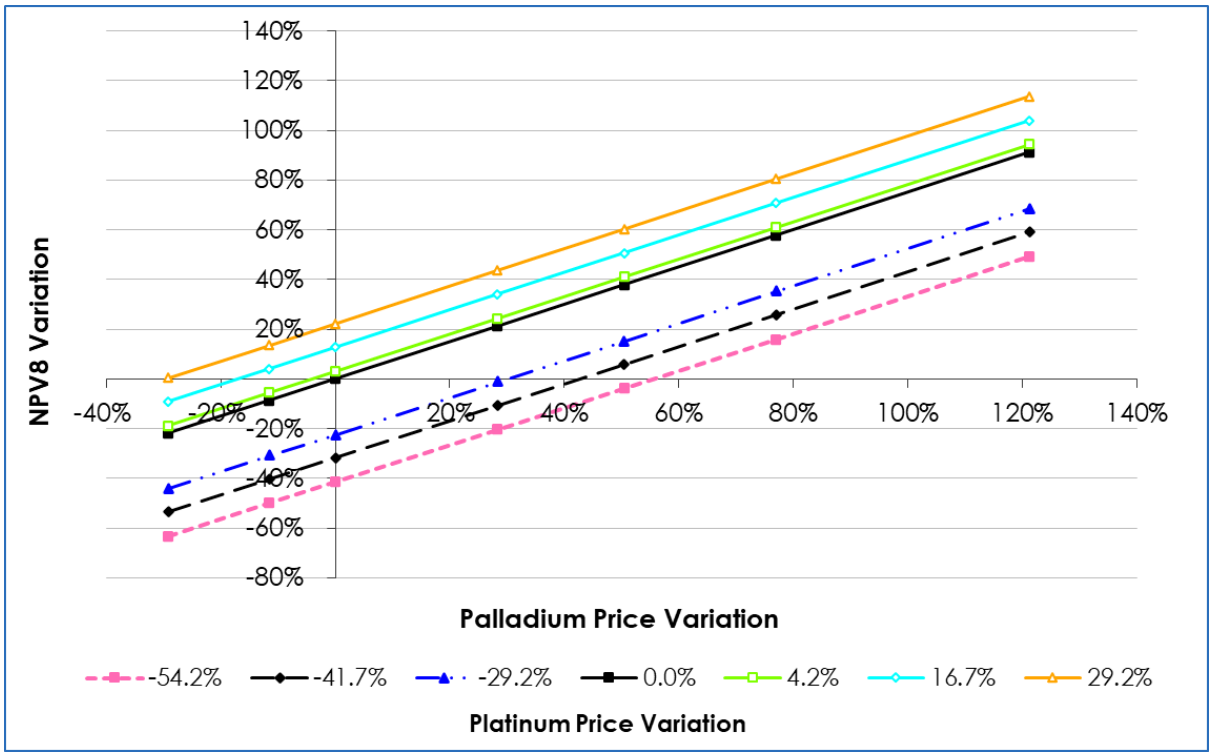

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Table 24.18 After-Tax NPV8% Platinum and Gold Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Gold Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,600	1,831	2,137	2,435	3,148	3,248	3,550	3,860
1,800	1,847	2,153	2,451	3,165	3,264	3,567	3,877
2,000	1,863	2,169	2,467	3,181	3,280	3,584	3,894
2,170	1,877	2,183	2,481	3,195	3,294	3,599	3,908
2,500	1,904	2,210	2,507	3,222	3,322	3,627	3,937
2,700	1,921	2,226	2,524	3,239	3,339	3,644	3,954
2,900	1,937	2,243	2,540	3,255	3,355	3,661	3,971

Table 24.19 After-Tax NPV8% Sensitivity to Platinum and Gold Prices

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Gold Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,600	-43%	-33%	-24%	-1%	2%	11%	21%
1,800	-42%	-33%	-23%	-1%	2%	12%	21%
2,000	-42%	-32%	-23%	-0%	3%	12%	22%
2,170	-41%	-32%	-22%	–	3%	13%	22%
2,500	-40%	-31%	-22%	1%	4%	14%	23%
2,700	-40%	-30%	-21%	1%	4%	14%	24%
2,900	-39%	-30%	-20%	2%	5%	15%	24%

Figure 24.12 Platinum and Gold Price Sensitivity

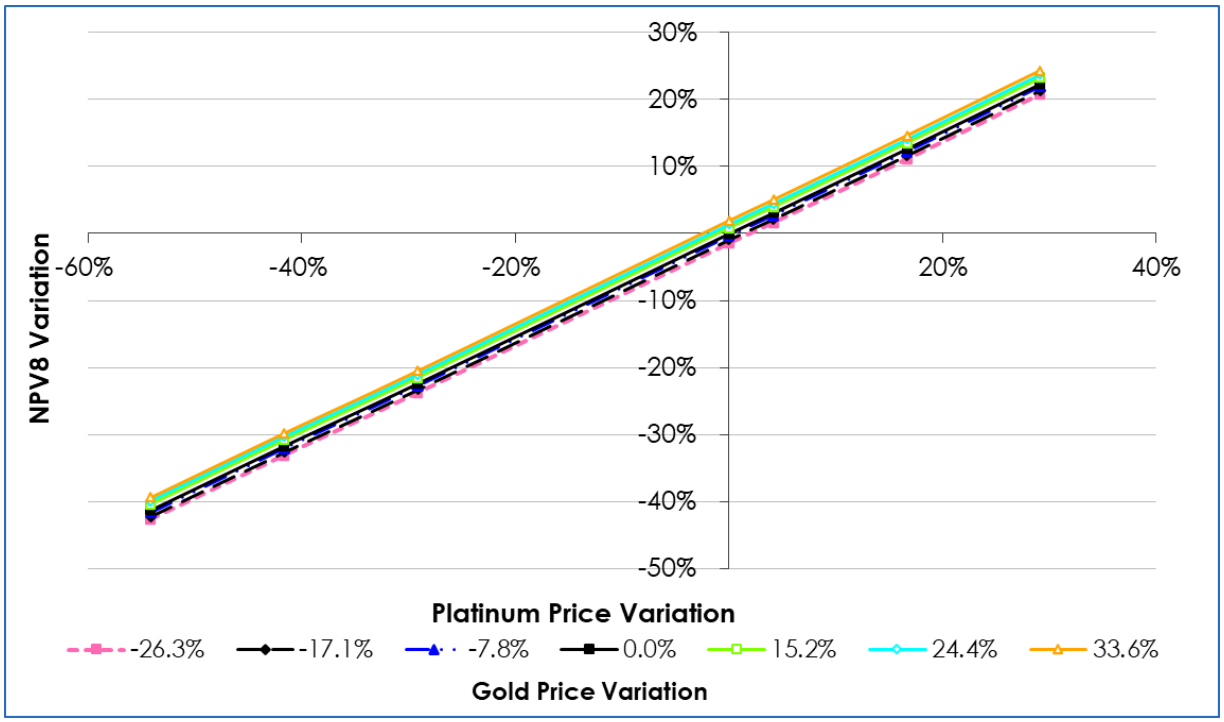

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Figure 24.13 Gold and Platinum Price Sensitivity

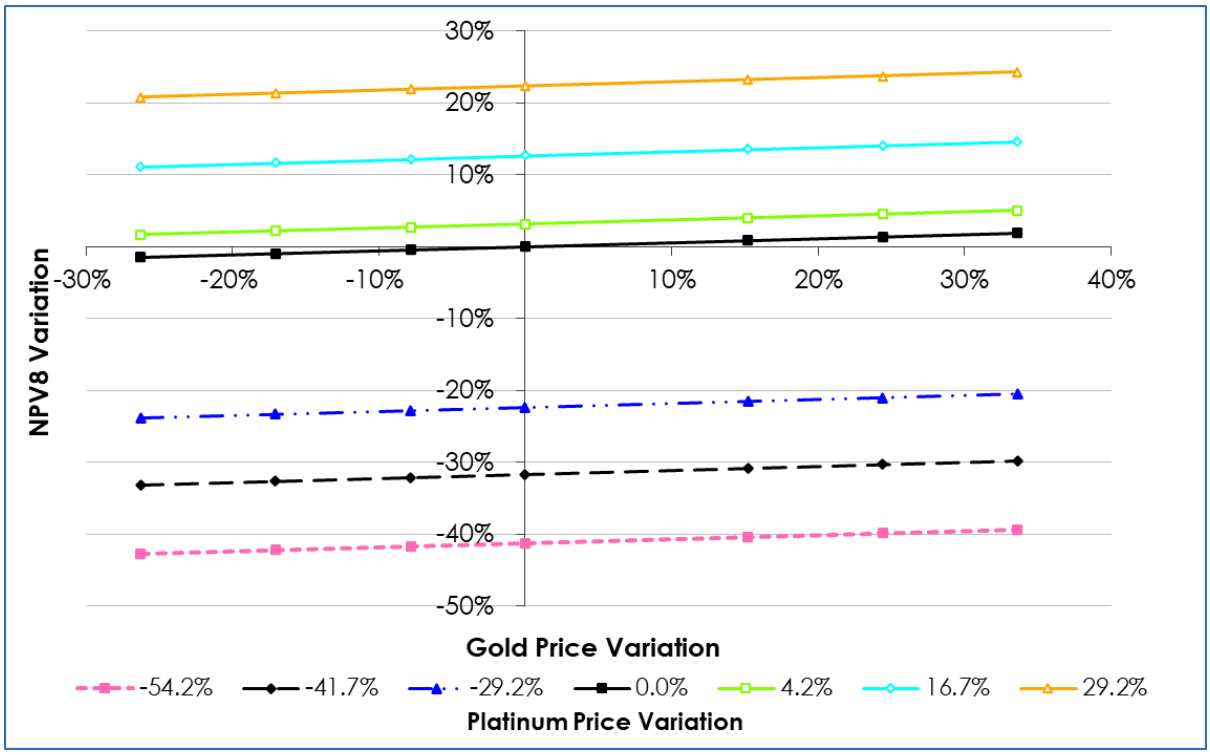

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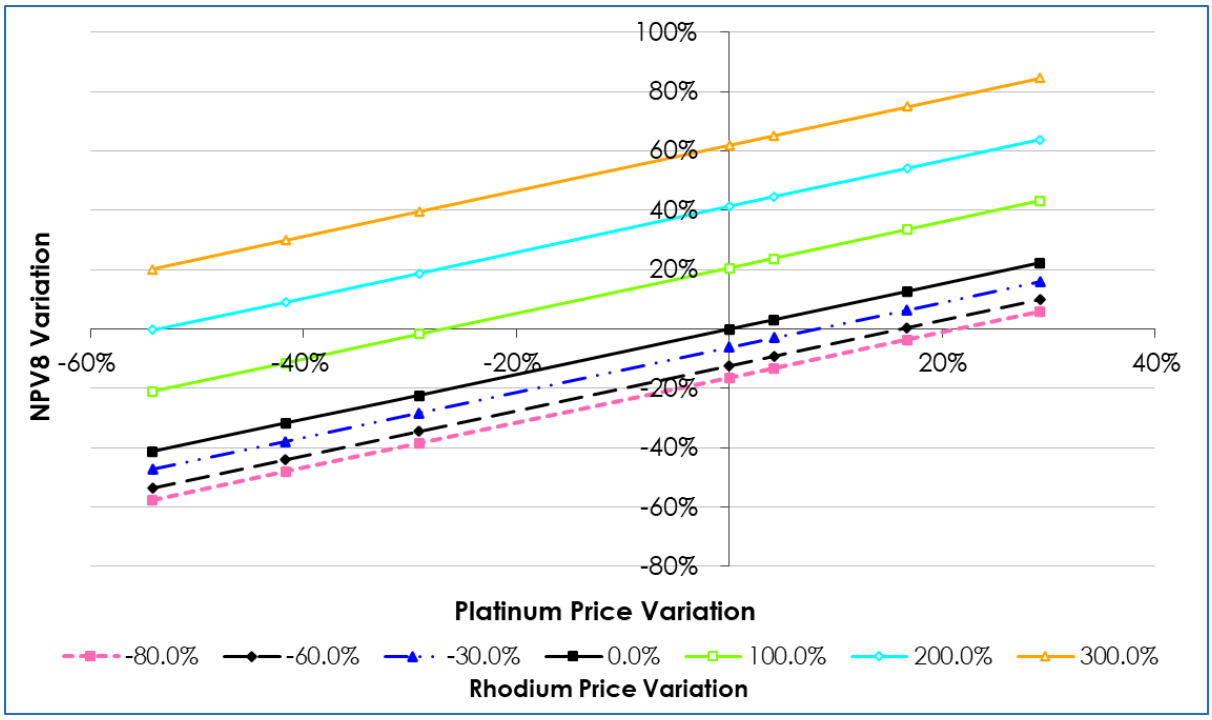
Table 24.20 After-Tax NPV8% Platinum and Rhodium Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Rhodium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,000	1,351	1,661	1,963	2,667	2,768	3,078	3,377
2,000	1,486	1,790	2,093	2,798	2,901	3,208	3,510
3,500	1,684	1,984	2,291	2,998	3,100	3,400	3,708
5,000	1,877	2,183	2,481	3,195	3,294	3,599	3,908
10,000	2,526	2,830	3,139	3,851	3,955	4,264	4,574
15,000	3,185	3,486	3,794	4,517	4,621	4,924	5,233
20,000	3,840	4,151	4,460	5,176	5,279	5,589	5,898

Table 24.21 After-Tax NPV8% Sensitivity to Platinum and Rhodium Prices

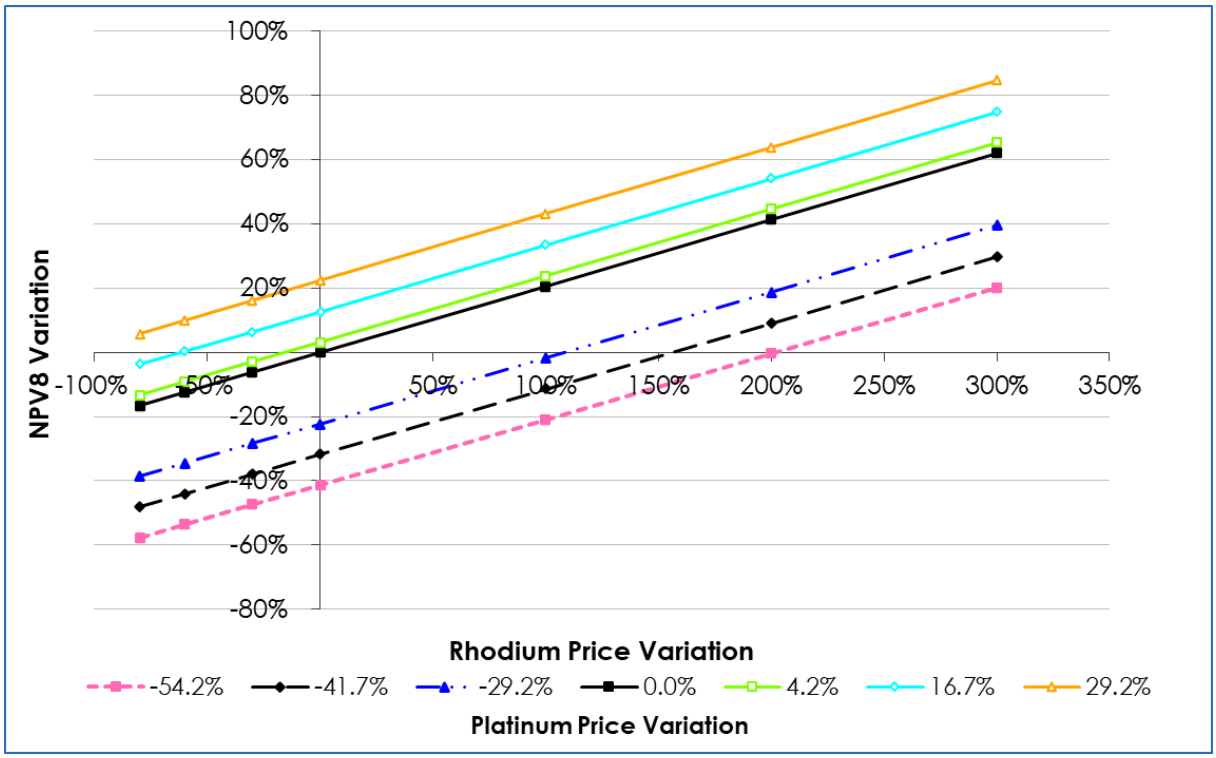
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Rhodium Price - $/oz	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
1,000	-58%	-48%	-39%	-17%	-13%	-4%	6%
2,000	-53%	-44%	-34%	-12%	-9%	0%	10%
3,500	-47%	-38%	-28%	-6%	-3%	6%	16%
5,000	-41%	-32%	-22%	–	3%	13%	22%
10,000	-21%	-11%	-2%	21%	24%	33%	43%
15,000	-0%	9%	19%	41%	45%	54%	64%
20,000	20%	30%	40%	62%	65%	75%	85%

Figure 24.14 Platinum and Rhodium Price Sensitivity



OreWin, 2025

Figure 24.15 Rhodium and Platinum Price Sensitivity



OreWin, 2025

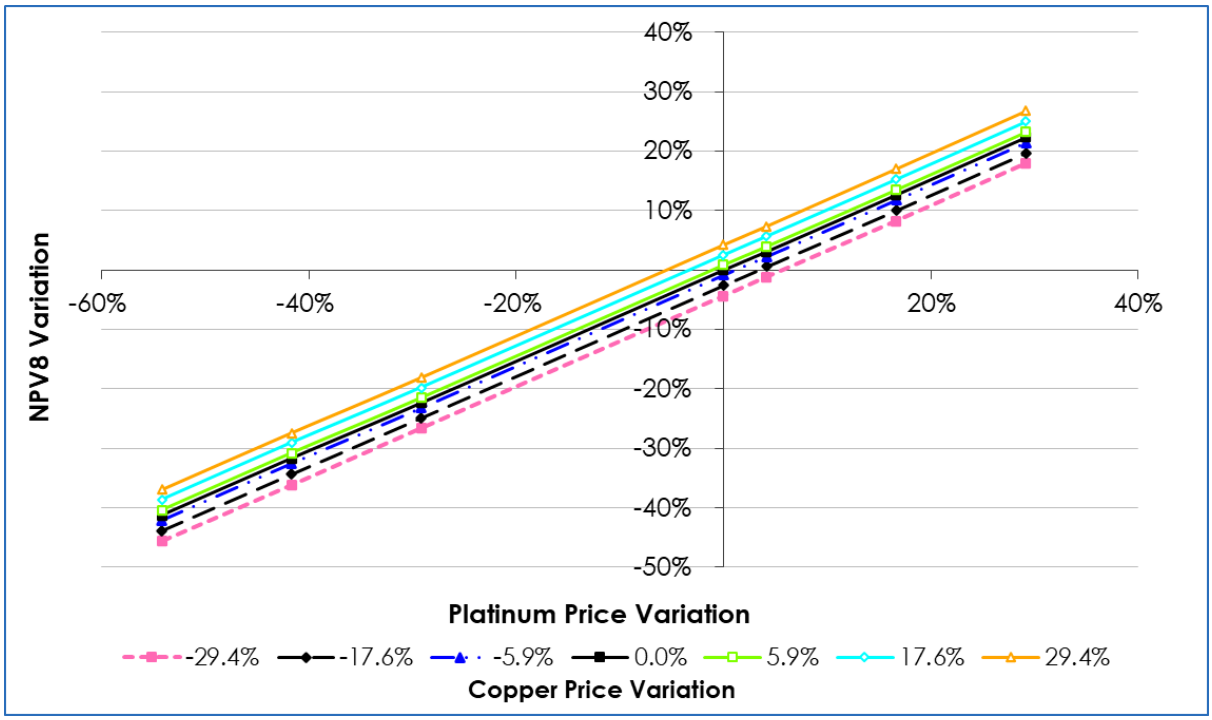
Table 24.22 After-Tax NPV8% Platinum and Copper Price Sensitivity

After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Copper Price - $/lb	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	1,741	2,042	2,347	3,056	3,159	3,459	3,768
3.50	1,795	2,098	2,400	3,112	3,213	3,515	3,824
4.00	1,850	2,155	2,453	3,168	3,267	3,571	3,880
4.25	1,877	2,183	2,481	3,195	3,294	3,599	3,908
4.50	1,905	2,211	2,508	3,222	3,322	3,626	3,937
5.00	1,960	2,267	2,564	3,276	3,377	3,682	3,993
5.50	2,015	2,320	2,617	3,330	3,430	3,739	4,049

Table 24.23 After-Tax NPV8% Sensitivity to Platinum and Copper Prices

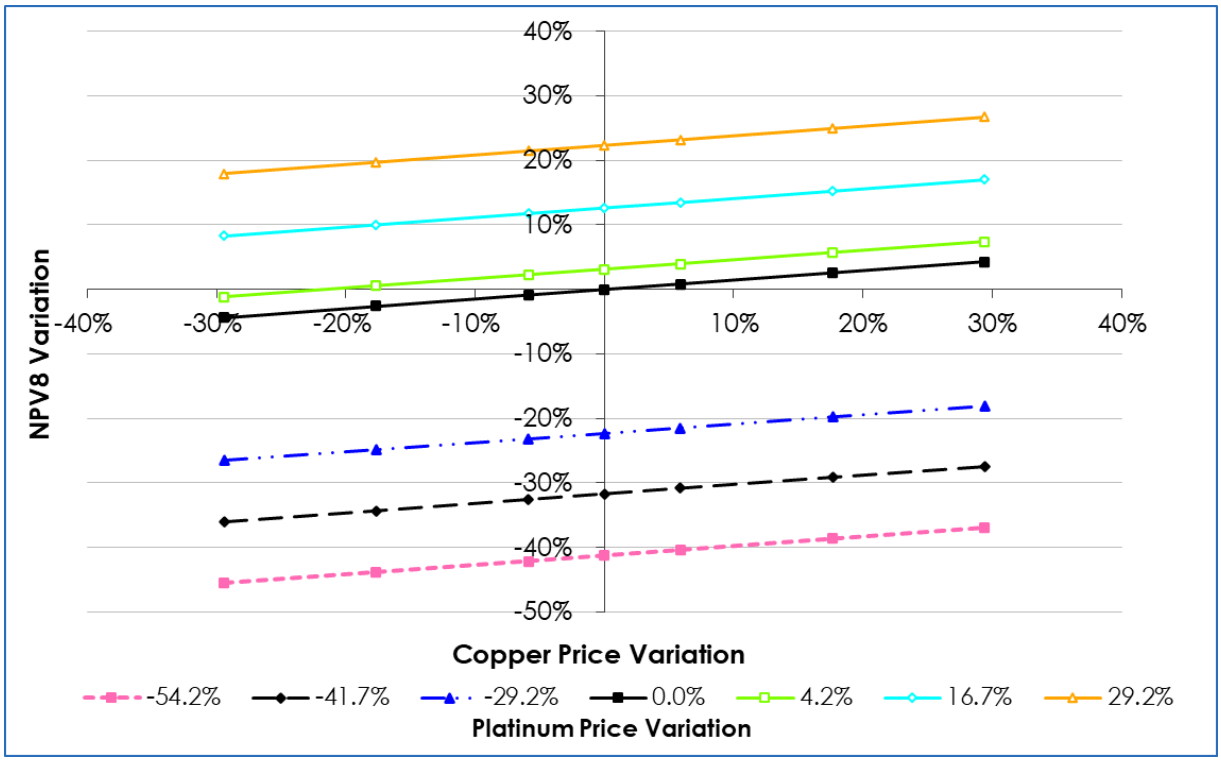
After-Tax NPV8% ($M)	Platinum Price ($/oz)						
Copper Price - $/lb	**550**	**700**	**850**	**1,200**	**1,250**	**1,400**	**1,550**
3.00	-46%	-36%	-27%	-4%	-1%	8%	18%
3.50	-44%	-34%	-25%	-3%	1%	10%	20%
4.00	-42%	-33%	-23%	-1%	2%	12%	21%
4.25	-41%	-32%	-22%	–	3%	13%	22%
4.50	-40%	-31%	-21%	1%	4%	14%	23%
5.00	-39%	-29%	-20%	3%	6%	15%	25%
5.50	-37%	-27%	-18%	4%	7%	17%	27%

Figure 24.16 Platinum and Copper Price Sensitivity



OreWin, 2025

Figure 24.17 Copper and Platinum Price Sensitivity



OreWin, 2025

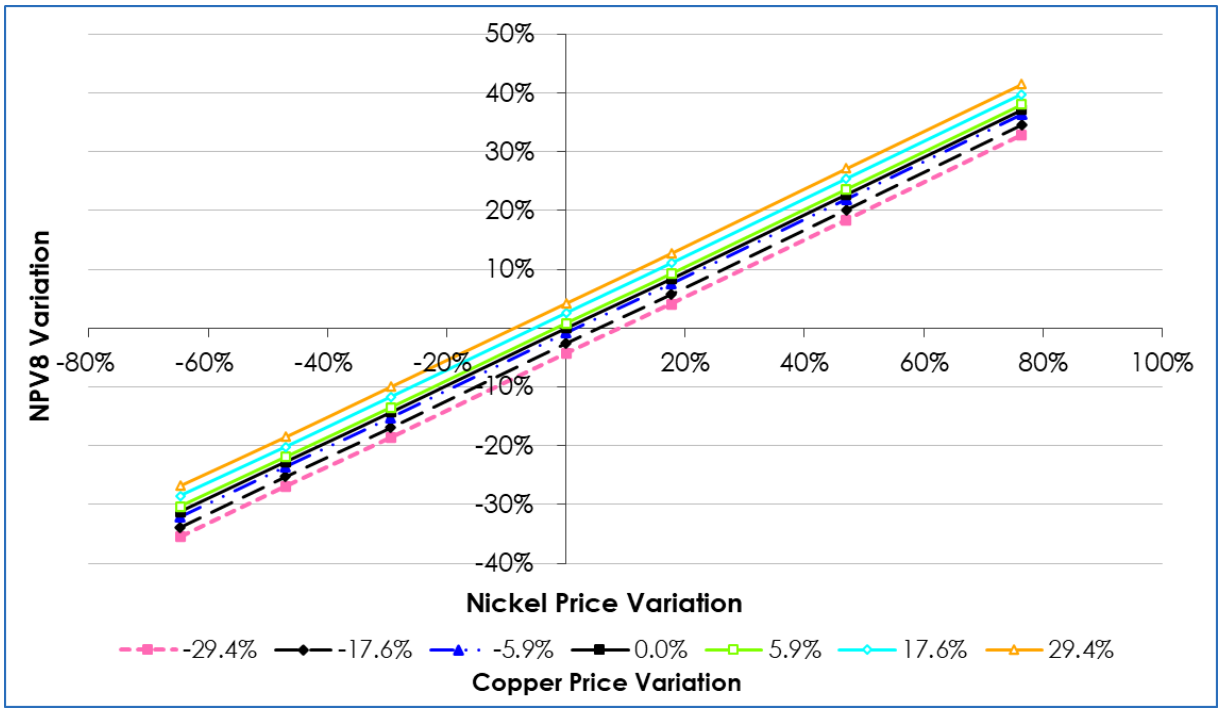
Table 24.24 After-Tax NPV8% Nickel and Copper Price Sensitivity

After-Tax NPV8% ($M)	Nickel Price ($/lb)						
Copper Price - $/lb	**3.00**	**4.50**	**6.00**	**8.50**	**10.00**	**12.50**	**15.00**
3.00	2,059	2,334	2,599	3,056	3,326	3,782	4,243
3.50	2,113	2,387	2,654	3,112	3,379	3,839	4,299
4.00	2,169	2,441	2,708	3,168	3,435	3,895	4,355
4.25	2,197	2,467	2,736	3,195	3,463	3,923	4,383
4.50	2,226	2,494	2,763	3,222	3,491	3,951	4,412
5.00	2,283	2,548	2,820	3,276	3,547	4,008	4,467
5.50	2,338	2,603	2,876	3,330	3,602	4,064	4,522

Table 24.25 After-Tax NPV8% Sensitivity to Nickel and Copper Prices

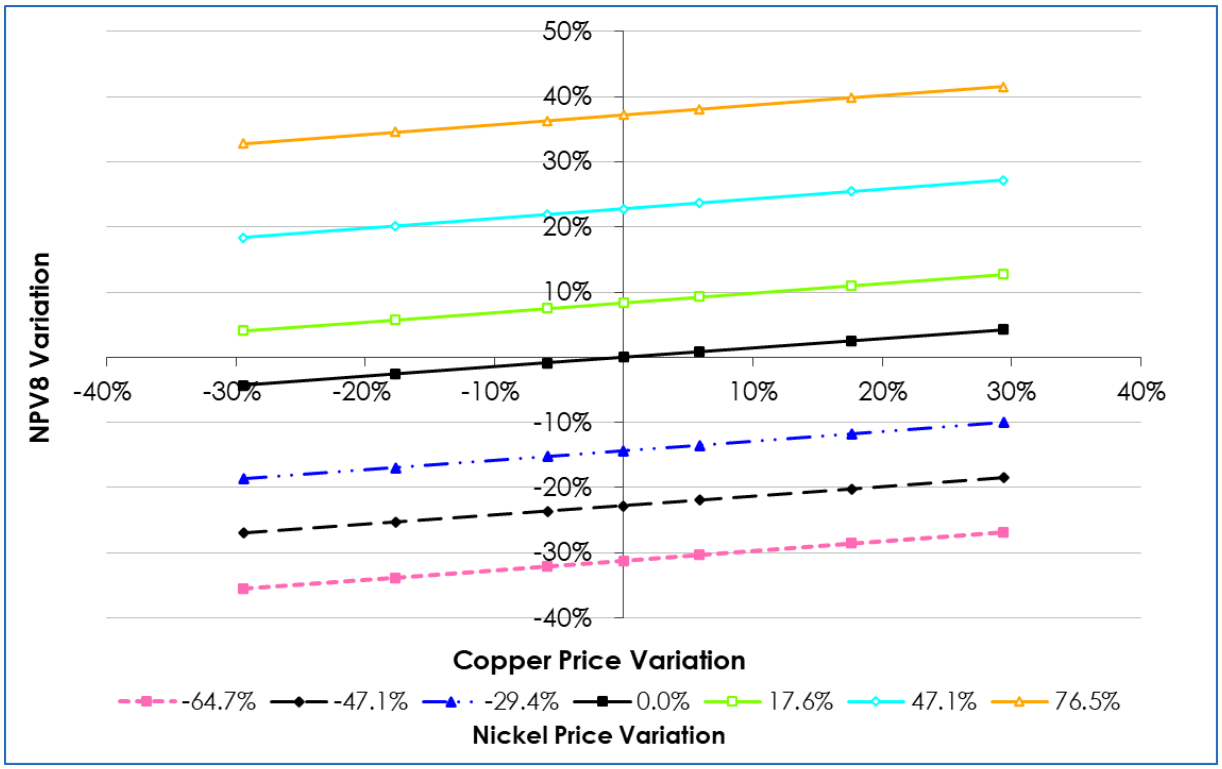
After-Tax NPV8% ($M)	Nickel Price ($/lb)						
Copper Price - $/lb	**3.00**	**4.50**	**6.00**	**8.50**	**10.00**	**12.50**	**15.00**
3.00	-36%	-27%	-19%	-4%	4%	18%	33%
3.50	-34%	-25%	-17%	-3%	6%	20%	35%
4.00	-32%	-24%	-15%	-1%	8%	22%	36%
4.25	-31%	-23%	-14%	–	8%	23%	37%
4.50	-30%	-22%	-14%	1%	9%	24%	38%
5.00	-29%	-20%	-12%	3%	11%	25%	40%
5.50	-27%	-19%	-10%	4%	13%	27%	42%

Figure 24.18 Nickel and Copper Price Sensitivity



OreWin, 2025

Figure 24.19 Copper and Nickel Price Sensitivity



OreWin, 2020

25 INTERPRETATION AND CONCLUSIONS

25.1 Platreef IDP25

Ivanplats should continue execution activities and to update the long-term development plans for Platreef. The Platreef IDP25 will provide the technical basis for Ivanplats to continue the Project financing and to continue marketing negotiations that have been undertaken to date. Examine the Platreef 2025 PEA and integrate it for the long-term strategic planning of Platreef.

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 Jeremy Witley 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. 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 south-west 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 Jeremy Witley 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 Jeremy Witley is of the opinion that the Mineral Resources for the Platreef Project, which have been estimated using drill core 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.

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 the mine plan. Based on this assessment, the Platreef Probable Mineral Reserve will support the Platreef 2025 FS with 129.7 Mt ore production for the life of the mine.

25.3.1 Mining Risks and Opportunities

25.3.1.1 Risks

The following is a list of potential mining risks for Platreef:

- Continued optimisation of the stope design and sequencing to improve the grade profile during the early years of production.
- Evaluating batch design to optimise cement and admixture usage minimising fill curing rates.
- Reducing the angle of the footwall and hanging wall in production stopes as a benefit of the definition drilling programme.
- Production and schedule constraints due to shaft pillar designs.
- Possible system downtime due to rehabilitation of silos.
- The amount of lining required in the ventilation raises once they are reamed.
- Timeliness of the definition drilling during the pre-production and early production stages.
- Logistics up and down Shaft 1 until Shaft 2 is commissioned in July 2028.

- Managing dilution and achieving the head grade in the ramp up.

25.3.2 Opportunities and Recommendations

The following is a list of potential opportunities Platreef may experience:

- Continued optimisation of the stope design and sequencing to improve the grade profile during the early years of production.
- Evaluating batch design to optimise cement and admixture usage minimising fill curing rates.
- Reducing the angle of the footwall and hanging wall in production stopes as a benefit of the definition drilling programme.

25.3.3 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.

25.4 Metallurgy

It is the opinion of the qualified person responsible for the metallurgical aspects of the Platreef 2025 IDP, Mrs Louise Lintvelt, 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 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.

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

A mini pilot plant campaign was conducted, primarily to produce bulk concentrate samples for downstream hydrometallurgical refining testwork and concentrate dewatering testwork. The added objective of deriving additional design data from the pilot runs was only partially achieved due to 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 testwork and concentrate dewatering testwork but did not provide comprehensive data to fully confirm the metallurgical performance as achieved in bench-scale locked cycle testwork.

The results for open circuit and locked cycle flotation testwork at SGS, Mintek and XPS over the period 2020–2022 (Post 2017 DFS) were found to fall within the ranges evidenced in the 2017 DFS variability testwork. This shows consistency between the various test campaigns. The locked cycle test results as derived during the 2017 DFS are considered adequate for deriving metallurgical performance projections.

The proposed Phase 2 comminution circuit which includes a multi-stage crusher circuit followed by ball milling is the preferred option for the concentrator based on the comminution characteristics and throughput requirements while also considering the future capacity expansion in Phases 3 and 4. The inclusion of an HPGR circuit for the tertiary crushing duty in Phase 2 was previously identified as an opportunity during the 2022 PDP FS which now has been incorporated in the process design.

The proposed flotation circuit is based on interpretation of the results obtained from the bench-scale open circuit and locked cycle 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 tailings handling and dewatering circuit is based on interpretation of the results obtained from the bench-scale thickening and filtration testwork. In Phase 2 a new tailings receiving, thickening and pressure filtration circuit, sized for the combined 4.07 Mtpa throughput, will be installed at the Rietfontein DTSF to treat the combined pumped concentrator tailings slurry. For Phase 2 a filtered tailings moisture content of 12–14% (w/w) will be targeted thus plate and frame pressure filters have been recommended for the Phase 2 tailings filtration duty.

The proposed phased concentrator capacity ramp up and modular design approach is also considered to be appropriate for this level of study.

25.5 Infrastructure

A number of mining projects are in the development phase on the Bushveld 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. The bulk power supply was installed and commissioned in August 2024.

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.

25.5.1 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.

25.5.2 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 IDP25

The Platreef IDP25 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 IDP25 will provide the basis for detailed planning of execution activities and to update the long-term development plans for Platreef.

26.2 Operational Drill Programme

Ivanhoe intends to implement a long-term delineation drilling programme as part of operational activities in support of detailed mine access and stope designs. Correct positioning of the footwall access drives will be important for efficient mining to ensure optimal grade and minimal ore losses. The drilling will need to be planned and sequenced at an operational level to ensure that sufficient geological information is available well ahead of mining.

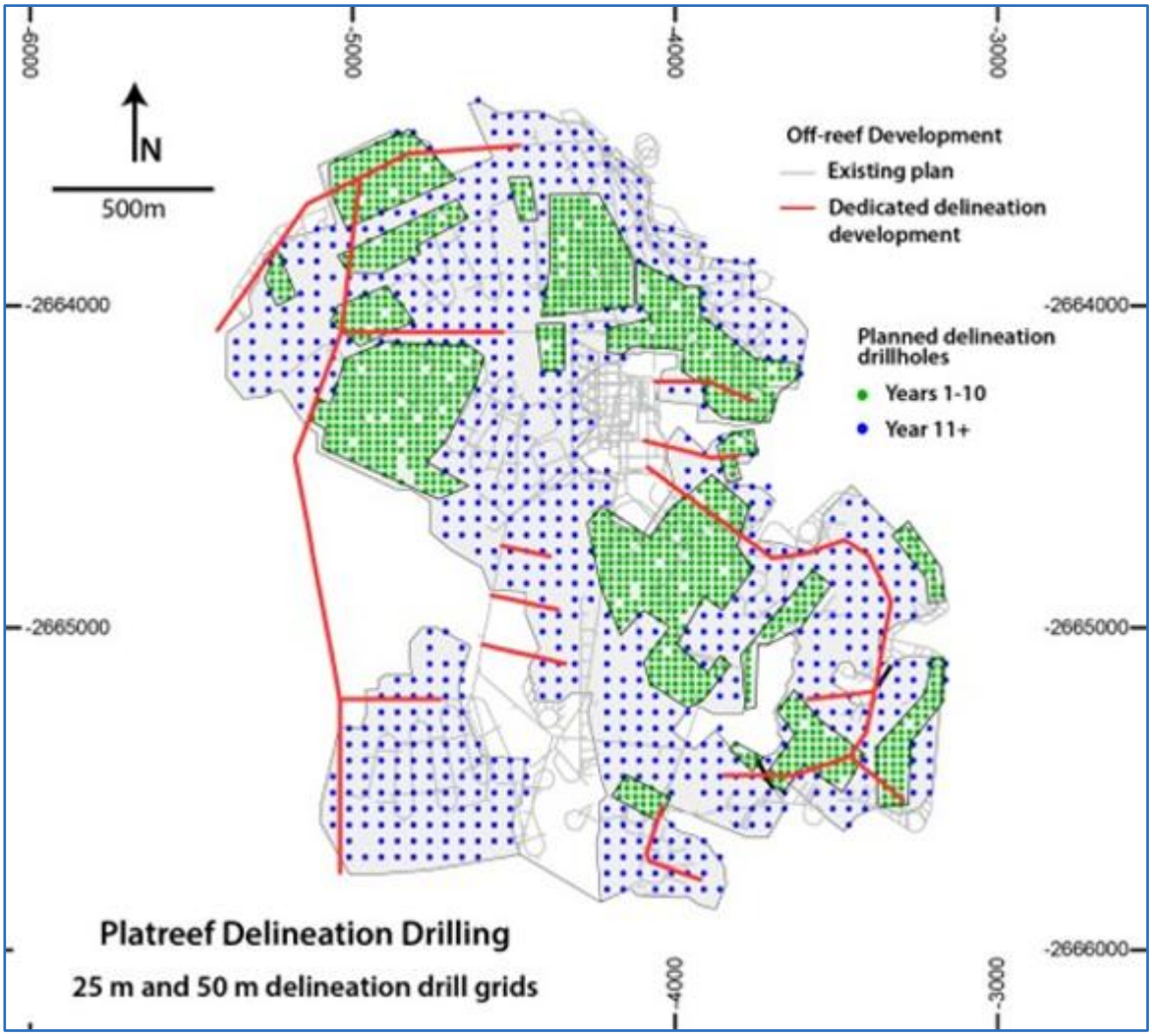
Ivanhoe's planned delineation drilling is separated by Ivanhoe into two periods:

- 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.

As numerous mining areas are developed in quick succession, the amount of drilling in earlier years will be relatively high, however it will stabilise as steady state production is reached and then reduce in later 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 IDP25.

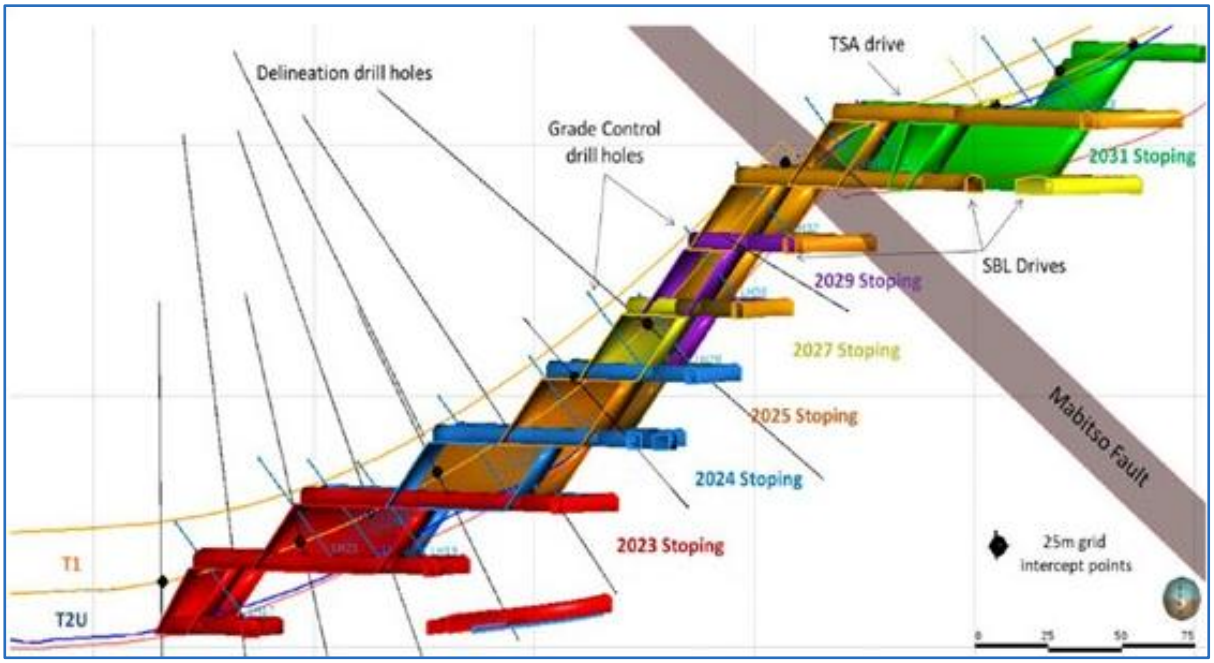
The QP has reviewed the estimates and given the assumptions in the proposed programme, considers the costs to be reasonable.

Figure 26.1 Proposed Delineation Drilling



Ivanhoe, 2022

Figure 26.2 Perspective View for Delineation Drilling



Ivanhoe, 2017

26.3 Resource Estimate

The QP recommends that the Mineral Resource model is updated when significant amounts of new data are available. Reconciliation with the updated Mineral Resource model using the results of the closely spaced underground delineation drilling programme will be important to understand the accuracy of the Indicated Mineral Resource on which the long-term plan is based.

26.4 Mineral Reserve Recommendations

The results and conclusions from IDP25 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.

No additional testwork is currently planned. The Phase 1 Project is currently being executed and includes the construction of an initial 0.77 Mtpa concentrator, which will be followed by a phased capacity expansion in Phase 2 and the future Phases 3 and 4. Further optimisation of the flow sheet during the Phase 1 commissioning, ramp up and optimisation phase should evaluate the potential for concentrate grade optimisation and improved 3PE+Au recovery by recycling the scavenger cleaner tailings to the rougher circuit. Additionally, the potential to conduct further trials of an SIBX reagent suite should be considered.

Open circuit and locked cycle flotation testing has consistently shown that as higher 3PE+Au concentrate grades are targeted (high upgrade ratio), the overall concentrate mass pull decreases, and consequently, the 3PE+Au recovery will be lower than when targeting a higher mass pull and lower concentrate grade (low upgrade ratio). However, it was also shown that if the circuits target an excessively high mass pull, the concentrate grades can be depressed without any notable gain in recovery. The final concentrate recovery and grade will need to be continuously optimised during full scale operations by targeting, controlling and optimising concentrate mass pull.

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).

27 REFERENCES

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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	<

Meaning	Symbol/Unit
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
Gigabit (10^9 bits)	Gbit
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
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
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolt-ampere reactive	kVAR
Kilowatt	kW
Litre	L
Pound	lb
Metre	m

Meaning	Symbol/Unit
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
Material take-off quantity (m^3)	MTO
Million (metric) tonnes per annum	Mtpa
Millivolts	mV
Megavolt ampere	MVA
Megawatt	MW
Moment Magnitude	M_W
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
Volt	V
Watts per m^2	W/m^2
Week (seven days)	wk
Wet metric tonnes	wmt
Weight per weight	w/w
Weight per volume	w/v

27.3 Glossary of Abbreviations and Terms

Table 27.2 Table of Abbreviations and Terms

Description	Abbreviation/Term
Atomic Absorption Spectroscopy	AAS
Adjustable current and time earth fault relay	ADIT
First of six hornfels xenoliths in the in the FWpnz domain	AHF
Annual Information Form	AIF
African Mineral Standards	AMIS
Drillhole prefix for the open pit programme, designated AMK at Macalacaskop 243 KR	AMK
Mineralised Zone A	AMZ
Air Natural Air Natural	ANAN
Ammonium Nitrate Fuel Oil	ANFO
American Petroleum Institute	API
Abrasion Resistant Lining	ARB
American Society of Mechanical Engineers	ASME
American Society for Testing and Materials	ASTM
Drillhole prefix for the open pit programme, designated ATS at Turfspruit 241 KR/Rietfontein 2 KS	ATS
Absolute Value Relative Difference	AVRD
Acoustic Televiewer	ATV
Australian Dollar	AUD
Absolute Value Relative Difference	AVRD
Bulk Air Cooler	BAC
Broad-Based Black Economic Empowerment	B-BBEE
K2014089596 (South Africa) (RF) Proprietary Limited	BEE Co
Bare Copper Earth Wire	BCEW
Bushveld Complex	BC
Second of six hornfels xenoliths in the in the FWpnz domain	BHF
Bikkuri Model	BIK
Basal Melagabbronorite	BMGN
Base Metal Sulfide	BMS
Mineralised Zone B	BMZ
Diamond drill core 36.5 mm diameter	BQ
Backscatter Electron Images	BSE
Canadian Centre for Mineral and Energy Technology	CANMET
Collision Avoidance System	CAS
Condenser Cooling Towers	CCT
Closed-Circuit Television	CCTV

Description	Abbreviation/Term
Compact Disc Read-Only Memory	CD-ROM
Third of six hornfels xenoliths in the in the FWpnz domain	CHF
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Carboxymethyl cellulose	CMC
Chinese Yuan Renminbi	CNY
Co-Operative Governance, Human Settlements and Traditional Affairs	CoGHSTA
K2014043822 (South Africa) (RF) Proprietary Limited	Community TrustCo
Mineralised Zone C	CMZ
Coronavirus disease	COVID-19
Chartered Professional	CP
Cemented Paste Fill	CPF
Clinopyroxenite	CPX
Coarse Reject Duplicates	CRD
Cemented Rock Fill	CRF
Certified Reference Materials	CRM
Comments and Response Report	CRR
Corporate Social Responsibility	CSR
Coefficients of Variation	CV
Critical Zone	CZ
Dry Bulb	DB
Diamond Drillhole	DDH
Drift and Fill	DF
Depth of Failure	DF
Definitive Feasibility Study	DFS
Electrical, Control and Instrumentation	EC&I
Fourth of six hornfels xenoliths in the in the FWpnz domain	DHF
Department of Mineral Resources	DMR
Department of Mineral Resources and Energy	DMRE
Mineralised Zone D	DMZ
Diameter Nominal	DN
Direct-On-Line	DOL
Diesel Particulate Matter	DPM
Democratic Republic of Congo	DRC
Department of Rural Development and Land Reform	DRDLR
Dewatering Technology Centre	DTC
Dry Tailings Storage Facility	DTSF
Department of Water and Sanitisation	DWS

Description	Abbreviation/Term
Earnings Before Interest and Taxes	EBIT
Environmental Authorisation	EA
Electrode-array-focussed resistivity	EAL
Engineering Council of South Africa	ECSA
Enterprise Development	ED
Exploratory Data Analysis	EDA
Energy Dispersive Spectroscopy	EDS
Effective Grinding Length	EGL
Fifth of six hornfels xenoliths in the in the FWpnz domain	EHF
Equivalent Linear Overbreak and Sloughing	ELOS
Environmental Management Programme	EMP
K2014043829 (South Africa) (RF) Proprietary Limited	Employee TrustCo
Electronic Multi-Shot	EMS
Mineralised Zone E	EMZ
K2014043815 (South Africa) (RF) Proprietary Limited	EntrepreneurCO
Expression of Interest	EOI
Engineering, Procurement, and Construction Management	EPCM
Early Streamer Emission	ESE
Environmental and Social Impact Assessment	ESIA
Euro	EUR
Fresh Air Pass	FAP
Fresh Air Rise	FAR
Footwall Assimilated Zone	FAZ
Feldspathic clinopyroxenite	FCPX
Froth Carry Rate	FCR
Fast-opening Filter Press	FFP
Fellow of the Geological Society of South Africa	FGSSA
Feldspathic harzburgite	FHA
Sixth of six hornfels xenoliths in the in the FWpnz domain	FHF
Fédération Internationale des Ingénieurs-Conseils	FIDIC
Forward Looking Statement	FLS
Front End loader	FEL
Flow Moisture Point	FMP
Mineralised Zone F	FMZ
Feasibility study	FS
Fellow of the Australasian Institute of Mining and Metallurgy	FSAIMM
Footwall	FW
Fall-Waveform-Sonic	FWFS

Description	Abbreviation/Term
General and Administration	G&A
British Pound sterling	GBP
Grade Shell Codes	GCODE
Gas Insulated Substation	GIS
Giant Mottled Anorthosite	GMA
Great North Fault zone	GNF
Global Positioning System	GPS
Gravity Transfer Sumps	GTS
Granite Vein	GV
Hemicellulose	HC
Heritage Impact Assessment	HIA
High-Density Polyethylene	HDPE
Historically Disadvantaged South Africans	HDSA
High Grade	HG
Human-Machine Interface	HMI
High-Pressure Grinding Roll	HPGR
Diamond drill core 63.5 mm diameter	HQ
Inductively Coupled Plasma	ICP
Inductively Coupled Plasma - Mass Spectrometry	ICP-MS
Inductively Coupled Plasma - Optical Emission Spectroscopy	ICP-OES
Internal Diameter	ID
Integrated Development Plan	IDP
International Electrotechnical Commission	IEC
Induced Polarisation	IP
Ingress Protection	IP
Internet Protocol	IP
Interim Protection of Informal Land Rights Act, 1996	IPILRA
Independent Power Producer	IPP
Internal Rate of Return	IRR
International Organization for Standardization	ISO
Integrated Water Use Licence	IWUL
Japanese Yen	JPY
Joint Venture	JV
Joint Water Forum	JWF
Low Angled Feature	LAF
Local Area Network	LAN
Locked Cycle Test	LCT
Lower Critical Zone	LCZ

Description	Abbreviation/Term
Light Delivery Vehicle	LDV
Light-Emitting Diode	LED
Local Economic Development	LED
Limpopo Department of Economic Development, Environment and Tourism	LEDET
Longevity Extension Factor	LEF
Low Grade	LG
Low Angle Granite Vein	LGV
Load-Haul-Dump	LHD
Longhole Open Stopes	LHOS
Longhole Stoping	LHS
Lip Loading	LL
Life-of-Mine	LOM
Longevity Reduction Factor	LRF
Low Voltage	LV
Lower Zone	LZ
Mottled anorthosite	MA
Member of Australasian Institute of Mining and Metallurgy	MAusIMM
Motor Control Centre	MCC
Unique integer code assigned to stratigraphic units	MCODE
Merensky Cyclic Unit	MCU
Management Discussion and Analysis	MD&A
Medium Grade	MG
Mokopane Interested and Affected Communities Committee	MIACC
Mokopane Interested and Affected Communities Development Forum	MIACDF
Mogalakwena Local Municipality	MLM
Two stages of ball milling in series and flotation	MMF
Mini Pilot Plant	MPP
Mineral and Petroleum Resources Development Act 28 of 2002	MPRDA
Mineral and Petroleum Titles Registration Office	MPTRO
Multi-Purpose Vehicle	MPV
Mining Right	MR
Mining Right Application	MRA
Mean Sea Level	MSL
Maptek's Stope Optimiser	MSO
Magnetic Susceptibility	Msus
Main Transmission Station	MTS
Mine Technical Services	MTS

Description	Abbreviation/Term
Medium Voltage	MV
Main Zone	MZ
Marginal Zone Norites	MZN
Non-Acid Forming	NAF
National Association of Testing Authorities (Australia)	NATA
Nominal Bore	NB
Norite Cycles	NC
Norite Cyclic Unit 2	NC2
National Environmental Management Act 107 of 1998	NEMA
Neutral Earthing Resistor	NER
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
Norwegian Krone	NOK
Net Positive Suction Head	NPSH
Net Present Value	NPV
Net Present Value at 8% Discount Rate	NPV8%
Diamond drill core 47.6 mm diameter	NQ
Diamond drill core 50.5 mm diameter	NQ2
Diamond drill core 44.9 mm diameter	NQ3
North Seeking Gyro	NSG
Net Smelter Return	NSR
National Water Act 36 of 1998	NWA
Original Equipment Manufacturers	OEM
Overhead line	OHL
Ordinary Kriging estimation method	OK
Olivine clinopyroxenite	OLCPX
Olifants River Water Resources Development Project	ORWRDP
Occupational Health and Safety Assessment Series	OHSAS
Optical Televiewer	OTV
Pipe Anchor	PA
Potentially Acid Forming	PAF
Programme for Accreditation of Laboratories in Canada	PALCAN
Pay As You Earn	PAYE
Positive Displacement	PD
Phased Development Plan	PDP

Description	Abbreviation/Term
Preliminary Economic Assessment	PEA
Power Factor Correction	PFC
Prefeasibility study	PFS
Platinum Group Element	PGE
Platinum Group Mineral	PGM
Programmable Logic Controller	PLC
Pyroxenite-Norited-Zone	PNZ
Public Participation Processes	PPP
Diamond drill core 85 mm diameter	PQ
Prospecting Right	PR
Point Resistance	PR
Pipe Support	PS
Particle Size Distribution	PSD
Photovoltaic	PV
Polyvinyl chloride	PVC
Quality Assurance and Quality Control	QA/QC
Qualified Person	QP
Return Air Rise	RAR
Reverse Circulation	RC
Radiating Cable Network	RCN
Rare Earth Elements	REE
Remote Inputs and Outputs	RIO
Rustenburg Layered Suite	RLS
Chinese Renminbi (also called Yuan)	RMB
Rock Mass Rating	RMR
Ring Main Unit	RMU
Run-of-Mine	ROM
Reasonable Prospects for Eventual Economic Extraction	RPEE
Rock Quality Designation	RQD
Republic of South Africa	RSA
South African Council for Natural Scientific Professions	SACNASP
South African Institute of Mining and Metallurgy	SAIMM
Semi-Autogenous Grinding	SAG
South African Heritage Resources Agency	SAHRA
South African National Roads Agency Limited	SANRAL
South African National Standards	SANS
South African Revenue Services	SARS
Self-Drilling Anchor	SDA

Description	Abbreviation/Term
System for Electronic Document Analysis and Retrieval (SEDAR) is a filing system developed for the Canadian Securities Administrators	SEDAR
Swedish Krona	SEK
Scanning Electron Microscopy	SEM
Steep Granite Vein	SGV
Safety, Health, Environment and Quality	SHEQ
Social Impact Assessment	SIA
Stay-In-Business	SIB
Sodium isobutyl xanthate	SIBX
Sodium isopropyl xanthate	SIPX
Single Line Diagram	SLD
Social and Labour Plan	SLP
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
Standard Reference Material	SRM
Species of Special Concern	SSC
Surface Use and Cooperation Agreement	SUCA
Steel Wire Armoured	SWA
Short-Wave Infrared	SWIR
Tennis Ball Marker	TBM
Total Cost Package	TCP
Turfspruit Cyclic Unit	TCU
Time Domain Reflectometers	TDR
Total Dissolved Solids	TDS
Totally Enclosed, Fan-Cooled	TEFC
Total Earth Pressure Cells	TEPC
Thickener Hydraulic Unit Area	THUA
Thabazimbi-Murchison Lineament	TML
Transportable Moisture Limit	TML
Diamond drill core 60.3 mm diameter	TNW
Transfer Pump Stations	TPS
Technical Report	TR
Tailings Storage Facility	TSF
Thickener Underflow Unit Area	TUFUA

Description	Abbreviation/Term
Unconfined Compressive Strength	UCS
Underground Distribution System	UDS
Upper Critical Zone	UCZ
Undifferentiated Contaminated Zone	UDCZ
Underground	UG
Upper Group 2	UG2
Unemployment Fund	UIF
Drillhole prefix for the underground programme, designated UMT (including Bikkuri); nearly all drilling is on Turfspruit 241 KR.	UMT
United States Dollar	USD
Unconfined Tensile Strength	UTS
Utility Vehicle	UV
Upper Zone	UZ
Vertical Crater Retreat	VCR
Very Early Smoke Detection Apparatus	VESDA
Vibrating Grizzly Feeder	VGR
Ventilation On Demand	VOD
Variable Speed Drive	VSD
Vertical Seismic Profile	VSP
Wet Bulb	WB
Work Breakdown Structure	WBS
Waste Management Licence	WML
Water Use Licence	WUL
Waste Water Treatment Works	WWTW
Waste Water Treatment Plant	WWTP
Cross linked polyethylene	XLPE
X-Ray Fluorescence	XRF
South African Rand	ZAR
Zebediela fault region	ZEBED