OreWin
Independent Mining Consultants

Ivanhoe Mines Ltd.

Platreef 2017 Feasibility Study

September 2017

Job No. 15002

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Stantec

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Title Page

Project Name:	Platreef Project
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Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	22 April 2016
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	22 April 2016
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Effective Date of Mineral Reserves:	24 May 2017

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Signature Page

Project Name:	Platreef Project
Title:	Platreef 2017 Feasibility Study
Location:	Limpopo Province, Republic of South Africa
Effective Date of Technical Report:	4 September 2017

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1 SUMMARY

1.1 Introduction

The Platreef 2017 Feasibility Study (Platreef 2017 FS) is an Independent NI 43-101 Technical Report prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe Mines Ltd. (Ivanhoe) on the Platreef nickel-copper-gold–platinum group element (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 2016 Resource Technical Report on the Platreef Project that have formed the basis of the Platreef 2017 FS, which summarises the current Ivanhoe development strategy for the Platreef Project.

Figure 1.1 Platreef Project Location

Figure supplied by Ivanhoe, 2014.

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

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

The South African Mining Right (MR) LP30/5/2/2/1/10067MR and Prospecting Right (PR) LP30/5/111/2/740PR collectively form the Platreef Project area.

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

A Joint Venture (JV) with Atlatsa Resources Corporation covers PR LP30/5/111/2/740PR.

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

On 6 June 2013, Ivanhoe filed a Mining Right Application (MRA) with South Africa's Department of Mineral Resources (DMR). In conjunction with the MRA, and in compliance with South African ownership requirements under the Mining Charter, Ivanhoe also announced that the Platreef Project ownership structure would be modified to include a B-BBEE partner. The B-BBEE partners acquired a 26% interest in the Platreef Project through B-BBEE Special Purpose Vehicle (SPV), a private company incorporated in South Africa that represents the interests of local communities, local entrepreneurs and employees.

1.2 Mineral Tenure and Surface Rights

Ivanplats obtained a MR in terms of Section 22 of the Mineral and Petroleum Resources Development Act 28 of 2002 (MPRD Act) from the DMR in May 2014, which was effected through notarial execution in November 2014, and 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) in June 2014.

A 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. The MR will continue to be in force until 3 November 2044.

The EA has since been amended twice: LEDET granted the first amendment in August 2014, and the second amendment in July 2015. A third amendment application is currently under review with the DMR.

Figure 1.2 shows the locations of the townships that have developed within the farming areas, including on farms that are outside the Platreef Project area.

Figure 1.2 Project Location and Farm Boundaries

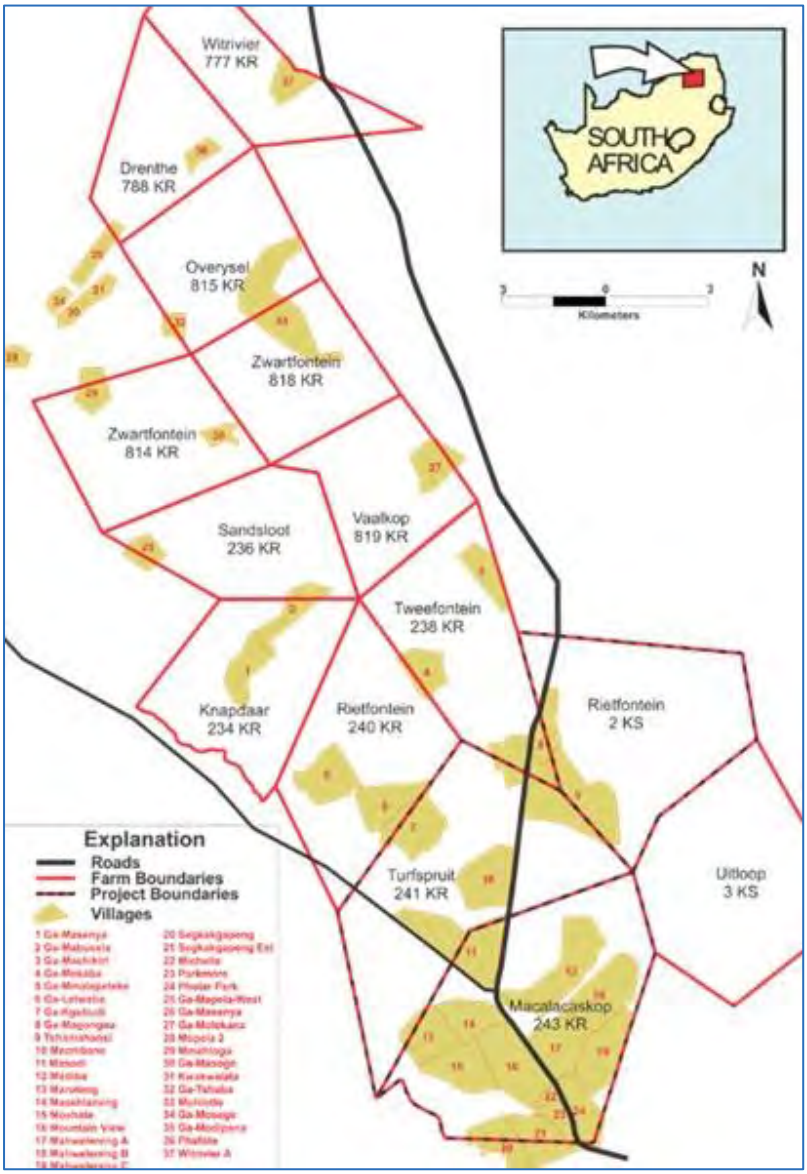


Figure supplied by Ivanhoe,2014.

1.2.1 Macalacaskop 243 KR and Turfspruit 241 KR

On 6 June 2013, Ivanhoe applied for a MR in respect of PGEs and all associated metals and minerals mined out of necessity and convenience together with the PGEs, over the Macalacaskop 243 KR and Turfspruit 241 KR farms. The MR application was accepted by the DMR on 17 July 2013, and granted on 30 May 2014; on the eve of the expiry of the PR, which was registered in the South African Mineral and Petroleum Titles Registration Office (MPTRO) on 9 February 2006, under registration number 55/2006 PR.

Ivanplats and the Regional Manager: Limpopo Region of the DMR notarially executed a MR on 4 November 2014 with DMR reference number LP30/5/2/2/1/10067MR ("the MR"). By virtue of the MR, Ivanplats is the sole and exclusive holder of the mining title in and to PGEs, gold, silver, nickel, copper, iron, vanadium, cobalt and chrome in respect of the farms Macalacaskop 243 KR and Turfspruit 241 KR (excluding areas comprising graveyards, built-up areas and protected areas). The MR commenced on 4 November 2014 and, unless cancelled or suspended in terms of clause 13 of the MR and/or Section 47 of the MPRD Act, will continue in force for a period of 30 years ending on 3 November 2044, renewable for further periods, each of which may not exceed 30 years at time.

In terms of Section 25(2)(a) of the MPRD Act, the holder of a MR 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 MR). The MPRD Act does not require that a right must be registered within a certain period of time; only that it must be lodged for registration. The MR was lodged for registration on 3 December 2014. Following a number of queries from the MPTRO, the MR was successfully registered 3 February 2017.

1.2.2 Rietfontein

Atlatsa Resources Corporation (Atlatsa; formerly Anooraq Resources Corporation) through its South African Subsidiary Plateau Resources Limited, holds exclusive prospecting rights for base minerals and precious metals on the Rietfontein 2 KS farm. Rietfontein 2 KS farm has an area of 2,878 ha. The mineral lease is identified as MPT76/2007PR. The PR was valid for a five-year period, and was to expire on 27 November 2011. Atlatsa received a three year renewal for their prospecting right on 2 October 2014. This PR is valid until 1 October 2017. This renewal is the last renewal for Atlatsa. In order to retain the title, they would have to apply for a mining right before expiry of the prospecting right, assuming of course that a case can be made for a MR. To date, Ivanhoe has advised that to the best of its knowledge, a MR has not been issued over Rietfontein. The JV is valid until the expiry of the PR.

1.2.3 Surface Rights

Surface rights over the Macalacaskop 243 KR, Turfspruit 241 KR and Rietfontein 2 KS farms are owned by the State and held in trust for the respective communities. Rights to prospect and mine the land are granted by the State.

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.

1.3 History and Exploration

1.3.1 Exploration

Early exploration on the Platreef mineralisation dates back to the 1960s. Subsequently, Rustenberg Platinum Holdings Limited, a wholly-owned subsidiary of Anglo American Platinum Corporation, began exploration on the Platreef Project in the 1970s. No data from either of these programmes were available for preparation of the Platreef 2017 FS.

Ivanhoe acquired a PR for both Turfspruit 241 KR and Macalacaskop 243 KR farms in February 1998 and subsequently entered into a JV with Atlatsa over the Rietfontein 2 KS farm in 2001. The JV agreement was updated in 2009.

The initial exploration focus was on delineation of mineralisation that could support open pit mining. From 2003 to 2007, Ivanhoe undertook studies involving concentrator/smelter options, metallurgical test work, 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 and 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 test work, geotechnical and hydrological investigations, seismic survey, social and environmental impact assessments, mineralogical studies, Mineral Resource and Mineral Reserve estimation and subsequent updates, a preliminary economic assessment, and a prefeasibility study.

1.3.2 Platreef 2015 PFS and Platreef 2014 PEA

In January 2015, Ivanhoe completed the Platreef 2015 PFS. Ivanhoe's development plan for the Platreef Project considers three phases of underground mining and concentrator expansion that were identified in the Platreef 2014 PEA. The Platreef 2014 PEA is a PEA as defined in NI 43-101 Standards of Disclosure for Mineral Projects, and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability but have reasonable prospects for eventual economic extraction. The development scenarios that were identified in the Platreef 2014 PEA are:

- Phase 1 Concentrator 4 Mtpa
- Phase 2 Concentrator 8 Mtpa
- Phase 3 Concentrator 12 Mtpa

Additional details of the Platreef 2015 PFS and the Platreef 2014 PEA are described in Section 6 History. Copies of the Platreef 2015 PFS and Platreef 2014 PEA can be downloaded from www.sedar.com.

1.4 Geology and Mineralisation

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

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

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

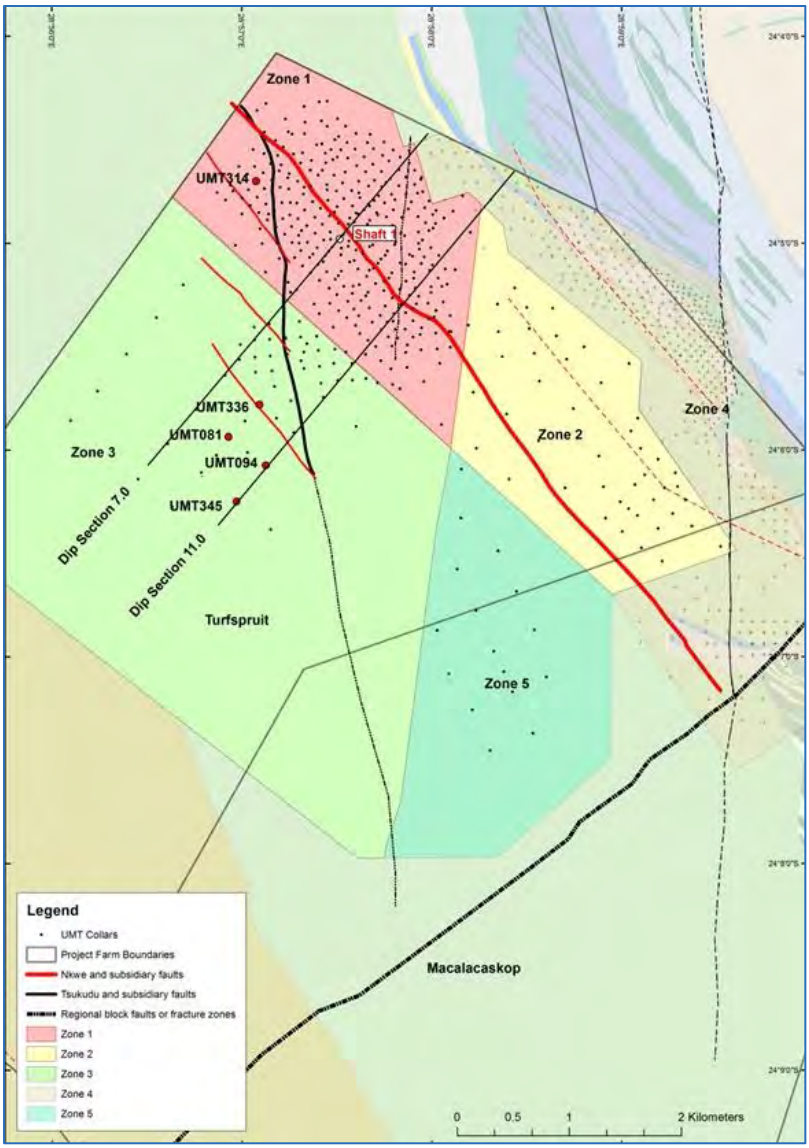
Within the TCU, high-grade PGE–Ni–Cu mineralisation is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, bound by chromitite stringers and containing coarse-grained sulphides; this is known as T2, with the mineralised portion referred to as T2MZ. The 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 sulphides (T1MZ) occurs near the top of the T1 feldspathic pyroxenite.

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

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

Figure 1.3 Project Exploration Zones Plan



Courtesy Ivanhoe, 2016

A revised 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 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.

Six faults are used to define seven fault blocks for the refined 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 NNW-SSE. These folds have subsequently been gently refolded with the minor fold axis (F2) trending ENE-WSW. The F1 folds are responsible for the apparent flattening of the Platreef basinward, the Macalacaskop syncline, the so called "T1-trough" and the overall 50° dip to the southwest along the open-pit fold limb. The minor folds are responsible for domes and basins within the larger folds such as the Bikkuri dome.

Broadly, Zone 1 or the 'Flatreef' can be envisioned 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.

Pyrrhotite, pentlandite and chalcopyrite occur as interstitial sulphides in the TCU lithologies. PGEs are mainly present as PGE–sulphides and PGE–Bi–Te and PGE–As alloys, that are fine-grained (< 10 µm) and may occur within base metal sulphides, 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 historic 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. 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/sheared zones.

1.5.1 Collar Surveys

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

1.5.2 Downhole Surveys

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

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

1.6 Sample Preparation, Analyses and Security

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

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

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

1.7 Data Verification

Amec Foster Wheeler E&C Services Inc (Amec Foster Wheeler) reviewed the sample chain of custody, quality assurance and quality control QA/QC procedures, and qualifications of analytical laboratories. In addition, Amec Foster Wheeler audited the assay database, core logging, and geological interpretations. Based on these reviews, Amec Foster Wheeler considers that the data are acceptable to support Mineral Resource estimation.

1.8 Mineral Resource Estimates

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

1.8.1 UMT-TCU Selectively Mineable Model

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

1.8.2 Additional Mutually-Exclusive Mineral Resource Models

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

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

1.9 Mineral Resource Statements

Table 1.1 summarises the combined Platreef Mineral Resources that are amenable to underground selective mining methods (UMT-TCU, UMT-BIK, UMT-FW). Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources are reported on a 100% ownership basis. The Qualified Persons for the estimate are Dr Harry Parker RM SME, and Mr Tim Kuhl, RM SME, both Amec Foster Wheeler employees. Mineral Resources have been estimated using core drill data, have been performed to industry best practices (CIM, 2003), 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 1.1 Platreef Mineral Resources; All Mineralised Zones (Base Case Highlighted)

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

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

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.

Amec Foster Wheeler reviewed provisional results of a seismic survey conducted during 2014 and completed some twined drillhole data analyses. Amec Foster Wheeler 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. Amec Foster Wheeler 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.

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 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.

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

1.11 Mineral Reserves

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in Section 14 of the Platreef 2017 FS. Only Indicated Mineral Resources have been used for determination of the Probable Mineral Reserve.

The Mineral Resource block model also includes the 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.

Prior to beginning stope design work and associated Mineral Reserve calculations, Stantec evaluated NSR cut-off values. The evaluation used updated mining cost estimates provided by Stantec as well as updated processing and G&A costs provided by Ivanhoe.

Economic cut-offs were established for each mining method and varied from $47.71–$58.53 per tonne, excluding capital recovery and profit margin. For the production schedule and Mineral Reserve, a declining cut-off was chosen.

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

A definitive mine plan based on detailed stope layouts supports the Mineral Reserve. Due to irregularities in the geometry of the mineralised zones, not all material meeting cut-off grade can be mined without incurring some dilution. Due to inefficiencies in final mining recovery from the stopes, small amounts of mineralised material are lost during final stope cleanout, and additional losses may occur in transit from the stopes to the mill. Hence, a mining recovery factor is applied to the diluted resources to account for these losses.

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

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

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

Table 1.2 and Table 1.3 show the total diluted and recovered Probable Mineral Reserve for Platreef.

Table 1.2 Platreef Probable Mineral Reserve – Tonnage and Grades as at 24 May 2017

Method	Ore (Mt)	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Ore Development	11.1	159.9	1.96	2.05	0.30	0.14	4.45	0.17	0.35
Longhole	93.1	152.1	1.88	1.95	0.29	0.13	4.25	0.16	0.33
Drift-and-Fill	20.4	182.0	2.28	2.23	0.37	0.15	5.03	0.18	0.37
Total	124.7	157.7	1.95	2.01	0.30	0.14	4.40	0.17	0.34

1. Mineral Reserves have an effective date of May 24, 2017. The Qualified Person for the estimate is Jon Treen (Stantec), P. Eng., with Professional Engineers of Ontario.
2. A declining NSR cut-off of $155/t to $80/t was used for the Mineral Reserve estimates.
3. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
4. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
5. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

Table 1.3 Platreef Probable Mineral Reserve – Contained Metal as at 24 May 2017

Method	Ore (Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Ore Development	11.1	0.7	0.7	0.1	0.05	1.6	42	85
Longhole	93.1	5.6	5.8	0.9	0.4	12.7	333	681
Drift-and-Fill	20.4	1.5	1.5	0.2	0.1	3.3	83	167
Total	124.7	7.8	8.0	1.2	0.5	17.6	457	932

1. Mineral Reserves have an effective date of May 24, 2017. The Qualified Person for the estimate is Jon Treen (Stantec), P. Eng., with Professional Engineers of Ontario.
2. A declining NSR cut-off of $155/t to $80/t was used for the Mineral Reserve estimates.
3. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
4. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
5. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

Based on the cut-off grade and mining criteria applied to the Platreef resource model, the Probable Mineral Reserve will support a 30-year mine life at a production rate of 4 Mtpa.

1.12 Geotechnical Investigations

The primary aim of the Platreef 2017 FS investigation was to increase the level of confidence in the current geotechnical database and to undertake various analyses, based on data from the mine site and laboratory testing, to provide geotechnical design parameters and optimise the mine design going forward.

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

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

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

1.13 Mining

1.13.1 Mining Method and Mine Design

The Platreef Project is designed based on highly-mechanised Longhole Stoping and Drift-and-Fill mining methods. The mine plan has been improved over the Platreef 2015 PFS by targeting higher-grade zones. This has increased the overall Life of Mine (LOM) grade and the grade profile in the early production years. It has also modified the top-cut development approach to dramatically reduce the amount of ore development on the critical path, and the ratio of ore development to Longhole Stoping.

The Mineral Reserve at 4 Mtpa only uses a third of the resource above an $80/t NSR cut-off. This allows the Platreef Project the opportunity to ramp up production. The rock handling infrastructure is designed for a total ore capacity of 6 Mtpa, so the ability to ramp the production up is available.

Primary access to the mine will be by a 1,104 m deep, 10 m diameter production shaft (Shaft 2). Secondary access to the mine will be via a 980 m deep, 7.25 m diameter ventilation shaft (Shaft 1). During mine production, both shafts will also serve as ventilation intakes. Three additional ventilation exhaust raises (Ventilation Raise 1, 2, and 3) are planned. Ventilation Raise 1 will be a 950 m deep, 6 m diameter raise located near the centre of the mining area and adjacent to the two intake shafts. Ventilation Raise 2 will be an 800 m deep, 6 m diameter raise located near the northern edge of the mining area. Ventilation Raise 3 will be a 725 m deep, 6 m diameter raise located near the southern edge of the mining area.

Three main access levels will be established as primary haulage levels. These are the 750, 850, and 950 Haulage Levels. Figure 1.4 shows the proposed shaft and raise locations and the main access levels in an elevated view (looking northeast). Mining access ramps will connect the haulage levels with the mining sublevels and other infrastructure. The mining sublevels will be developed from the ramps at regular vertical intervals in the production areas. Drilling and extraction levels for stopes will be driven from the sublevels. Ventilation raises and ore passes will also connect the sublevels with the main haulage levels.

Figure 1.4 Mine Access Layout

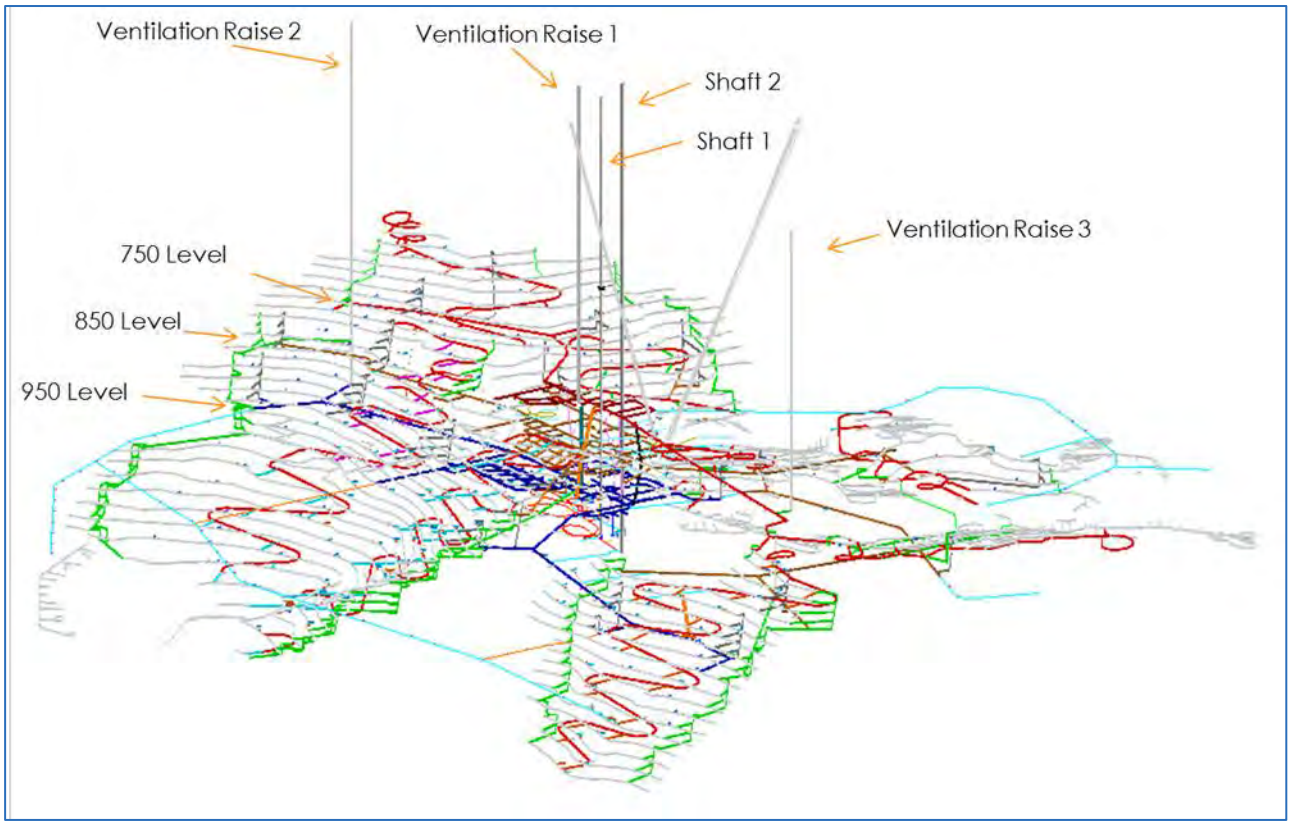


Figure by Stantec 2016.

The main mining methods will be Longhole Stoping and Drift-and-Fill mining. These methods provide a safe, mechanised, and productive mining plan. For the Platreef 2017 FS, 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. Also, this change increases the ratio of Longhole Stope tonnage to development tonnage when compared to the Platreef 2015 PFS. This has a positive impact both on productivity and cost. Figure 1.5 is an isometric view of the mining areas by method (looking north-east).

Figure 1.5 Elevated View of Mining Areas by Method

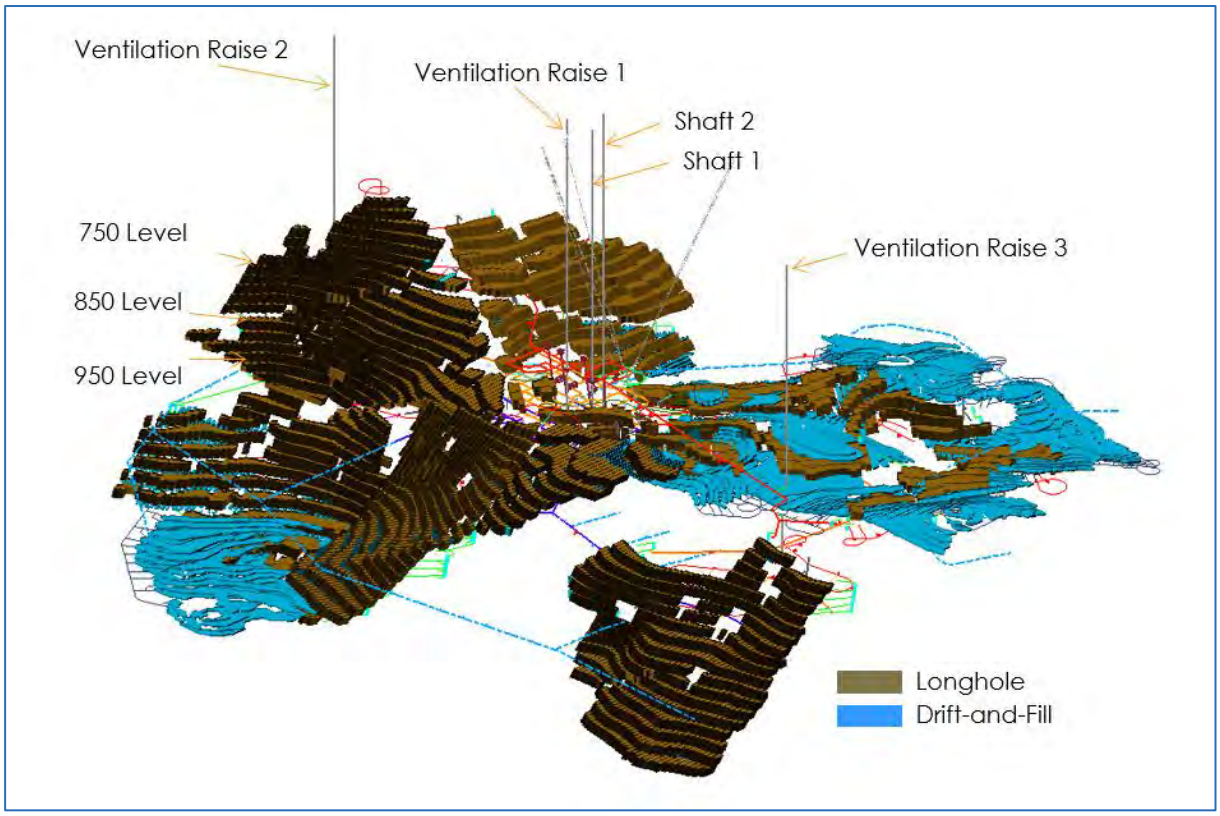


Figure by Stantec 2016.

1.13.2 Mine Development Plan

Mine development has been broken down into four main phases.

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

- Phase 4 – Lateral Development after Shaft 2 Commissioned
  - The fourth phase occurs after November 2021 when Shaft 2 is commissioned for full ore and waste rock hoisting. Additional production crews can now be added to meet the 21-month production ramp-up period to full production of 4 Mtpa. Development during the 30-year full production period focuses on just-in-time development and infrastructure to meet the production schedule.

1.13.3 Underground Infrastructure

Underground infrastructure involves several components, such as ore and waste handling systems, dewatering, maintenance shops, fueling, ventilation, etc. This Section includes the three areas: ore and waste handling (to the crusher), ventilation and refrigeration, and production return water (to but not including the main discharge pumps).

1.13.4 Ore and Waste Handling System

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

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

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

On each of the three main haulage levels, two ore and one waste truck dump will be located near Shaft 2. The truck dumps will be equipped with grizzlies and fixed hydraulic rock breakers. One main waste pass will extend from the 750 Level to the 1050 Level loading station. Two main ore passes will extend from the 750 Level to the 950 Level. At the 950 Level, the ore will be diverted to the coarse ore bins. The 950 Level truck dump grizzly will directly feed the coarse ore bin.

There will be two crusher stations, one below each coarse ore bin, midway between the 950 Level and the 1050 Level loading station. Rock will be reduced in size by the jaw crusher and fed into the crushed ore bin. Each crusher will have a capacity of 3 Mtpa for an overall capacity of 6 Mtpa. This will allow for future production plan increases with the current rock handling infrastructure.

Apron feeders will load the crushed ore or sized waste onto the high-speed weigh conveyor belt on the 1050 Level. A skip-load of ore or waste will be loaded onto the high-speed conveyor while the skips are in transit in the shaft. Upon arrival at the loading station, the load will be discharged directly from the conveyor into the skips via diverter chutes.

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

1.13.5 Mine Ventilation System

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

The ventilation system is designed as an exhausting pull system with a total airflow of 1,200 m^3/s (2,542 kcfm). The two main intake shafts (Shaft 1 and 2), located at the centre of the mining district, will provide fresh air, while Ventilation Raise 1 (near the intake shafts), Ventilation Raise 2 (in the north), and Ventilation Raise 3 (in the south) will be exhausts. Fresh air will be further distributed through the main stations on the 750, 850, and 950 Levels, as well as a series of ramps and haulages. Regulators will be used to direct air to the active mining areas. Exhausted raises have been incorporated into both ends of the mining sublevels.

A bulk air cooling system located on the surface was identified as the best option due to its simplicity and low construction and maintenance costs. The proposed refrigeration plant is designed for 28 MW of refrigeration capacity. All cooled air will intake into the mine via Shaft 1.

1.13.6 Pumping

During mine production, water will be collected in sumps located throughout the mine. The upper level sumps will transfer water through boreholes to lower level sumps where mine duty pumps will then transfer the discharge water through a pipeline and borehole system to the main pump and clarification system located on the 950 Level near Shaft 2.

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

Mine water inflow is estimated at 35 L/s during maximum production. The pumping system is designed for 150 L/s to account for spikes from initial groundwater inflows and paste backfill flushing. Each sump will not operate on a continuous basis but will have an overall 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.

1.14 Metallurgical Test Work Overview

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

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

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

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

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

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

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

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

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

A three-stage crushing (primary crushing within mining scope) and ball mill circuit has been identified as the preferred option, with lowest associated technical risk.

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

Once the optimum flowsheet had been derived, bench scale, batch open circuit flotation variability test work was performed.

In addition to the open circuit variability testing, locked cycle testing was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore schedule, with the focus placed on testing of an ore blend representing the first 5 years of mining. This locked cycle flotation testing achieved recoveries (3PE+Au) in the range 83.1%-88.7% with a final concentrate containing approximately 60 g/t-95 g/t (3PE+Au). Locked cycle testing on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries (3PE+Au) in the range 77.3%-85.5% with a final concentrate containing approximately 80 g/t-120 g/t (3PE+Au).

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

This relationship between 3PE+Au head grade, mass pull and final concentrate grade was incorporated into the recovery modelling in order to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as summarised in Figure 1.6.

Figure 1.6 3PE+Au Recovery as a Function of Concentrate Grade

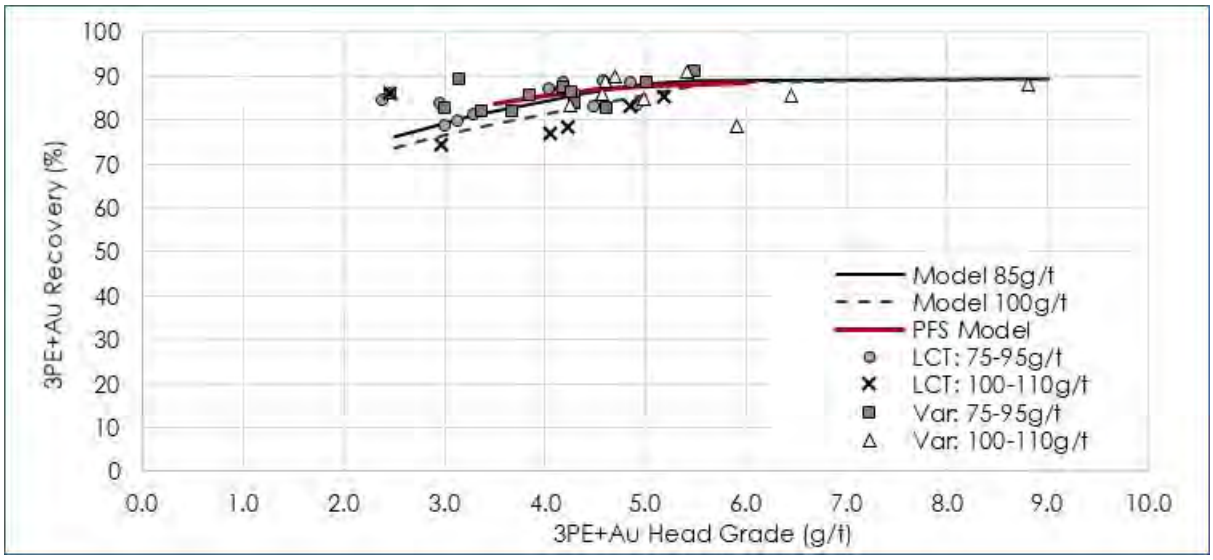


Figure by DRA,2017.

1.15 Recovery Methods

The Platreef concentrator is sized to process 4 Mtpa Platreef ore and includes all ore processing requirements from the Run of Mine (ROM) storage silos through to final concentrate load out and tailings disposal systems. The design is based on two modular 2 Mtpa milling and flotation circuits which were selected based on the concentrator production ramp-up profile. This modular approach with shared crushing, tailings disposal and concentrate handling circuits, allows for increased processing flexibility and introduces process redundancy whilst allowing for phasing of capital.

The process design has been developed based on the test work findings and assessments, various desktop level trade-off studies and relevant DRA design information. The Platreef concentrator flowsheet is based on a conventional three-stage crushing and ball milling circuit followed by flotation and dewatering stages. This single-stage milling and flotation process flowsheet is well known in the industry, and has been proven as a suitable processing route for various platinum ores. The flotation cleaner circuit design provides adequate flexibility to target different final concentrate grades. The process description is presented in Figure 1.7.

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

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

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

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

Figure 1.7 Concentrator Flowsheet

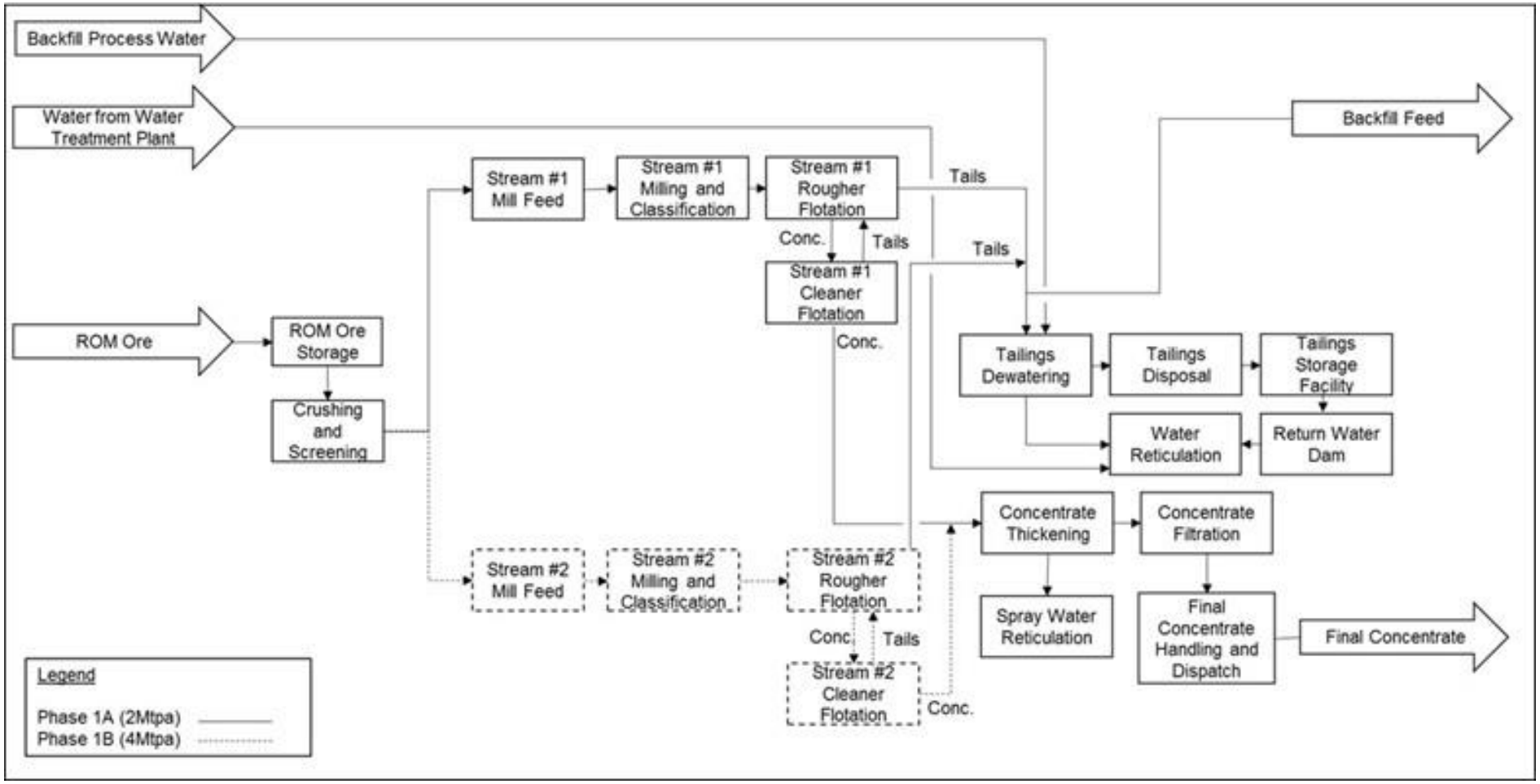


Figure by DRA 2016.

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 1.8 shows the Platreef 2017 FS designs of the Platreef Project location within local region.

Figure 1.8 Platreef Project and Local Region

Figure by Ivanhoe 2017

1.16.1 Bulk Water

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

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

Ivanplats requires an average bulk water supply of 7,500 m3/d for the operational phase of the mine. Ivanplats has been pursuing the following sources of bulk water:

- A supply from the Olifants River Water Resources Development Project (ORWRDP). This supply will be from the Phase 2B pipeline from the Flag Boshielo Dam;
- A local source of treated sewage effluent (grey water);
- A supply from Magalies Water from their Klipvoor Scheme in the Crocodile West Catchment; and
- A supply of treated sewage effluent (grey water) from the Polokwane Municipality.

A bulk water supply from the one or more of the above sources together with the local groundwater sources will be used to supply the mine. Ivanplats is an active member of the Joint Water Forum (JWF) which represents the collective interests of the mines in the Eastern and Northern Limbs. Ivanplats, through the JWF, is looking to secure water from the ORWRDP and/or the Polokwane Municipality.

1.16.2 Project Power Supply

The bulk power supply is to be sourced from Eskom, the South African national power utility. The application for 70 MVA of power has been submitted to Eskom, and the budget quote received. The budget quote was accepted and the appropriate deposits provided.

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

Upon completion of the works, these will be handed over to Eskom to form part of the utilities supply network. As represented in Figure 1.9, the Platreef Project is to be fed from the Eskom Borutho Main Transmission Station (MTS). Two 132 kV overhead line (OHL) feeder bays have been provided by Eskom.

Figure 1.9 Proposed Power Transmission

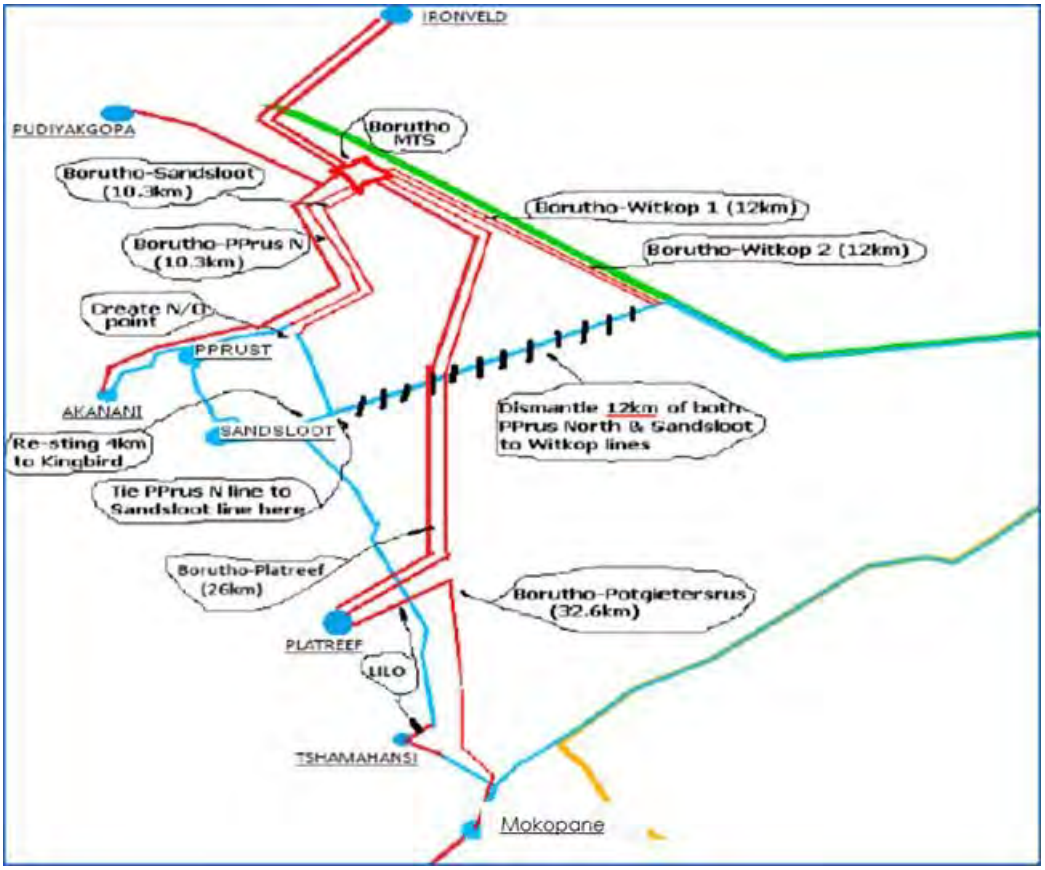


Figure supplied by DRA 2017.

From the Borutho MTS 2 x 26 km Kingbird 132 kV OHLs are to be constructed to feed the Platreef 132/11 kV substation. At the Platreef 132/11 kV substation, 3 x 132/11 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 80 MVA with a future capacity of 120 MVA.

The construction of the OHL and substation is forecast to commence in the second half of 2017 with completion and handover in the first half of 2019. The final budget quote, self-build and electricity supply agreements have been drafted.

The electricity supply agreement caters for a ramp-up period. From an initial supply of 15 MVA in 2019, the supply notified maximum demand (NMD) will be increased as required for the load build-up. Eskom further indicated that a supply increase to 100 MVA is not seen as a problem as Medupi power station will be fully operational beyond 2023. The application for increased power to 100 MVA will commence once the 70 MVA agreements have been concluded. The NMD is the maximum electrical power demand in kVA, over a half hour period.

In addition to ESKOM bulk power supply agreements and as part of the long-term sustainability plan, Ivanplats have entered into an expression of interest with an independent power producer. Ivanplats have indicated their interested of becoming an off-taker of up to 80MW of renewable energy. The qualifying criteria would be that at inception, all energy sold to Ivanplats will be at prices below the Eskom Megaflex tariff structure. The annual tariff increase will be limited to CPI, resulting in a predictable long-term energy pricing forecast as well as mitigation against Eskom's future pricing risk. The independent power producer model is that no capital investment is required from Ivanplats. The independent power producer would be constructing a solar PV farm in close proximity to Ivanplats, which will be able to serve multiple customers. Multiple off-takers will ensure the benefit of economies of scale and that the power purchase agreement (PPA) can be treated as an "off-balance-sheet" transaction, thus having no negative impact on Ivanplat's ROCE. Other benefits will include a lower carbon footprint, temporary and permanent job creation and economic stimulation of local secondary industries. With the rapid advancement of energy storage technology, it is envisaged that a second phase of this project will provide energy storage capacity and add significant flexibility in terms of energy usage during the peak tariff periods. This agreement will contribute positively to the long-term sustainability of the Platreef Project.

1.16.3 Access Roads

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

The N11 highway connects Mokopane with the South Africa/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 planned mine site.

1.16.4 Tailings Storage Facility

The proposed tailing storage facility (TSF) site is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed TSF site is considered a feasible site considering all applicable engineering standards for tailings storage facilities. The TSF will have an operating life of 32 years, during this time approximately 47.2 Mt of tailings will be stored within the TSF. The remainder of the tailings will be used as backfill in the underground mine. The design, however, also caters for an 8 Mtpa ramp up in production. The TSF is compliant in terms of required tonnage profile production split between the backfill requirement and TSF of 35% average, but is conservatively designed for 40% reporting to the TSF.

The hybrid paddock deposition methodology and tailings upstream raising scheme as proposed is feasible, with 100 m wide paddocks and the required drainage systems as proposed within the design. Over and above the standard drainage requirements and international best practice implemented, a containment berm is proposed downstream of the TSF and newly proposed N11 highway, which serves as an additional mitigation measure to ensure the safety of the downstream infrastructure. The tailings characteristics as identified during the Platreef 2017 FS, confirmed the hybrid paddocking deposition methodology as a feasible option. The proposed geometry of the TSF has an actual rate of rise of 0.9 mpa at final height and an average of 1.3 mpa over the life of the facility. The hybrid paddocking allows for a maximum rate of rise of 3.0 mpa, hence the proposed deposition methodology is well within acceptable operating standards.

The TSF will be a HDPE lined TSF which is compliant to the prescribed South African National Environmental Management Waste Act, regulation R636. A staged development (four pre-deposition stages) is a feasible strategy which will result in capital deferral over five year periods for the first 10 years and then 10 year deferral periods during the following 20 years. The stormwater management system and water balance for the TSF is compliant with the South African National Water Act (NWA) Government Notice 704 (GN 704). The dirty and clean water will be separated and the water routing structures are designed for the 1-in-50 year 24-hour storm event, with an acceptable freeboard. The 1-in-100 year 24-hour storm event can be contained within the storm routing structures with a reduced freeboard.

A minimum freeboard of 1.9 m must and can be achieved during the operations, which constitutes the minimum 0.8 m statutory freeboard including 1-in-100 year 24-hour storm event storage volume. The proposed decant system consisting of a floating barge and penstock tower and outfall pipeline is functional to meet the capacity requirement, and will be routed through a silt trap for desilting and into the return water dam (RWD) to be returned back to the plant for reuse.

The TSF as proposed in the Platreef 2017 FS indicates that the facility satisfies all considerations of stability investigated, namely static and pseudo static analysis considering both circular and translational failure modes. The TSF during operations shall be monitored according to South African National Standards (SANS) 10286, and the applicable tailings management frameworks must be implemented, which includes a rigorous surveillance and monitoring programme.

1.17 Market Studies and Contracts

The Platreef Phase 1 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/demand deficits. Favourable positioning on the cost curve, base metal diversification and a natural South African rand (ZAR) hedge should all conspire to make the concentrate attractive to South African toll smelters. Discussions between Ivanplats and interested parties are well underway, and Ivanplats expects this to result in a definitive tolling agreement.

Ivanplats prepared and updated marketing studies for the Platreef 2015 PFS and updated both content and assumptions for the Platreef 2017 FS. This Platreef 2017 FS is based on an examination of options around the Phase 1 production scenario of 4 Mtpa which would yield approximately 175,000 tpa concentrate. Ivanplats has not yet agreed specific offtake agreements for this Phase 1 production, as it is it is not typical industry practice to have such contracts in place five years before production commences. Ivanplats has a clearly defined development strategy to secure smelting and refining capacity that has considered potential treatment capacities, primarily in South Africa, and has mapped a development path for placing concentrates and expects capacity to become available by the time that steady state production is achieved.

With the establishment of a number of smaller PGM mining firms, toll smelting and refining contracts and 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 sulphate. The vast majority of these products are refined in South Africa, but some high-grade PGM residues are shipped overseas for final processing in Germany.

1.18 Environmental Studies, Permitting, Social, and Community Impact

In 2013, Ivanplats undertook the Environmental Impact Assessment (EIA) and Environmental Management Programme (EMP) for the Platreef Project in support of a Mining Right, Environmental Authorisation (EA) and Waste Management Licence (WML) application. Since the approval of these applications, Digby Wells has provided ongoing environmental advisory support on the Platreef Project and undertaken further regulatory applications, including an EIA/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 of the Masodi WWTW pipeline and the re-alignment of the TSF pipeline route, will require an amendment application process to be undertaken on the EA (estimated cost of R750 000.00); and
- 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 EIA 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 EIA 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 EIA 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. Ivanplats made commitments in terms of its Social and Labour Plan (SLP) for a five-year period between 2015 and 2019.

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 programs 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 runoff 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; and
- Noise impacts emanating from machinery and vehicles.

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

The main potential social impacts associated with the Platreef Project include some economic displacement due to a loss of access to cultivated land or other livelihood resources, influx in job seeking which, combined with the additional workforce, will place considerable pressure on local infrastructure and services, negative perceptions of project impacts and increased traffic volumes on roads in the vicinity of the local project area. Further to this, there are social risks due to the social environment under which the Platreef Project operates as well as stakeholder fatigue resulting from ongoing mining and exploration activities within the area. Community unrest poses the risk of striking, property destruction and interruptions of operation schedules. The various stakeholder engagement processes revealed a reoccurrence of issues raised by stakeholders regarding the Platreef Project. Stakeholder engagement is an ongoing process and a grievance mechanism has been developed to manage stakeholder concerns.

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

1.19 Production Summary

The production plan for the Platreef 2017 FS focused on maximizing higher-grade areas. The ore body was targeted to recover approximately 130 Mt at the highest NSR. This resulted in using declining cut-off grades and decreasing the NSR cut-off from $155/t NSR to $80/t NSR.

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

The mine production forecast is shown in Figure 1.10 and the key average annual production results over the planned 32 year mine life are shown in Table 1.4.

Figure 1.10 Production Forecast

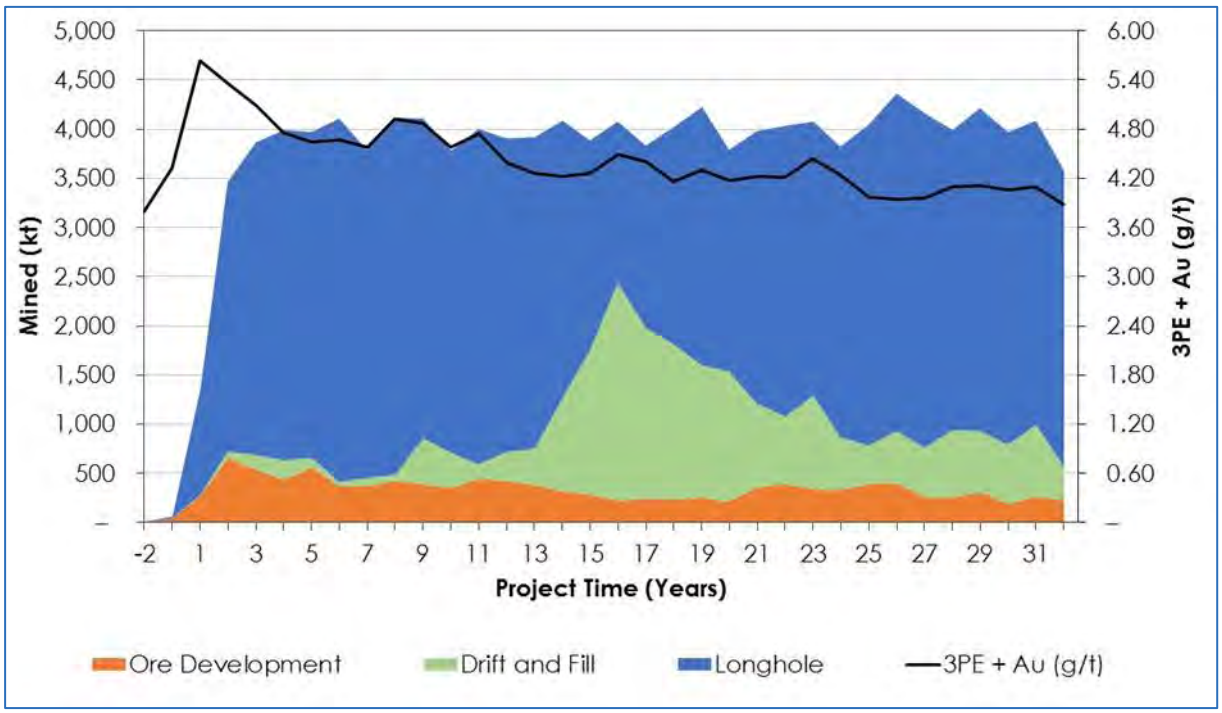


Figure by OreWin, 2017.

Table 1.4 Platreef 2017 FS Average Annual Production Forecast

Item	Unit	LOM AVERAGE
Mined and Processed	Mtpa	3.90
Platinum	g/t	1.95
Palladium	g/t	2.01
Gold	g/t	0.30
Rhodium	g/t	0.14
3PE + Au	g/t	4.40
Copper	% Cu	0.17
Nickel	% Ni	0.34
Concentrator Recoveries		
Platinum	%	87.4
Palladium	%	86.9
Gold	%	78.6
Rhodium	%	80.5
Copper	%	87.9
Nickel	%	71.9
Concentrate Produced	ktpa	174
Platinum	g/t	38.2
Palladium	g/t	39.1
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE + Au	g/t	85.1
Copper	% Cu	3.3
Nickel	% Ni	5.5
Recovered Metal		
Platinum	kozpa	214
Palladium	kozpa	219
Gold	kozpa	30
Rhodium	kozpa	14
3PE + Au	kozpa	476
Copper	Mlbpa	13
Nickel	Mlbpa	21

1. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.
2. Production over 32 years

1.20 Capital and Operating Cost Summary

The total pre-production and sustaining capital costs required, including contingency, sourced from the Platreef 2017 FS, are shown in Table 1.5.

Table 1.5 Total Projection Capital Cost

US$M	Pre-Production	Sustaining	Total
Mining			
Geology	12	9	20
Mining	779	349	1,129
Capitalised Operating Costs	63	–	63
Subtotal	854	358	1,213
Concentrator & Tailings			
Concentrator	240	6	246
Capitalised Operating Costs	0.05	–	0.05
Subtotal	240	6	246
Infrastructure			
Infrastructure	230	23	253
Site Costs	9	3	11
Capitalised Operating Costs	36	–	36
Subtotal	275	26	300
Owners Cost			
Owners Cost	44	8	52
Closure	1	16	17
Subtotal	45	24	69
Capex Before Contingency	1,413	414	1,827
Contingency	131	3	135
Capex After Contingency	1,544	418	1,962

1. Sustaining capital expenditure also includes 2023 construction capital expenditure
2. Totals vary due to rounding.

Mine site cash costs are summarised in Table 1.6. The revenues and operating costs, are presented in Table 1.7 along with the net sales revenue value attributable to each key period of operation.

The higher nickel and copper grades contribute to lower operating cash costs for the Northern Limb as illustrated by Figure 1.11. Among the current and future Northern Limb producers, the Platreef Project estimated cash cost of US$326/oz recovered 3PE+Au, net of copper and nickel by-product credits, ranks at the bottom of the cash-cost curve. The Net Cash Flow after Tax and the Cumulative Cash Flow after Tax is shown in Figure 1.12.

Table 1.6 Cash Costs After Credits

	US$/oz Recovered 3PE + Au		
	YEARS 1-5 AVERAGE	YEARS 1-10 AVERAGE	LOM AVERAGE
Mine Site cash costs	442.3	392.1	399.5
Realisation	266.6	304.3	339.8
Total Cash Costs Before Credits	708.9	696.4	739.2
Nickel Credits	304.5	306.5	334.4
Copper Credits	71.5	71.1	79.1
Total Cash Costs After Credits	332.9	318.9	325.7
Sustaining Capital Costs	25.4	26.1	25.0
Total Cash Costs After Credits & Sustaining Capital	358.3	345.0	350.7

Totals vary due to rounding.

Figure 1.11 Cash Cost Comparison Platreef and 2017 Producers

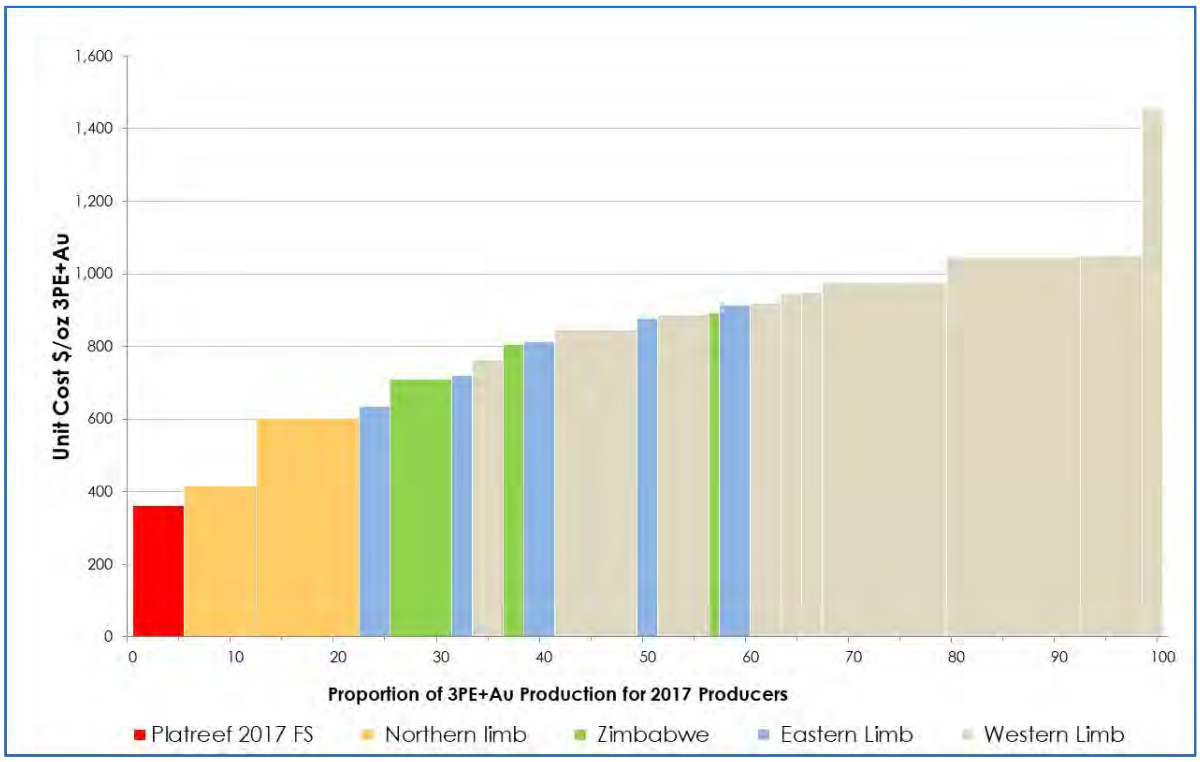


Figure by OreWin Chart shows cash cost including sustaining costs for producer cost data. Source: SFA (Oxford).

Table 1.7 Operating Costs and Revenues

	Life of Mine TOTAL US$M	US$/t Milled		
		YEARS 1-5 AVERAGE	YEARS 1-10 AVERAGE	LOM AVERAGE
Gross Sales Revenue	22,298	202.01	195.65	178.84
Less: Realisation Costs				
Transport	76	0.70	0.68	0.61
Treatment & Refining	4,014	36.36	35.22	32.19
Royalties	1,084	0.83	5.89	8.70
Total Realisation Costs	5,174	37.89	41.79	41.50
Net Sales Revenue	17,124	164.12	153.86	137.34
Site Operating Costs				
Mining	4,150	46.39	37.89	33.29
Processing & Tailings	1,303	10.91	10.64	10.45
Infrastructure	295	2.91	2.60	2.37
Site Cost	56	0.37	0.42	0.45
General & Administration	279	2.29	2.30	2.24
Total	6,083	62.87	53.86	48.79
Operating Margin	11,041	101.24	100.01	88.55
Operating Margin	50%	50%	51%	50%

Totals vary due to rounding.

Figure 1.12 Cumulative Cash Flow After Tax

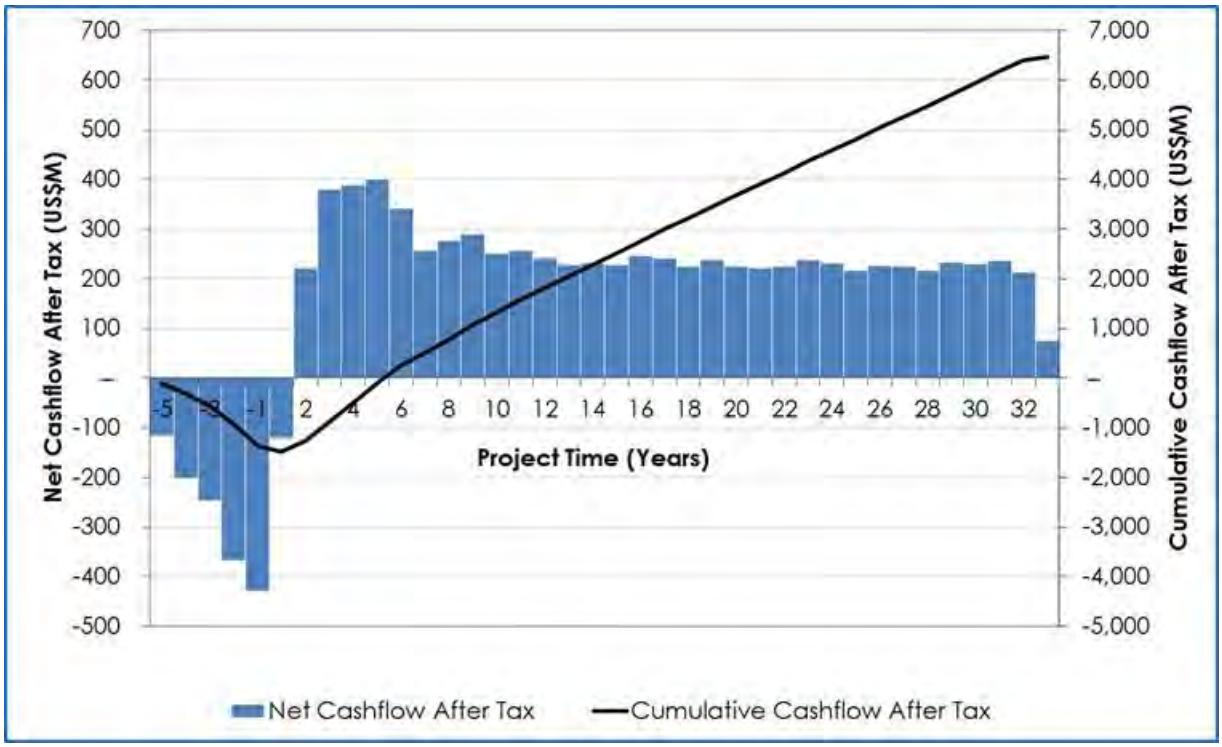


Figure Produced by OreWin, 2017.

1.21 Summary of Financial Results

The economic analysis uses price assumptions of US$1,250/oz Pt, US$825/oz Pd, US$1,300/oz Au, US$1,000/oz Rh, US$7.60/lb Ni, and US$3.00/lb Cu. The prices are based on a review of consensus price forecasts from financial institutions and similar studies that have recently been published. The basis of the operational framework of the mine used in the analysis is Republic of South Africa legislation. Costs estimated in ZAR have been converted to US$ at an exchange rate of 13 ZAR/US$. Key economic assumptions in the analyses are shown in Table 1.8.

Table 1.8 Economic Assumptions

Parameter	Unit	Financial Analysis Assumptions
Platinum	US$/oz	1,250
Palladium	US$/oz	825
Gold	US$/oz	1,300
Rhodium	US$/oz	1,000
Copper	US$/lb	3.00
Nickel	US$/lb	7.60
Copper Refining Charges	% Gross Sales	18%
Nickel Refining Charges	% Gross Sales	18%
Precious Metals Refining Charge	% Gross Sales	18%

The results of the financial analysis show an After Tax NPV8 of US$916M. The case exhibits an after tax IRR of around 14% and a payback period of around five years. The estimates of cash flows have been prepared on a real basis as at 1 January 2017 and a mid-year discounting is taken to calculate NPV. A summary of the financial results is shown in Table 1.9.

Table 1.9 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	8,897	6,471
	5.0%	2,794	1,961
	8.0%	1,392	916
	10.0%	838	500
	12.0%	461	217
	15.0%	99	-57
	20.0%	-212	-291
Internal Rate of Return		16.2%	14.2%
Project Payback Period (Years)		5.2	5.3

1.22 Summary of Platreef 2017 FS

The key features of the Platreef 2017 FS included:

- Development of a large, mechanised, underground mine is planned at an initial 4 Mtpa throughput scenario.
- Planned average annual production rate of 476 koz of platinum, palladium, rhodium, and gold (3PE+Au).
- Estimated pre-production capital requirement of approximately US$1.5 billion, including US$131 million in contingencies.
- After-tax Net Present Value (NPV) of US$916 million, at an 8% discount rate.
- After-tax Internal Rate of Return (IRR) of 14%.
- The Platreef 2017 FS maintains options available to accelerate expansions, to the 8 Mtpa or the 12 Mtpa scenarios, as the market dictates.

A summary of the key project physical and financial metrics is shown in Table 1.10.

Table 1.10 Operating Costs and Revenues

Item	Unit	Total
Mined and Processed	Mt	125
Platinum	g/t	1.95
Palladium	g/t	2.01
Gold	g/t	0.30
Rhodium	g/t	0.14
3PE + Au	g/t	4.40
Copper	% Cu	0.17
Nickel	% Ni	0.34
Concentrate Produced	kt	5,568
Platinum	g/t	38.2
Palladium	g/t	39.1
Gold	g/t	5.3
Rhodium	g/t	2.4
3PE + Au	g/t	85.1
Copper	% Cu	3.3
Nickel	% Ni	5.5
Recovered Metal		
Platinum	koz	6,846
Palladium	koz	6,994
Gold	koz	952
Rhodium	koz	436
3PE + Au	koz	15,228
Copper	Mlb	402
Nickel	Mlb	670
Key financial results		
Life of Mine	Years	32
Pre-Production	US$M	1,544
Mine-Site Cash Cost	US$/oz Recovered 3PE + Au	399.5
Total Cash Costs After Credits	US$/oz Recovered 3PE + Au	325.7
Total Cash Costs After Credits & Sustaining Capital	US$/oz Recovered 3PE + Au	350.7
Site Operating Costs	US$/t Milled	48.79
After-Tax NPV8%	US$M	916
After-Tax IRR	%	14.2
Project Payback Period	Years	5.3

Totals may not add due to rounding.

1. The economic analysis is based on Probable Mineral Reserves only.
2. 3PE+Au = platinum, palladium, rhodium and gold.
3. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
4. A declining Net Smelter Return (NSR) cut-off of US$155/tonne–$80/tonne was used in the Mineral Reserve estimate.
5. Metal price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.
6. All-in cash costs include sustaining capital costs.

1.23 Platreef Future Development Scenarios

Ivanhoe has identified significant Mineral Resources on the Platreef Project. The Platreef 2017 FS was applied to part of the selectively-mineable Mineral Resources within and adjacent to Turfspruit Cyclic Unit (TCU) mineralised zones. The Platreef 2017 FS analysed Phase 1, the 4 Mtpa Concentrator Case.

Ivanhoe's development plan for Platreef Project considers three phases of underground mining and concentrator expansion that were identified in the March 2014 Platreef Preliminary Economic Analysis (PEA). The Platreef 2014 PEA is a PEA as defined in NI 43-101 Standards of Disclosure for Mineral Projects, and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability but have reasonable prospects for eventual economic extraction. The development scenarios that were identified in the Platreef 2014 PEA are:

- Phase 1 Concentrator 4 Mtpa
- Phase 2 Concentrator 8 Mtpa
- Phase 3 Concentrator 12 Mtpa

The base case for the 2014 Platreef PEA analysis was Phase 2; the 8 Mtpa concentrator case. The development scenarios and additional options for the Platreef Project are shown in Figure 1.13. The development scenarios describe a staged approach where there is opportunity to expand the operation depending on demand, smelting and refining capacity, and capital availability. As Phase 1 is developed and taken into production, there is opportunity to modify and optimise the definition of Phases 2 and 3. This would allow changes to the timing or expansion capacity to suit the conditions at the time.

The options for a smelter and / or a base metal refinery (BMR) are still being studied, and their timing and sizing need to undergo further analysis. Opportunities for additional phases after Phase 3 may be available, and these will also require additional investigation.

Phase 1 includes the construction of a concentrator and other associated infrastructure to support a start-up to production at a nominal plant capacity of 4 Mtpa. Phase 2 includes an additional ramp-up to a plant capacity of 8 Mtpa. Phase 3 envisages a further ramp-up to a plant capacity of 12 Mtpa. All production is sourced from underground mining.

Phase 1 has recently commenced with the sinking of an exploration shaft (now called Shaft 1) to provide underground access to the Platreef mineralisation for the purpose of obtaining a bulk sample and will provide primary ventilation intake to assist with early underground development.

In the current design, Shaft 2 has a 6 Mtpa capacity. This, combined with the conversion of Shaft 1 to a production shaft with a 2.5 Mtpa capacity, may present an opportunity to achieve the anticipated production rate of 8 Mtpa in Phase 2 as described in the Platreef 2014 PEA (March 2014). Although such provision will result in delays to the current base case schedule and an increase in capital expenditure, it helps the project with an efficient transition to the expansion Phase 2.

Each development phase has the same underlying plan for the construction and operation of a concentrator processing facility, with the capacity aligned to the requirements of each phase.

Figure 1.13 Platreef Project Development Scenarios

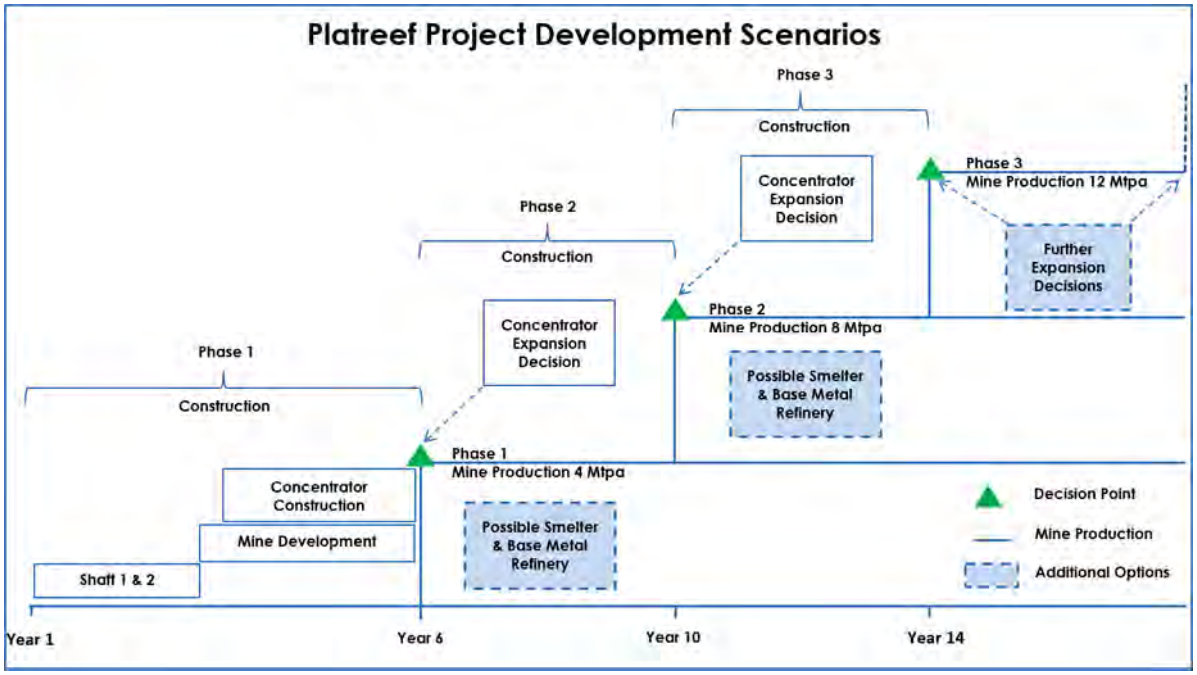


Figure by OreWin, 2014.

1.24 Conclusions

1.24.1 Platreef 2017 Feasibility Study

The Platreef 2017 FS 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 2017 FS will provide the basis for detailed planning of execution activities and to update the long-term development plans for Platreef.

The mine plan in the Platreef 2017 FS has been improved over the Platreef 2015 PFS by targeting higher-grade zones. This has increased the overall LOM grade and the grade profile in the early production years. It has also modified the top cut development approach to dramatically reduce the amount of ore development on the critical path, and the ratio of ore development to Longhole Stoping. This improvement in the grade profile has meant that the results of the economic analysis in the Platreef 2017 FS are similar to those of the Platreef 2015 PFS even though there has been a reduction in the long-term metal prices and changes in costs.

1.24.2 Geology and Mineral Resource Estimates

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

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

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

This represents exploration upside for the Platreef Project. Drill intercepts ≥ 2.0 g/t 3PE+Au in the FW domains are narrow, and suggest selective mining would be required. Grade continuity is best observed at a 1.0 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).

1.24.3 Mining

The mine plan and expenditure schedule presented herein is reasonable. The plan is based on Platreef 2017 FS data and established mining practice. The resource model and geotechnical parameters provided to Stantec 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.

1.24.4 Metallurgy

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

The test work programmes have been conducted by parties well versed in the processing of ores from the Bushveld Igneous Complex. The necessary checks and balances have been applied to ensure that the test work 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 test work for the purposes of the Platreef 2017 FS are deemed to be sufficient.

The proposed circuit is considered to be the preferred option for the concentrator. The use of a multi-stage crusher circuit followed by a single stage milling circuit is considered to be the option of least risk to the project.

The proposed flotation circuit is based on interpretation of the results obtained from the bench-scale flotation test work. 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 modular approach used for Phase 1 is considered to be appropriate for this level of study.

1.24.5 Infrastructure

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

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

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

The Eskom bulk power supply application made in 2011 is not sufficient, and Ivanplats needs to engage with Eskom to revise the current application to NMD of 100 MVA as a matter of urgency. However, the current 70MVA application will be sufficient for the initial start-up of 2 Mtpa.

The availability of skilled labour resources for both construction and operational phases of the Platreef Project is limited, and the training programme will have to be closely monitored to ensure that the correct skills development is done at the correct time, depending on the phase of the project. These skilled resources form part of the human resource development programs outlined in the Social and Labour Plan which has been approved as part of the MRA commenced in 2015 as part of the Social and Labour Plan commitments over the next 5 years.

1.25 Recommendations

1.25.1 Platreef 2017 Feasibility Study

The Platreef 2017 FS will provide the technical basis for Ivanplats to commence 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.

Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of Phases 2 and 3 should be re-evaluated using the findings of the Platreef 2017 FS.

1.25.2 Delineation Drilling

Ivanhoe has that proposed stope positions and stope boundaries should be delineated in more detail as development towards mineralization occurs. The drill spacing would 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.

1.25.3 Geology and Mineral Resource Estimates

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

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

1.25.4 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. Stantec recommends that delineation drilling be completed as prioritised in the current schedule to better define the contacts for proper infrastructure placement.

1.25.5 Metallurgical

The metallurgical test work programme has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes. To further evaluate optimisation opportunities and confirm additional detail design parameters, a mini pilot plant test work programme is proposed and will be undertaken as part of the project implementation phase. The mini plant test work programme will be conducted at Mintek in 2017 using an 8 t composite sample comprised of the remaining drill core sample from the variability drilling programme. The mini plant sample selection process will aim to target the expected life of mine ore blend and feed grade. The mini pilot plant testing should focus on establishing and/or confirming the following:

- The effect of rougher flotation feed PSD for the pilot scale primary rod mill and secondary ball mill circuit as compared to the bench-scale flotation performance using a laboratory rod mill
- Determination of the optimal high chrome grinding media specification
- Evaluate pilot-scale flotation mass pull as compared to bench-scale testing
- Measuring the steady-state flotation concentrate solids concentrations and confirm the expected cleaner circuit operating density
- Evaluate the impact of scavenger cleaner tails recycle
- Quantify the benefit of column flotation and establish the optimal location within the flowsheet
- Evaluate the impact of reagent build up in recycle streams and optimize reagent addition requirements
- Evaluate the effect of operating with process water sourced from Mokopane

- Mach reactor scouting tests to evaluate potential opportunities for improved fines recovery
- Generate tailings sample for confirmatory thickening test work on a classified feed sample, to confirm the design parameters
- Generate sufficient cleaner concentrate for thickening and filtration test work, to confirm the design parameters

1.25.6 Infrastructure

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

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

Ivanplats is in the process has notified of and requested from Eskom the 30 MVA additional power demand required for the Platreef Project.

2 INTRODUCTION

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

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

A Joint Venture (JV) with Atlatsa Resources Corporation covers Prospecting Right LP30/5/111/2/740PR. Together, these two prospecting rights form the Platreef Project. Holdings in the Platreef Project are through South African subsidiary Ivanplats (Pty) Ltd (Ivanplats).

For the purposes of the Platreef 2017 FS, the name Ivanhoe refers interchangeably to, Ivanhoe Mines Ltd., the predecessor company named Ivanhoe Nickel and Platinum Ltd, and to Ivanplats. Ivanplats was formerly named Platreef Resources and African Minerals.

2.1 Terms of Reference

The Platreef 2017 FS is an Independent Technical Report (the Report) for the wholly-owned Platreef nickel–copper–gold–platinum group element (PGE) project (the Platreef Project) located near Mokopane, in the Limpopo Province of the Republic of South Africa.

The Platreef 2017 FS 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 2017 FS:

- OreWin Pty Ltd (OreWin): Overall report preparation, general and administration costs, and financial model.
- Amec Foster Wheeler E&C Services Inc (Amec Foster Wheeler): Geology and Mineral Resource estimation.
- SRK Consulting Inc. (SRK): Mine geotechnical recommendations.
- Stantec Consulting International LLC (Stantec): Underground mine plan and Mineral Reserve.

- DRA Projects SA (Pty) Ltd (DRA): Process engineering and infrastructure.
- Golder Associates Africa (Pty) Ltd: Tailings Storage Facility, Bulk Water and Enviromental, Social and Community Impact.

The Platreef 2017 FS uses metric units of measure. The currency used is Quarter 3 2016 United States dollars (US$), 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 to1.2, 1.3.2, 1.17, 1.19 to 1.24.1, 1.25.1; Section 2; Section 3; Sections 4.1 to 4.2.4, 4.3 to 4.6; Section 6.2; Section 19; Sections 21.1 to 21.3, 21.4.6 to 21.5, 21.8 to 21.9; Section 22; Section 23; Section 24; Section 25.1; Section 26.1; and Section 27.
- Dr Harry Parker, SME Registered Member (2460450), employed by Amec Foster Wheeler E&C Services Inc as a Consulting Mining Geologist and Geostatistician, and Timothy Kuhl, SME Registered Member (1802300), employed by Amec Foster Wheeler E&C Services Inc as a Principal Geologist, were responsible for: Sections 1.3.1, 1.4 to 1.10, 1.24.2, and 1.25.2 to 1.25.3; Section 2; Section 3; Section 6.1; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8 to 10.9, 10.10.1; Section 11; Section 12; Section 14; Section 25.2; Sections 26.2 to 26.3; and Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting Inc. as Principal Consultant, was responsible for: Section 1.12; Section 2; Section 3; Section 10.10.3; Section 16.1; and Section 27.
- Jon Treen P. Eng. (Mining), PEO (90402637), employed by Stantec Consulting International LLC as Mining Business Line Leader, was responsible for: Sections 1.11, 1.13; 1.24.3, 1.25.4; Section 2; Section 3; Section 15; Sections 16.2 to 16.5; Sections 21.4.1 to 21.4.2, 21.4.7, 21.8 to 21.9; Sections 25.3 to 25.4; Section 26.4 to 26.5; and Section 27.
- Val Coetzee, B.Eng. (Chemical), M.Eng. (Mineral Economics), Process Manager, DRA Projects (Pty) Ltd, was responsible for: Sections 1.14 to 1.15, 1.16, 1.16.2, 1.16.3, 1.24.4 to 1.24.5, 1.25.5 to 1.25.6; Section 2; Section 3; Sections 5.1 to 5.5.1, 5.5.3 to 5.7; Sections 10.7, 10.10.2; Sections 13.1 to 13.8, 13.10 to 13.11; Section 17; Sections 18.1.1, 18.2 to 18.4., 18.7 to 18.16; Sections 21.4.3 to 21.4.5, 21.4.7, 21.6 to 21.9; Sections 25.5 to 25.6; Sections 26.6 to 26.7; and Section 27.
- Francois Marais, employed as Principal Strategic Advisor Engineering by Golder Associates Africa (Pty) Ltd, B.Eng., Pr Eng ECSA, Sections 1.16.1, 1.16.4, 1.18; Section 2, Section 3.4, Sections 4.2.5, 4.7 to 4.10; Section 5.5.2; Section 13.9; Sections 18.1.2, 18.5 to 18.6; Section 20; Sections 21.4.7, 21.8 to 21.9; Section 26.8; and Section 27.

2.3 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Bernard Peters visited the property for two days in February 2010 and for one day in April 2010; on 8 November 2012, on 9 October 2014 and on 17 April 2017. The site visits included briefings from Ivanhoe geology and exploration personnel, site inspections of potential areas for mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area around the Platreef Project site. Bernard Peters has also visited the Ivanhoe office in Sandton South Africa on several other occasions for meetings with Ivanhoe personnel and consultants working on the Platreef Project.
- Dr Harry Parker has made numerous visits to the Platreef Project site between September 2001 and 2003, in 2009, 2010, 2011, 2012 and 2014 and most recently from 6–10 July 2015. During the site visits, Dr Parker personally inspected drill core and surface outcrops, drilling platforms, and sample cutting and logging areas, held discussions on geology and mineralisation with Ivanhoe's staff, and reviewed geological interpretations with staff.
- Mr Timothy Kuhl visited the site from 26 March to 9 April 2010, 19 July to 3 August 2011, 25 January to 3 February 2012, and again from 27 November 2012 to 12 December 2012. Most recently, Mr. Kuhl was at site 13 May to 25 June 2015 and 8 July to 3 August 2015. During these trips, he audited drill data obtained since Amec Foster Wheeler's 2007 database audit (DaSilva, 2007), obtained QA/QC data, field checked drill collars, and collected witness samples for check assays. He also inspected drill core, surface outcrops, and sample cutting and logging areas. Discussions were held with Ivanhoe's staff about project geology and mineralisation; geological interpretations were reviewed, and potential locations of major infrastructure were viewed.
- Mr William Joughin has visited the site for one day during 2011, 23-24 May 2013, 21-22 January 2015 and 9 June 2015 to inspect drill core and to plan the geotechnical investigations. Additional site visits were conducted by SRK staff for quality assurance of the Ivanplats geotechnical logging.
- Mr Jon Treen participated in the Platreef Project kick-off 8-9 June 2015 in South Africa, which included an overall kick off meeting with Ivanhoe staff and all project consultants, and a site visit in the afternoon on 9 June 2015 included briefings from Ivanhoe geology, exploration personnel, discussion of the status of the ongoing geotechnical programme, examination of representative drill core, and inspection of the Shaft 1 surface works. On 10 June 2015 a mining specific alignment meeting with Ivanhoe, DRA, and Stantec occurred. Mr. Treen attended the Platreef Project Kick off and alignment workshop in South Africa to determine scopes and battery limits for the study.
- Mr Val Coetzee visited the site during October 2014 for a general site inspection and visited the Mintek laboratory where the current metallurgical test work is underway.
- Francois Marais conducted a site visit to the proposed mining site during August 2017 which included a specific site visit to the proposed Rietfontein TSF site. Francois Marais 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.

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:	4 September 2017
Mineral Resource Amenable to Selective Underground Mining Methods:	22 April 2016
Bikkuri Mineral Resource Amenable to Selective Underground Mining Methods:	22 April 2016
UMT-FW Mineral Resources Amenable to Underground Mining Methods:	22 April 2016
Supply of the Last Drillhole Information Used in the UMT Models:	24 July 2015
Effective Date of Mineral Reserves:	24 May 2017

2.5 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of the Platreef 2017 FS were used to support preparation of the Report. Additional information was provided by Ivanhoe as supporting information for the QPs.

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

Other supporting information was sourced from Ivanhoe.

Metric units of measurement have been used in the Platreef 2017 FS except where noted, and currency is expressed in US dollars unless stated otherwise.

3 RELIANCE ON OTHER EXPERTS

3.1 Project Ownership, Mineral Tenure, Permits and Agreements

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

QPs Bernard Peters, Harry Parker and Tim Kuhl have fully relied upon, and disclaim responsibility for, information derived from legal experts for this information through the following documents:

- Ivanplats 2017: Platreef 4 Mtpa Feasibility Study, Section 15 –Title and Legal, DRA Report Number: DRA-J0283-STU-REP-909, Ivanplats Report Number: 1051-EV-00-209.
- Ivanplats 21 June 2016: Re Retfontein: email from J Abrahams
- Ivanplats 21 November 2014: Platreef 2014 PFS Property Description and Location.
- Webber Wentzel 2014: Legal Opinion: The South African Mineral Title held by Platreef Resources (Pty) Limited: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanhoe, 25 March 2014.
- Webber Wentzel, 2012a: Legal Opinion: The South African Mineral Title held by Platreef Resources (Pty) Limited: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanplats, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited, Amec Foster Wheeler, 7 September 2012.
- Webber Wentzel, 2012b: Plateau Resources (Pty) Limited: Prospecting Right 740PR in respect of the Farm Rietfontein 2-KS: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanplats, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited, Amec Foster Wheeler, 7 September 2012.
- Leppan Beech Inc., Attorneys, 2009: Opinion Requested Regarding the Integrity of Prospecting Right Protocol 06/2006: letter opinion prepared by Leppan Beech Inc., Attorneys on behalf of Ivanhoe Nickel and Platinum Ltd., 12 November 2009, with two annexes.
- Harrison, M., 2010: Opinion on Various Issues Pertaining to Platreef Resources (Pty) Limited's Prospecting Right; Renewal of the Right and Mining Right Application: letter opinion prepared by Harrison Attorneys on behalf of Ivanhoe Nickel and Platinum Ltd, 12 September 2010.

This information is used in Section 1.4 and Section 4 of the Platreef 2017 FS and in support of the Mineral Resource estimate in Section 14.

3.2 Legal, Political, Royalties and Taxes

The assumptions for legal, political, royalties and taxes have been provided by Ivanhoe and are based on the KPMG memorandum: subject: Updated commentary on specific tax consequences applicable to an operating mine in the Republic of South Africa 10 May 2017 and the document Platreef 4Mtpa Feasibility Study, Section 15 –Title and Legal, DRA Report Number: DRA-J0283-STU-REP-909, Ivanplats Report Number: 1051-EV-00-209 , Ivanplats (Pty) Ltd, July 2017.

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

3.3 Marketing

Bernard Peters, the QP for the marketing assumptions, has relied on Ivanhoe and disclaims responsibility for the marketing assumptions in Section 19. Ivanhoe provided the following documents that have been used:

- King, F 2014a: Platreef Resources Marketing Study: F King January 2014. Marketing report for Platreef.
- King, F 2014b: Review of Platreef Marketing and Fund Raising Execution Plan: F King August 2014. Letter regarding Platreef options for market outlets, toll treatment and commercial terms.
- Platreef Resources (Pty) Ltd 2014a: Platreef Pre-Feasibility Study Section 13 – Marketing September 2014: PFS marketing report.
- Platreef Marketing Studies and Contracts, F King, June 2017. Marketing report for Platreef.

Dr Parker and Tim Kuhl have also fully relied upon and disclaim responsibility for this information in support of the Mineral Resource estimate in Section 14.

3.4 Environmental

Francois Marais, 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 Amec Foster Wheeler 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.

Dr Parker and Tim Kuhl have also fully relied upon and disclaim responsibility for this information in support of the Mineral Resource estimate in Section 14.

4 PROPERTY DESCRIPTION AND LOCATION

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 was effected through notarial execution on 4 2014, and 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 EA has since been amended twice: LEDET granted the first amendment on 19 August 2014, and the second amendment on 30 July 2015, and a third amendment application is currently under review with the DMR.

4.1 Location

The Platreef Project is located at about 24º05'S and 28º59'E. The Project is located in the Limpopo Province of the Republic of South Africa on three farms: Turfspruit (3,561 ha), Macalacaskop (4,281 ha) and Rietfontein (2 878 ha) Figure 4.1.

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. The boundaries for Prospecting Right (PR) LP30/5/111/2/740PR correspond to the perimeter of the Rietfontein (2 KS) farm. Collectively, the MR and PR form the Platreef Project area. Khaki areas on the plan depicted in Figure 4.1 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 4.1 Project Location and Farm Boundaries

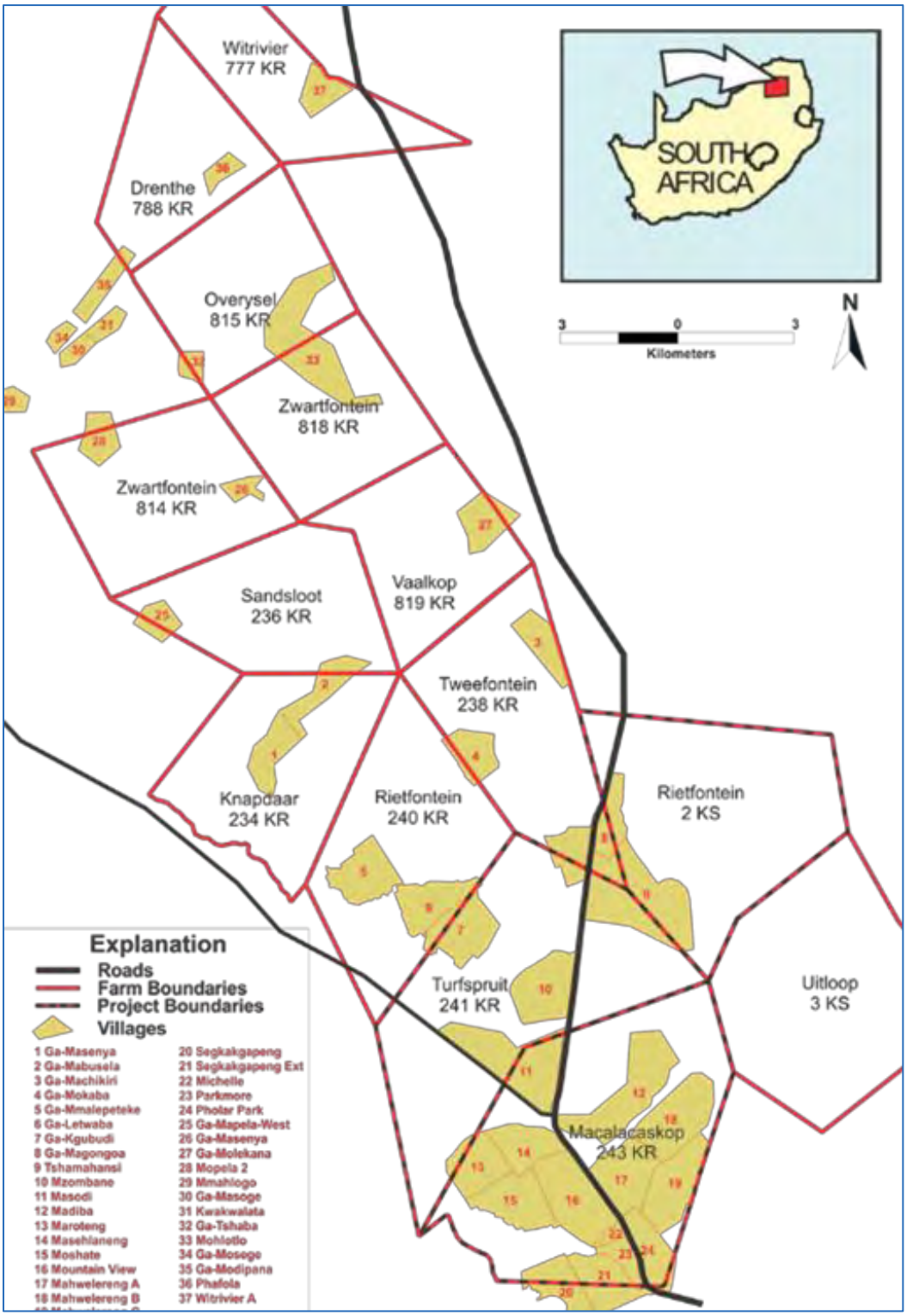


Figure by Ivanplats, 2017.

4.2 Property and Title in South Africa

Information in this sub-section is based on information compiled in the Platreef 2017 FS. The QPs have not independently verified the information.

The Minerals Act 50 of 1991 (the 1991 Act), effective 1992, was the previous legislation governing mining-related issues in South Africa. Under the 1991 Act, mining rights were privately held.

The Mineral and Petroleum Resources Development Act (MPRD Act) No. 28 of 2002, which came into force in May 2004, and replaced the 1991 Act, provides the new regulatory framework for South Africa's mining and minerals industry. The MPRD Act is centred upon mineral rights reverting to the State, and a 'use-it-or-lose-it' principle, ensuring that if a legal entity, such as a mining company, fails to use its mineral rights, it will lose those rights after a certain period. The MPRD Act also has provisions for the State to have the powers to force a mineral rights holder to abandon development projects if the State is of the opinion that the project is not producing at the most efficient levels or is a threat to environmental sustainability or community health. The Department of Mineral Resources (DMR) (formerly the Department of Minerals and Energy (DME)) administers the MPRD Act. The DMR has discretionary powers for awarding conversions of mining rights from the 1991 Act to the MPRD Act. These powers are primarily used in relation to Black Economic Empowerment (BEE) and social-upliftment objectives.

Under the South African Mining Charter of 2004 (the Charter), companies are required to divest a portion of their investments to Historically Disadvantaged South Africans (HDSAs), as a condition of the conversion of old-order mining rights to new-order mining rights. In the Charter, mining company ownership targets for HDSAs are set at 15% during the first five years and 26% in 10 years. A special case was made for state-owned rights where no mining or prospecting operations had previously been conducted. In this instance, the HDSA target was 51% for a one-year period from 1 May 2004. After 1 May 2005, the Charter targets apply.

Mining companies were given up to two years to apply for prospecting permit conversions and five years to apply for mining licence conversions for existing operations. In order to convert a 1991 exploration and mining right under the 1991 Act (old order) to a 2004 exploration and mining right (new order), the holder was expected to lodge a social and labour plan, and to provide an undertaking that outlined how the holder intended to expand mining industry opportunities for HDSAs.

A holder of a mineral right is expected, under the terms of the MPRD Act, to ensure that a mineral resource is optimally exploited. In addition, a rights holder is only entitled to a mining or prospecting right to the extent that the ground holding is actively worked. A planned exploration or mining work programme is required, and must be followed, or corrective measures may be taken by the DMR.

Trade in mining or prospecting rights, such as transfers between parties, or sales, can only be concluded with the approval of the DMR.

4.2.1 Mineral Property Title

A Prospecting Right (PR) is a new-order right (i.e. granted under the MPRD Act) that is valid for up to five years, with the possibility of a further extension of three years. The PR can be obtained either by the conversion of existing old-order PRs (i.e. granted under the 1991 Act or earlier acts) or through new applications.

A Mining Right (MR) is a new-order right issued in terms of the MPRD Act that is valid for up to 30 years, with the possibility of further extension periods, each of which may not exceed 30 years at a time. A MR can be obtained either by the conversion of an existing old-order MR, or as a new-order MR subject to the exercise of the exclusive right of the holder of a new-order PR, or subject to an application for a new MR. Although the MPRD Act does not specifically indicate the Republic of South Africa as the owner of unmined minerals, the ability of a land owner to exercise absolute rights over minerals found on or under their land has been nullified. A landowner retains the ultimate surface rights ownership, but not the minerals ownership.

4.2.2 Mining Rights History and Status

Turfspruit 241 KR and Macalacaskop 243 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.

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 PGEs and all associated metals and minerals mined 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 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.

Ivanplats and the Regional Manager: Limpopo Region of the DMR notarially executed a mining right on 4 November 2014 with DMR reference number LP30/5/2/2/1/10067MR ("the MR"). 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 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.

The MPRDA requires the holder of a MR to 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 MR). The MR was lodged for registration on 3 December 2014. Following a number of queries from the MPTRO, the mining right was successfully registered 3 February 2017.

The version of the MR, which was executed on 4 November 2014 and registered by the MPTRO on 3 February 2017, contained a rider requiring restructuring to increase the participation interest of the Historically Disadvantaged South Africans from the agreed 3% to a minimum of 5% ("the BEE Restructuring Condition").

Ivanplats lodged an appeal 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 in terms of which the BEE Restructuring Condition was withdrawn.

Ivanplats is currently engaging with the DMR and the MPTRO to determine the correct procedure to record this change.

Internal appeals against the grant of the mining right to Ivanplats were lodged with the DMR by the following persons/entities:

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

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. 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. These appeals are currently pending. 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 MR) 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 the following rights and obligations:

1. 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 or underground infrastructure which may be required for the purpose of mining;
2. To prospect or mine for its own account on or under that land for the mineral for which such right has been granted;
3. To remove and dispose of any such mineral found during the course of prospecting or mining;
4. Subject to the National Water Act, 1998 (Act No. 36 of 1998), to 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
5. To 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.

Rietfontein 2 KS

Atlatsa Resources Corporation (formerly Anooraq Resources Corporation) through its South African subsidiary, Plateau Resources Limited, holds an exclusive 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 Rietfontein Prospecting Right was initially valid for a period of five years, expiring on 27 November 2011. Prior to the expiry date, on 22 August 2011, Plateau Resources lodged an application to renew the Rietfontein Prospecting Right for a further period of three years. The renewal was eventually granted in 2014, and the Rietfontein Prospecting Right will finally expire on 1 October 2017.

4.2.3 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.

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"). 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, the farms Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS are zoned "Mining 2" which 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". There are no prohibited land uses listed in respect of land zoned "Mining 2".

To date, no land use management scheme has been adopted in respect of the Mogalakwena Local Municipality under SPLUMA. Accordingly, the land uses for which the properties may be used continue to be those as stipulated in the Scheme.

Ivanplats desires various portions of the relevant properties to be zoned more specifically for mining, industrial, unspecified and "special" land uses, 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 has been prepared and is substantially ready for submission to the relevant municipal planning tribunal. To allow submission, a special power of attorney is required on behalf of the nominal landowner of the farms, the Minister of Rural Development and Land Reform, on behalf of the Republic of South Africa.

The Minister 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 the following Section.

4.2.4 Surface Rights

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 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 with alternative land, or another suitable form of compensation, 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.

Surface Leases

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

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

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

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

The effect is that the Minister of Rural Development and Land Reform will only execute a notarial lease agreement if the lease agreement is approved by the relevant community in accordance with its usage and custom. By law, this usage and custom decision 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 called, 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. Figure 4.2 depicts surface lease areas.

Figure 4.2 Surface Lease Areas

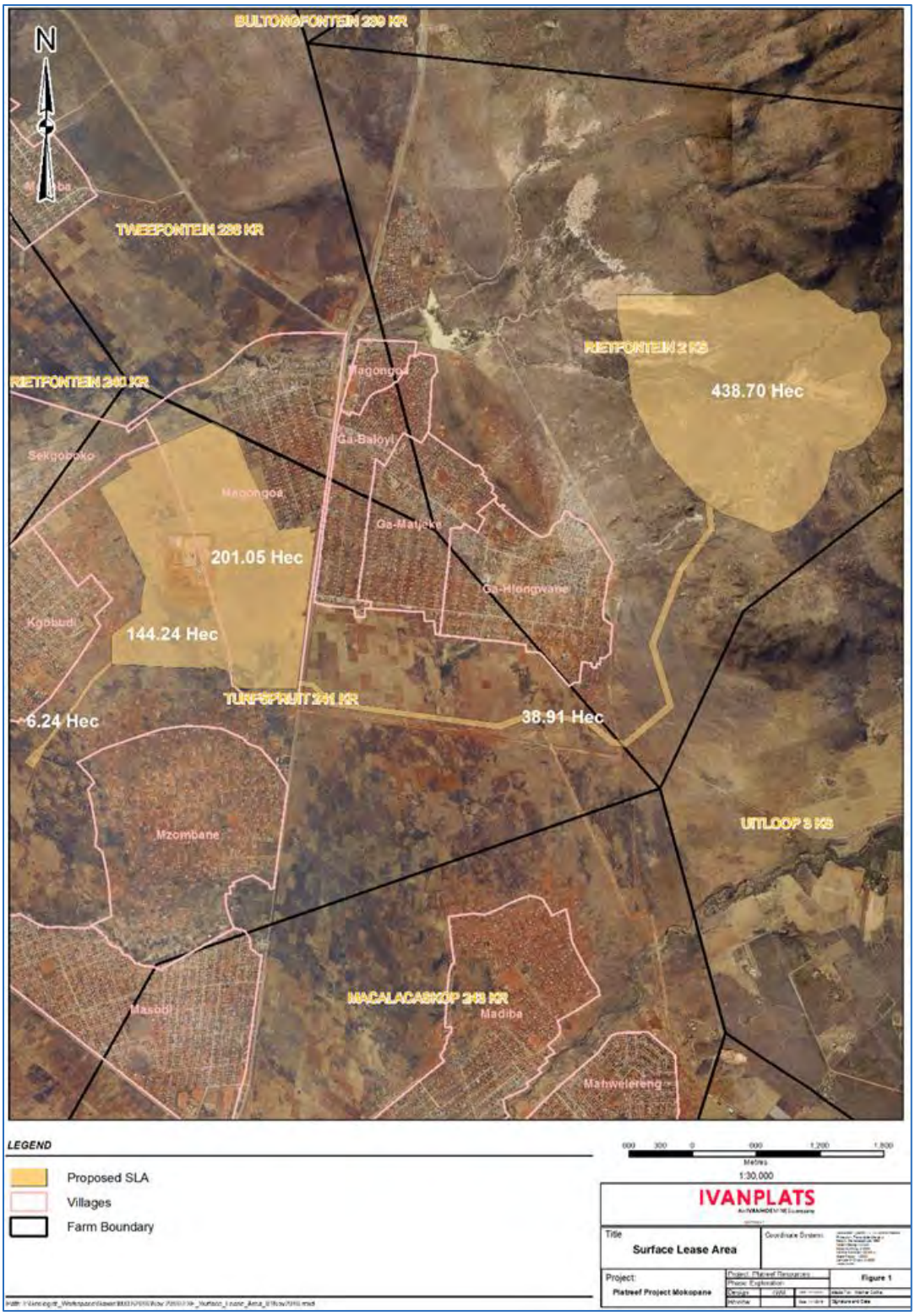


Figure by Ivanplats, 2017.

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; then
- Failing agreement, the Land Claims Commission may refer the claim to the Land Claims Court for final determination.

The Land Claims Court may:

- 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 challenges 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.

The provisions 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.

4.2.5 Environmental Regulations

On 2 August 2010, new Environmental Impact Assessment (EIA) regulations came into effect in South Africa. These new regulations were designed to align the 2006 environmental regulations with the National Environmental Management Act (NEM Act), and to streamline the EIA process. Within the EIA regulations, specified timeframes for receipt of an EIA were stipulated, and some timeframes, such as the end of the calendar year, were excluded from public consultation processes and in the counting of days for both decisions and lodging of appeals.

Under the regulations, lists of activities requiring environmental authorisation prior to commencement were revised to four notices:

- Listing Notice 1: stipulates the activities requiring a Basic Assessment Report (BAR). These are typically activities that have the potential to impact negatively on the environment. However, due to the nature and scale of such activities, such impacts are generally known.
- Listing Notice 2: identifies the activities requiring both a scoping exercise and an Environmental Impact Report (EIR). These are typically considered to be large-scale or highly-polluting activities, and the full range of potential impacts need to be established through a scoping exercise prior to the activity being assessed.
- Listing Notice 3: includes activities that will only require an environmental authorisation through a basic assessment process if the activity is undertaken in one of the specified geographical areas indicated in that listing notice. Geographical areas differ from province to province. An example of such a listing would be erection of a cell phone mast.
- Listing Notice 4: identifies activities 'in identified geographical areas that have been subjected to a pre-assessment process using a spatial development tool and which require environmental authorisation prior to commencement'.

Triggering a listed activity requires that environmental authorisation be obtained before the commencement of the activity. Section 24F of the NEM Act provides that any person who commences a listed activity without the necessary authorisation commits an offence (Section 49A(1)(a)) and, if convicted, such a person may be liable to a fine not exceeding R10 million or imprisonment for a period not exceeding ten years, or to both such fine and such imprisonment (Section 49B (1)).

The NEM Act regulates listed activities, and the granting of environmental authorisations. However, the MPRD Act No. 28 of 2002, provided for environmental regulation of mining operations, such as environmental management programmes which had to be approved in respect of mining operations and financial provision for rehabilitation and closure.

In 2007–2008, the Department of Environmental Affairs (DEA) and Department of Mineral Resources (DMR) agreed that environmental regulation of the mining industry would be removed from the purview of the MPRD Act and would be wholly regulated under the NEM Act. This was to be the 'One Environmental System' for the mining industry.

The implementation of this version of the 'One Environmental System' was given effect by the MPRD Amendment Act, 49 of 2008 and the NEM Amendment Act, 62 of 2008.

A three phased transition was envisaged under the above specified 2008 Amendment Acts. In phase 1, the status quo in respect of mining operations (i.e. the approval of an environmental authorisation under the NEM Act for triggered listed activities and an EMPR under the MPRD Act), was said to remain in place for 18 months following the enactment of the 2008 MPRD Amendment Act or the enactment of Section 2 of the 2008 NEM Amendment Act, whichever was the later enactment).

The 2008 MPRD Amendment Act was the later enactment. When it came into force on 7 June 2013, the three-phased transition started, and Phase 1 was said to come to an end on 8 December 2014. Phase 1 was to be followed by two further phases spanning a three-year period to move the environmental regulation entirely to NEM Act, under the guise of the DEA.

This approach was subsequently replaced by the enactment of the NEM Amendment Act, No. 25 of 2014 that came into force on 3 September 2014.

The new approach does away with the three-phased transition of environmental regulation, but rather on 3 September 2014 the shift in environmental regulation transitioned immediately to NEM Act, in terms of which environmental authorisations will need to be granted for mining operations. In addition to this, the DMR now retains its competence to regulate environmental management in the mining industry and now has the power to grant environmental authorisation for activities related to mining under the NEM Act.

Despite the coming into force of the NEM Amendment Act, 2014, the DEA issued a statement on 4 September 2014 to state that the transition can only be effectively implemented from 8 December 2014. Further, the Office of the Chief State Law Adviser issued a legal opinion on the effect of the commencement of the NEM Amendment Act, 2014, which also confirms that the transition can only be effectively implemented once all the complementary laws and regulations have come into force. The effective implementation is dependent upon the commencement of the MPRD Amendment Bill B15-2013. It is also dependant on Section 38B of the 2008 MPRD Amendment Act being brought into force. The enactment of this Section was delayed in the proclamation of the Amendment Act (Proclamation 17).

The transition is further dependant on the enactment of the proposed National Appeal Regulations, National Exemption Regulations, new EIA Regulations and listing notices and Financial Provisioning and Closure regulations under the NEM Act, certain regulations under the National Water Act (NWA) and the amendment of the MPRD Act Regulations to remove regulations relating to the environment.

Currently, National Appeal Regulations, National Exemption Regulations, new EIA Regulations and listing notices and Financial Provisioning and Closure regulations have been published in draft form under NEM Act, and the 'One Environmental System' began to be rolled out on 8 December 2014.

The Draft EIA Regulations have been published for comment and are intended to replace the 2010 regulations. The Draft EIA Regulations contain substantive changes to timeframes.

Ivanhoe Mines Ltd followed due process during the EIA and other permitting processes and all applications were completed complaint to the applicable Environmental Regulations.

4.3 Republic of South Africa Fiscal Environment

The taxes and royalties that apply to the Republic of South Africa are described in Section 22.2.2.

4.4 Project 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%
- K2014043829 (South Africa) (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 Platreef Communities Umbrella Trust, a trust established for the benefit of the members of the 20 communities/villages forming part of the Mokopane Community.

Employee Trust Co is 100% owned by the Platreef Equity Participation 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 4.3.

Figure 4.3 Ivanplats' Ownership Structure

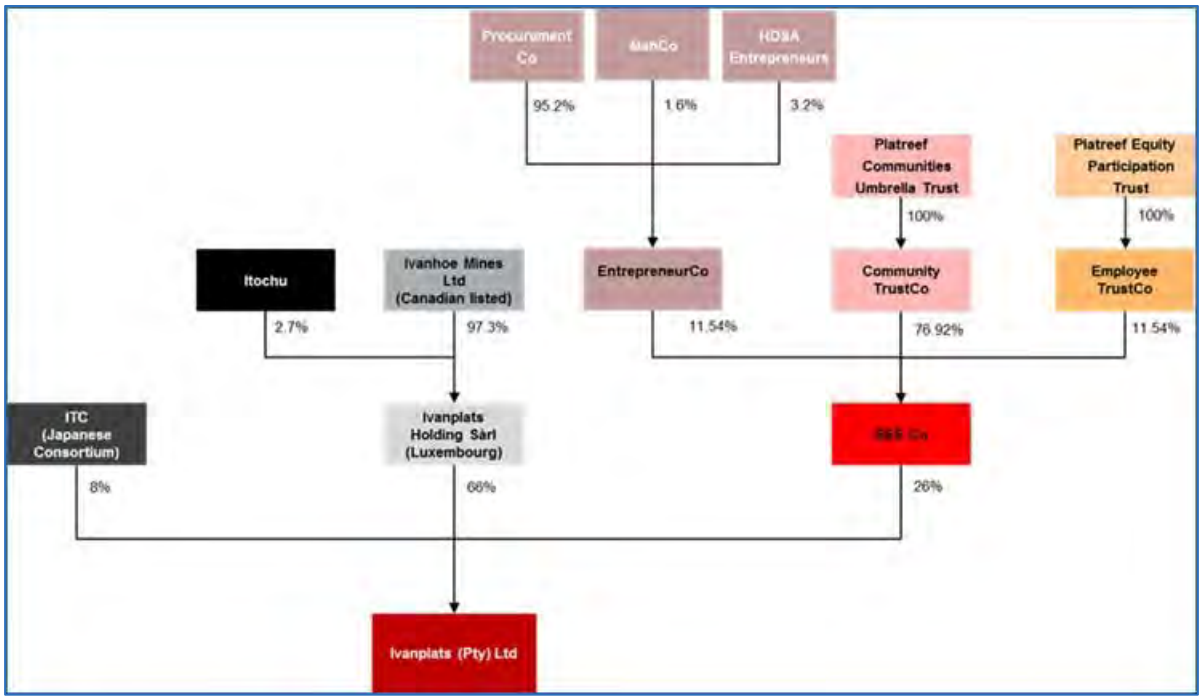


Figure by Ivanplats, 2017.

The shareholding structure of Ivanplats, and in particular the structure of BEE Co and its various shareholders, came about as the result of the successful implementation of Ivanplats' ground-breaking broad-based black economic empowerment (B-BBEE) transaction.

The final B-BBEE transaction 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 DMR as regulator. This process culminated in a transaction structure which has been approved by the DMR and which, as far as Ivanhoe is aware, is unique in South Africa in terms of the significant level of economic participation it affords to historically disadvantaged South Africans (HDSAs).

The main empowerment vehicle (BEE Co) is 100% owned by HDSAs in a structure that separately houses the interests of HDSAs and facilitates the funding of the acquisition of their shares, while shielding individuals from liability for repayment of the loans.

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.

Until such time as Ivanplats declares dividends to its shareholders, it pays an annual "trickle dividend" in the amount of R11 million to the Platreef Communities Umbrella Trust, enabling the trust to carry out community projects in accordance with its objectives. In cases where Ivanplats generates sufficient profit to declare dividends, vendor-funded shareholders will receive cash payments equal to 20% of the dividends to which they are entitled, with the remaining 80% being applied towards the repayment of the vendor loans until such time as the loans are repaid. From the time that the vendor loans have been repaid in full, all shareholders will receive their full dividends.

4.5 Encumbrances

The Ivanplats mining right contains the following 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;
- 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; 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.

4.6 Property Agreements

4.6.1 Anooraq (Atlatsa) Agreement

A 'Settlement and New Project Agreement', dated 11 December 2009, (the 2009 Anooraq Agreement), was concluded between Ivanhoe Nickel and Platinum Ltd and Anooraq Resources Corporation (Anooraq), now known as Atlatsa Resources Corporation (Atlatsa; the holding company of Plateau Resources). The 2009 Anooraq Agreement superseded and replaced respective rights and obligations of Ivanhoe and Anooraq under a 2001 Earn-in Agreement. Under the 2001 Earn-in Agreement, Anooraq had granted to Ivanhoe the right to earn a 50% participating interest in the Rietfontein prospecting licence.

The 2009 Anooraq Agreement also terminated arbitration and other proceedings, and created a new legal and business relationship between the two parties.

The 2009 Anooraq Agreement contained the following key elements:

- Anooraq contributed the Rietfontein PR and the Rietfontein 2 KS farm. Ivanhoe contributed a defined portion of the Turfspruit PR and the Turfspruit 241 KR farm. This combined area became known as the joint venture property (the Property). Under the agreement, both parties retain their existing prospecting rights in respect of mineral properties in their own names, but make these rights and technical information on the properties available to the joint venture.
- Both parties agreed to evaluate the possibility of development and open pit mining activity on the Property, and if supported by a positive feasibility study, to commence mining.
- The agreement envisaged that Ivanhoe would hold an initial interest of 94% in the Property, and Anooraq a 6% interest in the Property, provided that the joint venture contemplates an open pit mining operation that incorporates the Rietfontein mineral property.
- Ivanhoe is operator of the joint venture.
- For so long as Anooraq holds an interest in the joint venture, it is entitled to appoint a member to a technical committee, established to facilitate consultation and discussion with Ivanhoe with respect to joint venture operations.
- Expenditure during completion of a feasibility study will be borne by Ivanhoe. Anooraq would have no obligation to make any financial contribution, i.e. would be free-carried. This time-frame is termed the Carried Interest Period. During the Carried Interest Period, Anooraq must make payments to keep its prospecting permits in good standing and make other payments and filings as required to the South African government authorities to maintain its interest in the Rietfontein 2 KS farm.
- On completion of a feasibility study, Anooraq has two choices. The company can elect to contribute to expenditures in proportion to the initial interest held by Anooraq in the Property, in which case a new agreement, termed the Definitive Participation Agreement would come into effect. Where funding is less than the amount that would be expected in proportion to the initial interest, the company that is providing less funding would have its interest in the Property diluted, and the other party's interest in the Property would be increased by the same amount as that dilution. Alternatively, Anooraq can relinquish its interest in the Property for a 5% NSR royalty payable on any mineral products extracted from the Rietfontein prospecting lease.
- A BEE provision will be required to obtain a MR. In this instance, the 2009 Anooraq Agreement states that Anooraq will not be obliged to reduce its interest in the Property, but that Ivanhoe's interest will be reduced. From an ownership perspective, a BEE requires that at least 26% of the holder of the rights is owned and controlled by HDSAs. Anooraq may increase its Property interest as part of a BEE transaction if this is in accordance with appropriate South African laws, and if Anooraq meets the local ownership requirements for a BEE transaction.

A provision was made within the 2009 Anooraq Agreement for potential underground mining activities. In this instance, the agreement states:

If, and to the extent that, a feasibility study contemplates the extraction of mineral products from both the Turfspruit property and the Rietfontein property by way of sub-surface mining, and at the conclusion of the Carried Interest Period, Anooraq elects to maintain its interest in the project as a participating interest, the Property will be deemed to include those areas of the Turfspruit property and the Rietfontein property, respectively, from which the feasibility study contemplates that mineral products will be extracted by way of sub-surface mining, and the respective interests of the parties will be adjusted:

(a) in the case of the interest of Anooraq, by dividing the total value of mineral products that the feasibility study contemplates will be extracted exclusively from the Rietfontein property by the total value of mineral products that the feasibility study contemplates will be extracted from the entire project property and multiplying the resulting quotient by 100; and

(b) in the case of the interest of Ivanhoe, by dividing the total value of mineral products that the feasibility study contemplates will be extracted exclusively from the Turfspruit property by the total value of mineral products that the feasibility study contemplates will be extracted from the entire project property and multiplying the resulting quotient by 100.

Under clause (b), any proposed underground mine that extracts 100% of its mineral products from the Turfspruit property would result in a 100% interest for Ivanhoe.

Legal opinion indicates that while the settlement agreement that was entered into between Ivanhoe and Anooraq remains a valid and binding agreement, it is not legally competent for a holding company to create rights and obligations for a subsidiary under South African law. This opinion noted that in order to give effect to the provisions of the settlement agreement, the holders of the PR will be required to enter into a separate agreement. Depending on the structure and provisions of that agreement, it will require various consents in terms of Section 11 and Section 102 of the MPRD Act.

It is a reasonable expectation that, at the current stage of project knowledge, such agreements could be enacted, and therefore that declaration of Mineral Resources on PR LP30/5/1/1/2/872PR can be supported.

Ivanhoe has advised that an offer has been made to Atlatsa to outright purchase the Atlatsa rights to the Rietfontein 2 KS farm.

4.7 Environmental Studies

Information on environmental studies is based on the studies by Ivanhoe and Digby Wells and is included in Section 20.

4.8 Permits

Permits to support mine development activities are discussed in Section 20.

4.8.1 Current Permits

Ivanhoe advised that exploration activities have been conducted in compliance with applicable laws in South Africa with the exception that the existing Prospecting Works Programme and EMPR were contravened when more drillholes were completed than the total number of drillholes granted to be drilled in the permits. Ivanhoe subsequently submitted a Section 102 application for approval of an amended Environmental Management Plan (EMPlan) and Prospecting Works Programme in order to accommodate future prospecting. The Section 102 application was submitted to the DMR on 16 May 2012 and subsequently replaced with a revised application to amend lodged together with a bulk sampling application as lodged with the authorities on 21 September 2012 (see Section 4.10.2). The application was approved on 29 August 2013.

On 26 October 2012, the DMR served Ivanhoe with a directive in terms of Section 93 of the MPRD Act. The directive ordered Ivanhoe to cease all prospecting operations pending the conclusion of new surface use agreements with the occupants of the land (communities) in the presence of the Department of Rural Development and Land Reform (DRDLR). On 28 May 2013, the DMR notified Ivanhoe that it had satisfied the directives imposed on it and lifted the order dated 26 October 2012, allowing Ivanhoe to continue with its prospecting operations.

4.9 Significant Risk Factors

There are two significant permitting risks to project development: resettlement of township occupants, and renewal of the Rietfontein licence. The resettlement risk is reduced where underground mining operations are conducted versus open pit operations.

4.10 Comments on Section 4

In the opinion of the QP Bernard Peters, the information discussed in this Section supports the Platreef 2017 FS. The following key points are of note:

- Information provided by legal experts and Ivanhoe supports Ivanhoe's ownership claims to PR MPT55/2006PR. Ivanhoe applied for a MR on 6 June 2013, prior to expiry of PR MPT55/2006PR in May 2014.
- A JV with Atlatsa over the Rietfontein 2 KS farm is covered under the terms of the 2009 Anooraq Agreement, and is currently in force.
- Ivanhoe and Atlatsa may need to prepare additional legal agreements between the South African-registered subsidiary companies to meet South African law with respect to the 2009 Anooraq Agreement.
- Ivanhoe advised that it has submitted an offer to outright purchase the rights to Rietfontein 2 KS from Atlatsa. Atlatsa is still considering the offer.
- Rietfontein 2 KS is critical to open pit mining on Turfspruit 241 KR. Underground mining is not contemplated on Rietfontein 2 KS; however, some of the necessary infrastructure to support such activities, such as a tailings dam, may be located within the Rietfontein 2 KS area.

- Should an open pit operation be envisaged, there will likely be mine disturbances associated with the development and mining of any open pit projecting beyond the current Platreef Project boundary, particularly onto the adjacent Tweefontein farm. Arrangements would have to be made with adjacent landowners in this instance. Mining lease applications require appropriate supporting documentation, including completion of a scoping report, EIA, development of an environmental management programme, and a requirement to meet B-BBEE provisions.
- The Right MR was granted in favour of Ivanplats on 30 June 2014, and notarially executed on 4 November 2014, signifying the formal activation of the MR. The MR will continue to be in force until 3 November 2044. The MR allows a company to mine and process minerals optimally from the mining area for a maximum period of 30 years, which may be extended upon application for further periods, each of which may not exceed 30 years at a time.
- Surface rights within the areas of the Rietfontein 2 KS, Macalacaskop 243 KR, and Turfspruit 241 KR farms belongs to the national government. 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 Mamashala 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 activities to date have been conducted within the regulatory framework required by the South African Government.
- Based on information discussed in Section 20 of the Report, collection of baseline environmental data has commenced. The current state of knowledge on environmental and permit status for the Platreef Project supports the declaration of Mineral Resources. Additional permits will be required for project development.
- A gazetted land claim has been lodged over the Rietfontein 2 KS farm; information provided to Ivanhoe by the DRDLR indicates a non-gazetted claim by the Mokopane Tribe over the area covered by PR MPT76/2007PR.
- Should an open pit mining scenario be considered, provision will need to be made for relocation of villages and infrastructure that exists in the likely footprint area of an open pit mine. The impact of an underground operation will involve a smaller surface area, (which could be mitigated if fill were introduced after mining to mitigate subsidence), so that there are likely to be fewer relocation requirements.
- There have been instances where drilling programmes have been affected by short-term access issues, most recently in 2012. Over the more than 15 years that Ivanhoe has been conducting exploration activities, the company has previously managed to reach resolutions such that the planned work has been able to be completed.

- While additional consultation may be necessary with the communities and the DRDLR to allow for future work programmes, this consultative process is reasonably well understood. The lack of internal community agreement, however, remains as a risk to the Platreef Project.
- 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.

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°C to 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°C to 18°C.

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

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

5.3 Local Resources

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

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

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 are employed in the mining sector. The Ivanplats SLP makes provision for extensive training programs to train the local communities to develop the necessary skills. Skilled trade positions and professional staff will have to be recruited from outside the area.

5.5 Infrastructure

5.5.1 Power Supply

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

The South African electricity utility Eskom Holdings SOC Limited (Eskom) has advised that sufficient power is not available at present in the Mokopane area due to transmission line limitations and generating shortfalls. Medupi's first unit (1 of 6) of 800 MW was brought online in August 2015. Since this unit has been online, South Africa has not experienced any load shedding.

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

Ivanplats has requested that Eskom complete the budget quote study for 70 MVA that considers a premium supply option. The Environmental Impact Assessment (EIA) process is complete, and the design fees for the 70 MWA have been paid. The latest forecast energisation date of the Platreef Eskom incoming substation is H2 '18.

The Platreef Project power requirement for a 4 Mtpa underground mine has been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats is in the process of notifying and requesting from Eskom the 30 MVA additional power demand for the Platreef Project.

As power is required for the initial mine development (shaft sinking) prior to the main power supply being available, an agreement for 5 MVA of temporary construction power was concluded with Eskom. This power will be supplied from the Mahwelereng 33/11 kV substation, which is located 7 km from the Platreef Project site.

The 5 MVA construction power supply has been concluded and is currently in use.

Any power requirements prior to and exceeding the supply of temporary construction power will be supplied by diesel generated sets.

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 is a participant in the Olifants River Water Resource Development Project (ORWRDP), which is designed to deliver water for domestic and industrial (mining) purposes to the Eastern and Northern Limbs of the Bushveld Complex. Ivanplats is also a member of the Joint Water Forum (JWF), which facilitates and coordinates discussions with the various participants in the water scheme. These participants were required to indicate their projected water requirements from the scheme in order for the total capacity to be determined. This was done, and the overall capacity required for the scheme is made up of 62 MLpd for domestic use and 78 MLpd for industrial projects i.e., a total of 140 MLlpd.

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

The Department of Water Affairs (DWA) has stated that all water for the Northern Limb (including any potential mining operation on the Platreef Project) would be supplied through the ORWRDP. A number of possible water sources to augment the supply system have been investigated, and most promising is acid mine drainage (AMD) from the Witbank coalfields. Another possible source is the transfer of water from The Vaal river system. Either of these sources will be treated and pumped into the Olifants River.

Another potential short-term source of water is ground water in the Platreef Project area. Ground water sources have been identified, and water-use licenses for a number of boreholes on Uitloop 3 KS farm were authorised by the DWA during 2014 (Figure 5.2).

Figure 5.1 Location Plan Flag Boshielo Dam and Proposed Water Pipeline

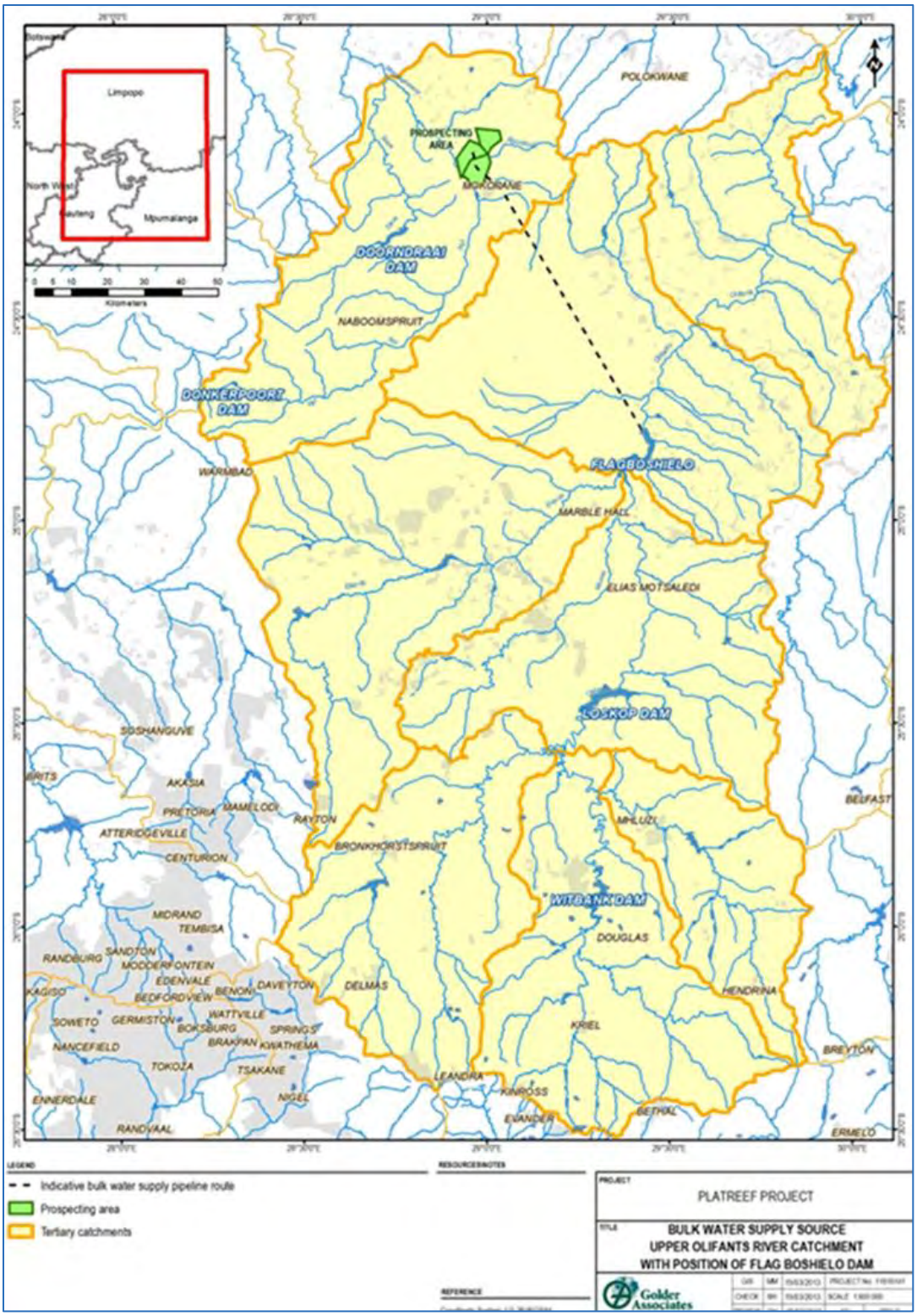


Figure courtesy Ivanhoe, 2013.

Figure 5.2 Water Bore Location Plan

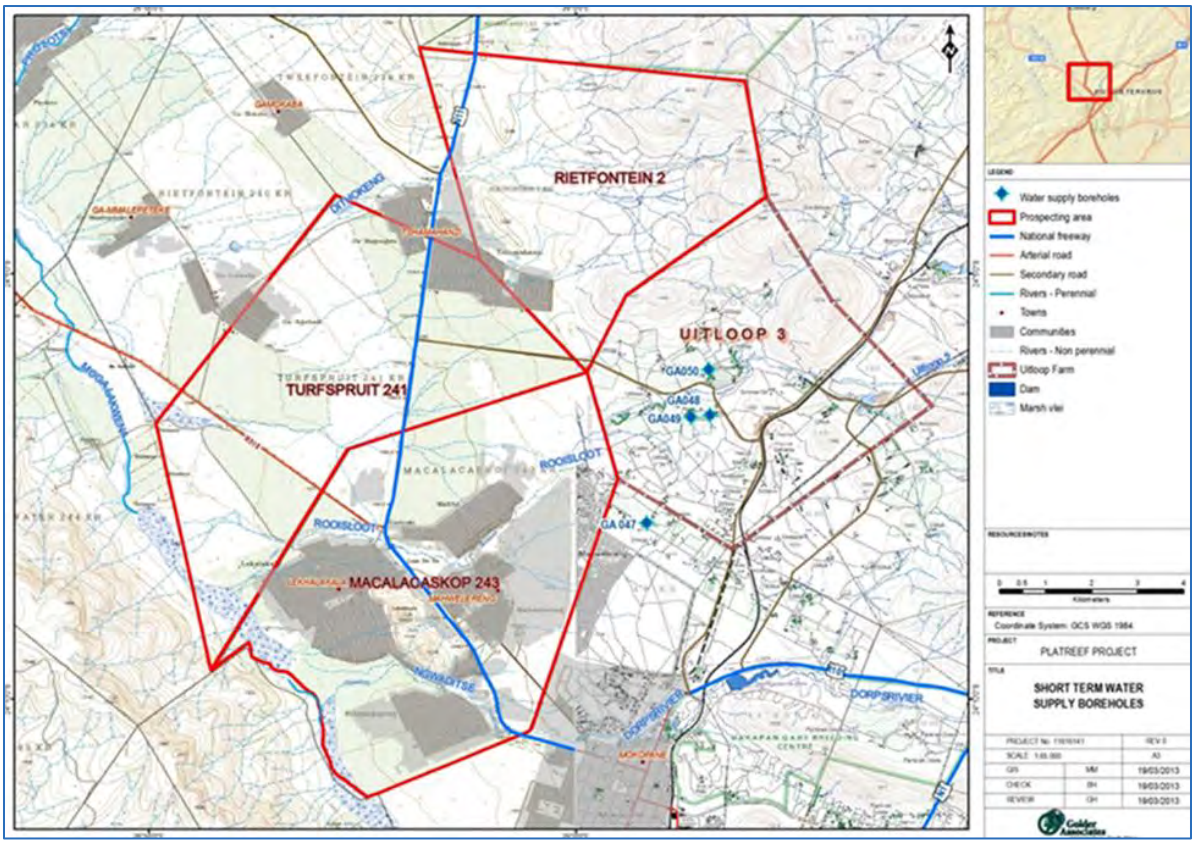


Figure courtesy Ivanhoe, 2013.

5.5.3 Access Roads

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

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

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

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

5.6 Physiography

The Rietfontein 2 KS, Macalacaskop 243 KR, and Turfspruit 241 KR farms are located in a broad valley on flat terrain with a gradual westerly slope. There is very little topographic relief on the farms; however, to the east and west of the farms, semi-parallel, north-south trending high ridges flank the valley floor. A portion of the eastern ridge system trends onto the Rietfontein 2 KS farm, adjacent to Turfspruit 241 KR. Figure 5.3 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 5.3 Platreef Project Physiography

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

5.7 Sufficiency of Surface Rights

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

6 HISTORY

6.1 Previous Work

During the 1970s, regional exploration was undertaken over the Platreef mineralised zone (the Platreef) by Rustenberg Platinum Holdings Limited (Rusplats), a wholly-owned subsidiary of Anglo American Platinum Corporation (Amplats). Rusplats reportedly drilled several widely-spaced drillholes along the Platreef on Turfspruit 241 KR and Macalacaskop 243 KR farms. This drilling followed-up earlier work by the predecessor of Amplats during the 1960s. No data from either of these programmes were available for the Platreef 2017 FS.

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

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

The initial exploration focus was on delineation of mineralisation that could support open pit mining. From 2003–2007, Ivanhoe undertook studies involving concentrator/smelter options, metallurgical test work, 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 and 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, Parker et al., 2016).

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. A detailed description of the open-pit resource is available 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 is available in the September 2012 Technical Report (Parker et al., 2012).

6.2 Platreef 2014 PEA

Ivanhoe's development plan for the Platreef Project considers three phases of underground mining and concentrator expansion that were identified in the Platreef 2014 PEA. The Platreef 2014 PEA is a PEA as defined in NI 43-101 Standards of Disclosure for Mineral Projects, and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability but have reasonable prospects for eventual economic extraction. The development scenarios that were identified in the Platreef 2014 PEA are:

- Phase 1 Concentrator 4 Mtpa.
- Phase 2 Concentrator 8 Mtpa.
- Phase 3 Concentrator 12 Mtpa.

The base case for the Platreef 2014 PEA analysis was Phase 2 the 8 Mtpa concentrator case. The development scenarios and additional options for the Platreef Project are shown in Figure 6.1. The development scenarios describe a staged approach where there is opportunity to expand the operation depending on demand, smelting and refining capacity and capital availability. As Phase 1 is developed and taken into production there is opportunity to modify and optimise the definition of Phases 2 and 3. This would allow changes to the timing or expansion capacity to suit the conditions at the time.

The options for a smelter and or a base metal refinery (BMR) are still being studied, and their timing and sizing need to undergo further analysis. Opportunities for additional phases after Phase 3 may be available, and these will also require additional investigation.

Phase 1 includes the construction of a concentrator and other associated infrastructure to support a start-up to production at a nominal plant capacity of 4 Mtpa. Phase 2 includes an additional ramp-up to a plant capacity of 8 Mtpa. Phase 3 envisages a further ramp-up to a plant capacity of 12 Mtpa. All production is sourced from underground mining.

The three phases were costed and the economic analyses reported in the Platreef 2014 PEA. Each phase has the same underlying plan for the construction and operation of a concentrator processing facility, with the capacity aligned to the requirements of each phase. Infrastructure constructed to support the mine is also common to all phases. A comparison of the production and key financial results including NPV at 8% discount rate (NPV8) and Internal Rate of Return (IRR) of the Platreef 2014 PEA is shown in Table 6.1.

Figure 6.1 Platreef Project Development Scenarios

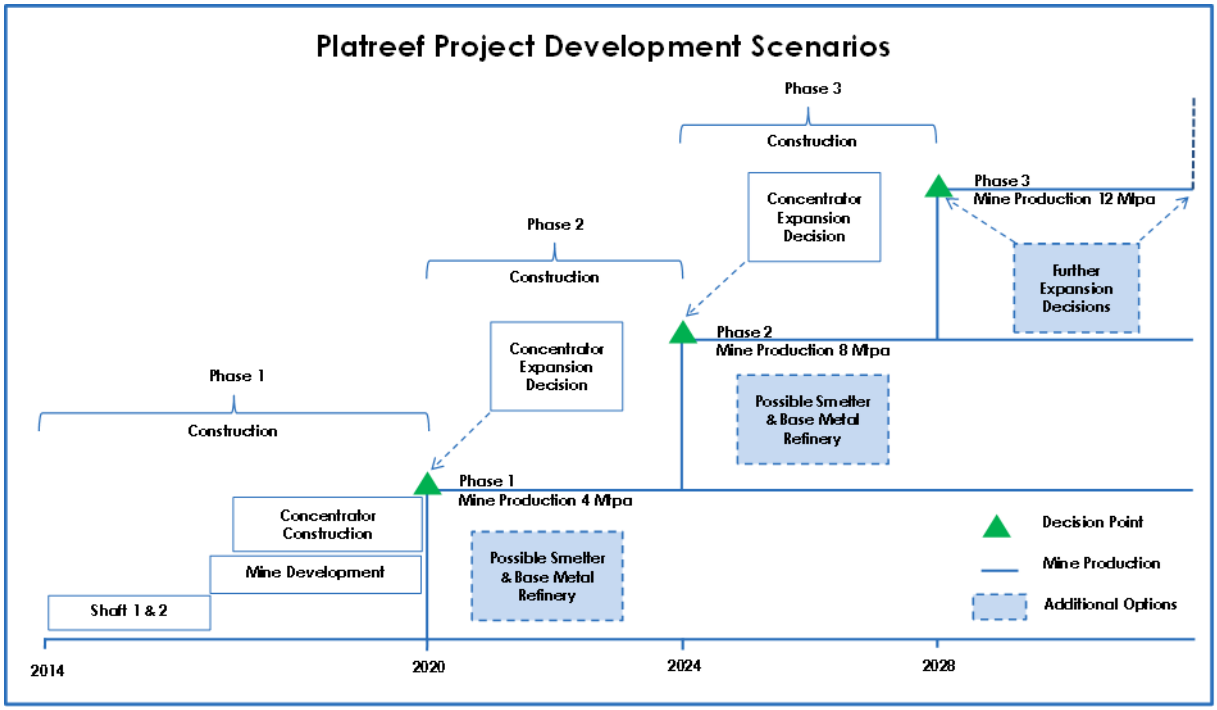


Figure by OreWin, 2014.

Table 6.1 Platreef 2014 PEA Results Table Caption

Item	Units	Phase 1	Phase 2	Phase 3
Milling Rate	Mtpa	4	8	12
Mined	Mt	117	219	310
Milled	Mt	117	219	310
Platinum	g/t	1.84	1.70	1.71
Palladium	g/t	1.93	1.78	1.77
Gold	g/t	0.27	0.27	0.27
Rhodium	g/t	0.13	0.12	0.12
Copper	%	0.16	0.16	0.16
Nickel	%	0.34	0.35	0.34
Concentrate	kt	4,665	8,771	12,396
Platinum	g/t	40.5	37.0	37.3
Palladium	g/t	42.4	38.7	38.4
Gold	g/t	5.3	5.2	5.2
Rhodium	g/t	2.8	2.6	2.6
Copper	%	3.6	3.6	3.5
Nickel	%	5.8	6.0	5.8
Platinum (Recovered Metal)	koz	6,075	10,444	14,864
Palladium (Recovered Metal)	koz	6,362	10,915	15,294
Gold (Recovered Metal)	koz	789	1,462	2,065
Rhodium (Recovered Metal)	koz	413	742	1,050
Copper (Recovered Metal)	Mlb	368	695	957
Nickel (Recovered Metal)	Mlb	599	1,160	1,582
Life of Mine	years	30	30	30
Pre-Production Capital	US$M	1,525	1,719	1,769
Mine Site Cash Cost	US$/oz 3PE+Au	412	425	441
Total Cash Costs After Credits	US$/oz 3PE+Au	367	341	371
Site Operating Costs	US$/t Milled	48.22	45.63	47.39
After Tax NPV8	US$M	897	1,620	2,179
After Tax IRR	%	13	14	15
Project Payback Period	years	5.59	6.40	7.55

1. The economic analysis is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised.
2. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability but have reasonable prospects for eventual economic extraction.
3. Price assumptions of US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$1,700/oz rhodium US$3.00/lb for copper, and US$8.35/lb nickel.
4. Each Phase has a 30 year life.
5. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.

6.2.1 Platreef 2014 PEA Financial Results Summary

The economic analysis uses price assumptions of US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$1,700/oz rhodium, US$3.00/lb for copper, and US$8.35/lb nickel. The prices are based on a review of consensus price forecasts from financial institutions and similar studies that were published in 2014. The basis of the operational framework of the mine used in the analysis is Republic of South Africa legislation.

Comparison of the results of the financial analysis for each phase shows that there is a progressive increase in NPV for the three phases. The After Tax NPV8, IRR, and project payback period for each phase are shown in Table 6.2 and Figure 6.2. The After Tax NPV8 for Phase 1 is US$897M, for Phase 2 it is US$1,620M, and for Phase 3 it is US$2,179M. There is an increase in IRR from Phase 1 to Phase 2 and from Phase 2 to Phase 3. As the phased expansions progress the payback period increases as capital is committed over a longer time horizon.

Table 6.2 Platreef 2014 PEA After Tax Financial Results

		Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
NPV8 (US$M)	Undiscounted	6,992	12,527	17,078
	5%	2,040	3,593	4,818
	8%	897	1,620	2,179
	10%	449	868	1193
	12%	149	374	554
	15%	-133	-77	-17
IRR		13.37%	14.34%	14.88%
Project Payback Period	(Years)	5.59	6.40	7.55

Figure 6.2 Platreef 2014 PEA After Tax Npv8

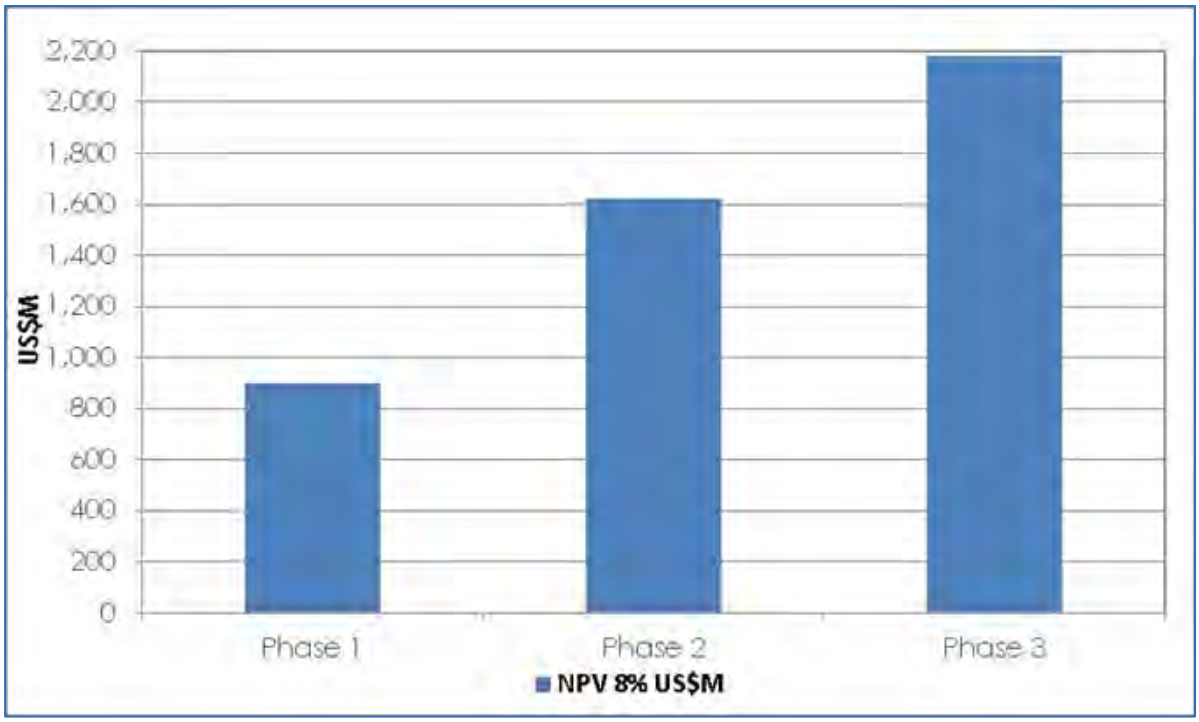


Figure by OreWin, 2014.

Cash flow increases with each concentrator expansion. There is an increase in IRR from Phase 1 through to Phase 3, and the payback period also increases with each phase. The cumulative cash flow after tax for each phase over the years can be seen in Figure 6.3 and Figure 6.4.

Figure 6.3 Platreef 2014 PEA Cumulative Cash Flow After Tax

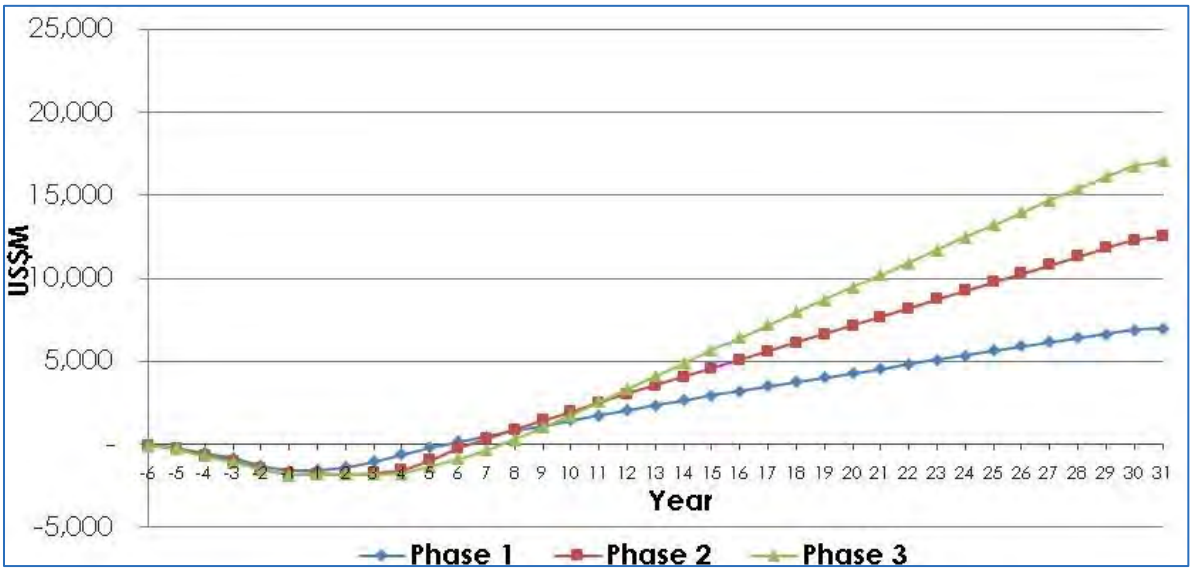


Figure by OreWin, 2014.

Figure 6.4 Platreef 2014 PEA Cumulative Cash Flow After Tax (Initial Years)

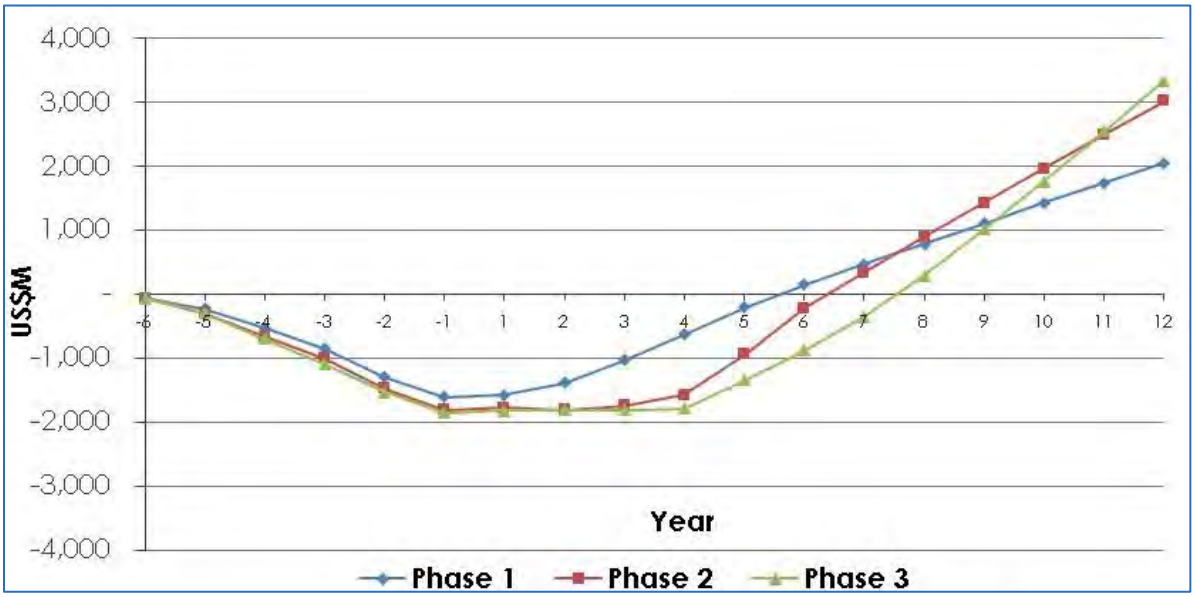


Figure by OreWin, 2014.

6.2.2 Platreef 2014 PEA Production Summary

The key average annual production results over the 30 year mine life are shown in Table 6.3.

Table 6.3 Platreef 2014 PEA Production Summary

		Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
Total Mined and Processed (30 years)	Mt	117	219	310
Platinum	g/t	1.84	1.70	1.71
Palladium	g/t	1.93	1.78	1.77
Gold	g/t	0.27	0.27	0.27
Rhodium	g/t	0.13	0.12	0.12
Copper	%	0.16	0.16	0.16
Nickel	%	0.34	0.35	0.34
Recoveries (Life of Mine Average)				
Platinum Recovery	%	88.21	87.15	87.24
Palladium Recovery	%	87.63	86.85	86.77
Gold Recovery	%	76.69	76.72	76.72
Rhodium Recovery	%	85.92	86.62	86.62
Copper Recovery	%	87.89	87.90	87.84
Nickel Recovery	%	69.13	69.47	69.05
Concentrate Produced (Life of Mine Average Annual Production)				
Concentrate	ktpa	156	292	413
Platinum	g/t	40.5	37.0	37.3
Palladium	g/t	42.4	38.7	38.4
Gold	g/t	5.3	5.2	5.2
Rhodium	g/t	2.8	2.6	2.6
3PE+Au	g/t	90.9	83.6	83.5
Copper	%	3.6	3.6	3.5
Nickel	%	5.8	6.0	5.8
Metal Sold (Life of Mine Average Annual Production Metal Units per Year)				
Platinum	koz	203	348	495
Palladium	koz	212	364	510
Gold	koz	26	49	69
Rhodium	koz	14	25	35
3PE+Au	koz	455	785	1,109
Copper	Mlb	12	23	32
Nickel	Mlb	20	39	53

6.2.3 Platreef 2014 PEA Capital and Operating Cost Summary

The pre-production capital cost, including contingency, for each phase is provided in Table 6.4. The sustaining and expansion and total capital costs are shown in Table 6.5 and Table 6.6. The operating costs are summarised in Table 6.7 to Table 6.9. The cash costs denominated in US$/oz Payable 3PE+Au show that Phase 2 has a slightly lower cash cost than the other phases.

Table 6.4 Platreef 2014 PEA Pre-Production Capital Cost

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$M	US$M	US$M
Mining			
Underground	540	633	673
Capitalised Pre-production	24	24	25
Subtotal	564	657	698
Processing			
Concentrator	201	201	201
Subtotal	201	201	201
Infrastructure			
Bulk Water/Power	76	76	76
Tailings Dam	39	46	39
General Infrastructure	29	29	29
Closure Costs	–	–	–
Subtotal	144	151	144
Indirects			
Drilling and Studies	–	19	19
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing and Infrastructure: EPCM	37	37	37
Subtotal	172	207	211
Owners Cost			
Capitalised G&A	26	26	26
Mining	60	79	79
Processing and Infrastructure	17	18	17
Subtotal	103	123	122
Capital Expenditure Before Contingency	1,185	1,338	1,376
Mining Contingency	221	259	272
Processing and Infrastructure Contingency	120	122	120
Capital Expenditure After Contingency	1,525	1,719	1,769

Table 6.5 Platreef 2014 PEA Sustaining and Expansion Capital Cost

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$M	US$M	US$M
Mining			
Underground	679	1,524	2,347
Capitalised Pre-production	–	–	–
Subtotal	679	1,524	2,347
Processing			
Concentrator	103	383	652
Subtotal	103	383	652
Infrastructure			
Bulk Water/Power	8	49	90
Tailings Dam	–	3	49
General Infrastructure	3	3	3
Closure Costs	14	19	30
Subtotal	26	75	173
Indirects			
Drilling and Studies	–	–	19
Mining: Indirects	–	–	–
Mining: EPCM	–	–	–
Processing and Infrastructure: EPCM	4	33	63
Subtotal	4	33	81
Owners Cost			
Capitalised G&A			
Mining	–	–	–
Processing and Infrastructure	2	14	29
Subtotal	2	14	29
Capital Expenditure Before Contingency	814	2,029	3,282
Mining Contingency	124	354	572
Processing and Infrastructure Contingency	36	146	266
Capital Expenditure After Contingency	974	2,528	4,120

Table 6.6 Platreef 2014 PEA Total Capital Cost

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$M	US$M	US$M
Mining			
Underground	1,219	2,157	3,020
Capitalised Pre-production	24	24	25
Subtotal	1,243	2,181	3,045
Processing			
Concentrator	305	584	854
Subtotal	305	584	854
Infrastructure			
Bulk Water/Power	84	125	166
Tailings Dam	39	49	89
General Infrastructure	32	32	32
Closure Costs	14	19	30
Subtotal	169	225	317
Indirects			
Drilling and Studies	–	19	38
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing and Infrastructure: EPCM	41	70	99
Subtotal	177	240	292
Owners Cost			
Capitalised G&A	26	26	26
Mining	60	79	79
Processing & Infrastructure	19	32	46
Subtotal	105	137	151
Capital Expenditure Before Contingency	1,998	3,367	4,659
Mining Contingency	344	613	844
Processing and Infrastructure Contingency	156	268	386
Capital Expenditure After Contingency	2,499	4,247	5,888

Table 6.7 Platreef 2014 PEA Unit Operating Costs

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$/oz Payable 3PE+Au		
Mine Site Cash Cost	412	425	441
Realisation Cost	402	416	413
Total Cash Costs Before Credits	814	840	854
Copper Credits	-81	-89	-86
Nickel Credits	-367	-411	-397
Total Cash Costs After Credits	367	341	371

Table 6.8 Platreef 2014 PEA Total Operating Costs and Revenues

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$M	US$M	US$M
Revenue			
Gross Sales Revenue	23,375	41,644	58,358
Less: Realisation Costs			
Transport	334	628	887
Refining Charges	4,207	7,540	10,577
Government Royalty	938	1,628	2,261
Total Realisation Costs	5,479	9,796	13,726
Net Sales Revenue	17,896	31,849	44,632
Site Operating Costs			
Mining	3,755	7,109	10,874
Processing and Tailings	1,251	2,226	3,096
G&A	618	670	717
Total	5,624	10,006	14,687
Operating Margin	12,273	21,843	29,945

Table 6.9 Platreef 2014 PEA Production Summary

	Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
	US$/t Milled	US$/t Milled	US$/t Milled
Revenue			
Gross Sales Revenue	200.41	189.92	188.31
Less: Realisation Costs			
Transport	2.86	2.86	2.86
Refining Charges	36.07	34.39	34.13
Government Royalty	8.04	7.42	7.30
Total Realisation Costs	46.97	44.67	44.29
Net Sales Revenue	153.44	145.25	144.02
Site Operating Costs			
Mining	32.19	32.42	35.09
Processing and Tailings	10.73	10.15	9.99
G&A	5.30	3.06	2.31
Total	48.22	45.63	47.39
Operating Margin	105.22	99.62	96.63

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

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

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

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

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

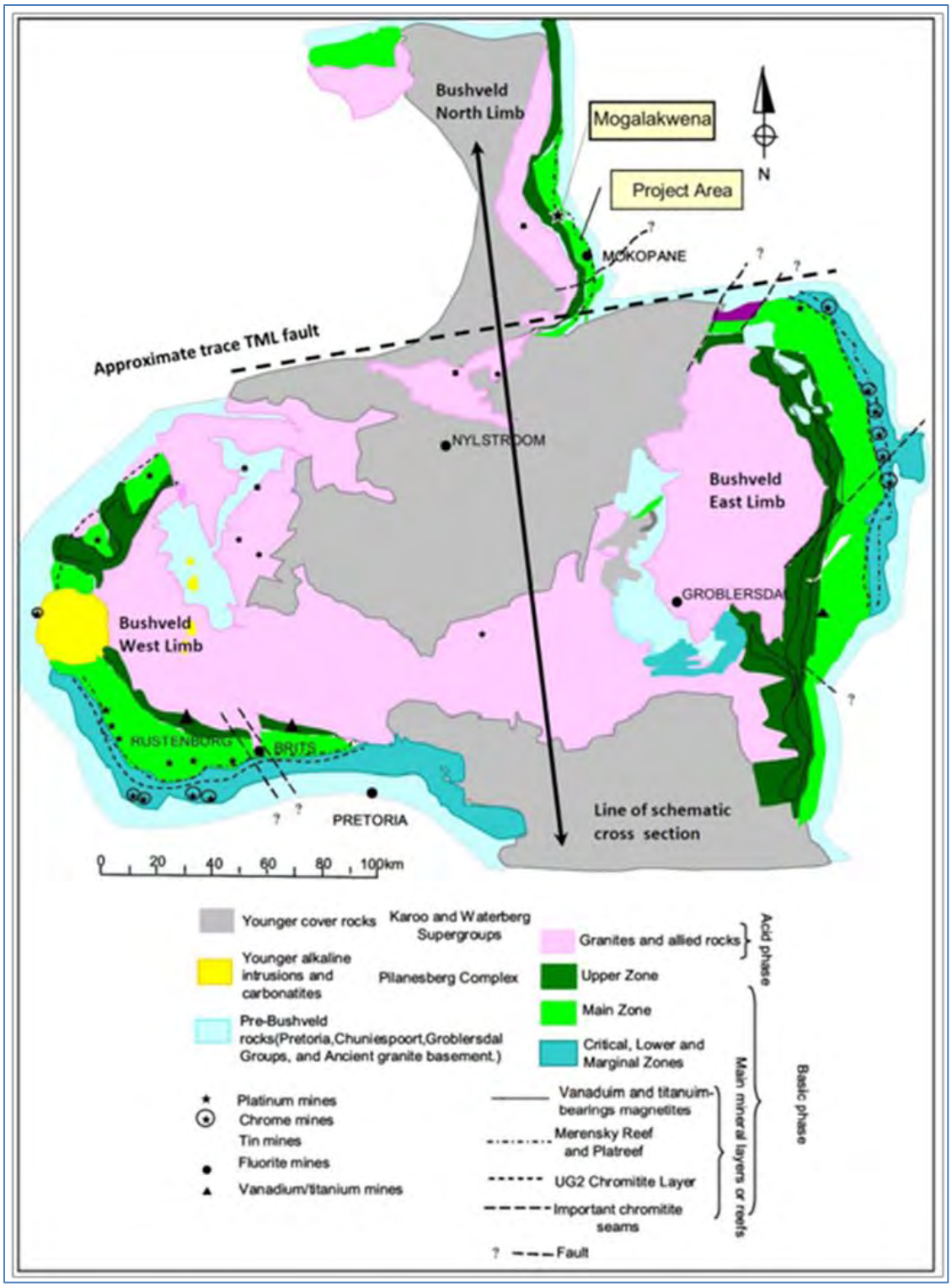
In the East and West Limbs of the BIC, the RLS was intruded into the Magaliesberg Formation of the Proterozoic Transvaal Supergroup. In the North Limb, the RLS intrudes progressively older country rocks northward (Magaliesberg Formation, Malmani Subgroup and Duitschland Formation). Figure 7.1 provides a location and regional geology map for the BIC. Figure 7.2 provides a diagrammatic cross-section through the BIC.

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

7.2 Project Geology

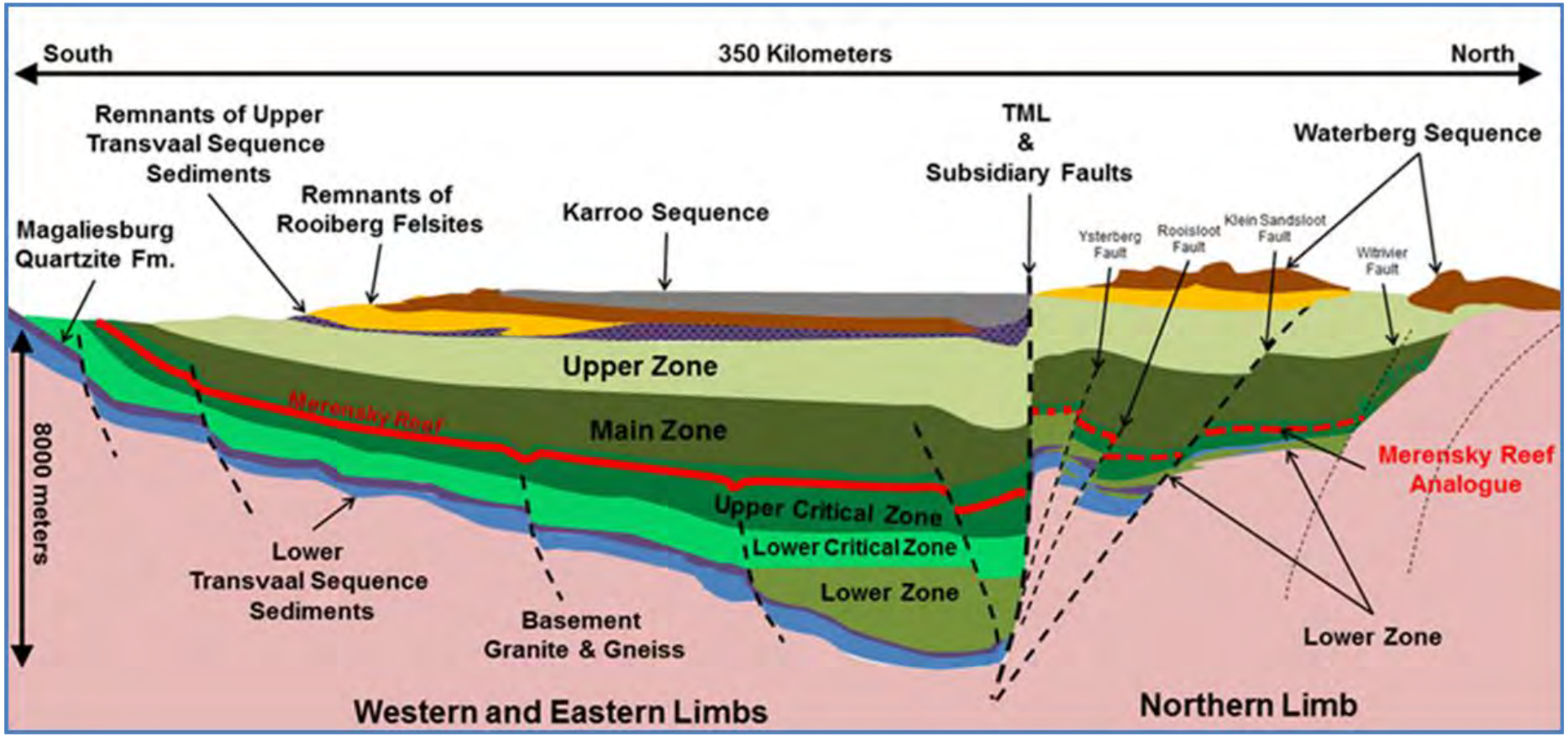
Figure 7.3 shows the Project geology projected to surface. The locations of Zones 1 to 5 referred to in the following geologic discussion are shown in Figure 7.4.

Figure 7.1 Regional Geological Plan of The Bushveld Complex



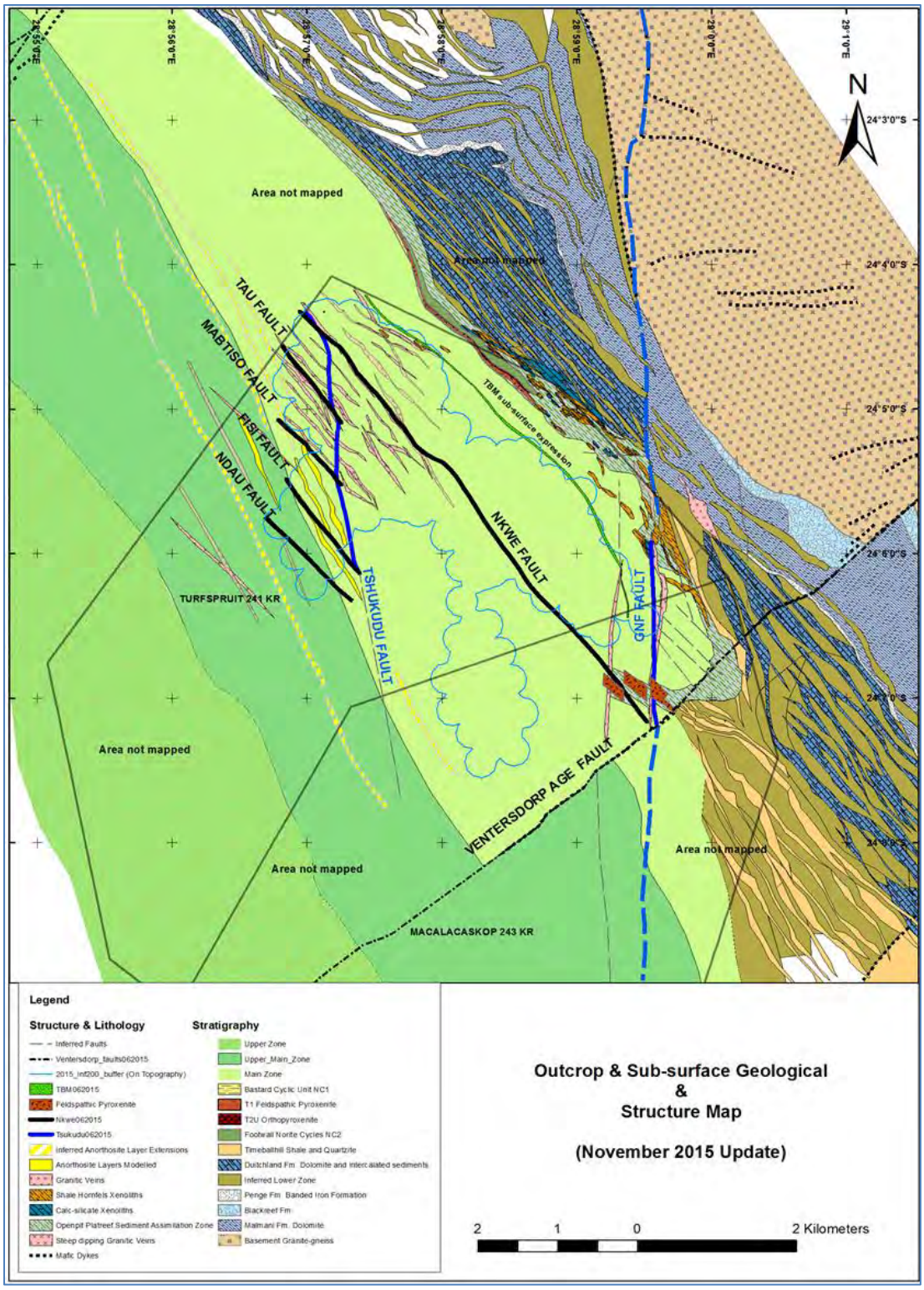
Modified after Viljoen and Schürmann, 1998; section line represents location of section in Figure 7.2

Figure 7.2 Schematic Cross-Section Through the Bushveld Igneous Complex



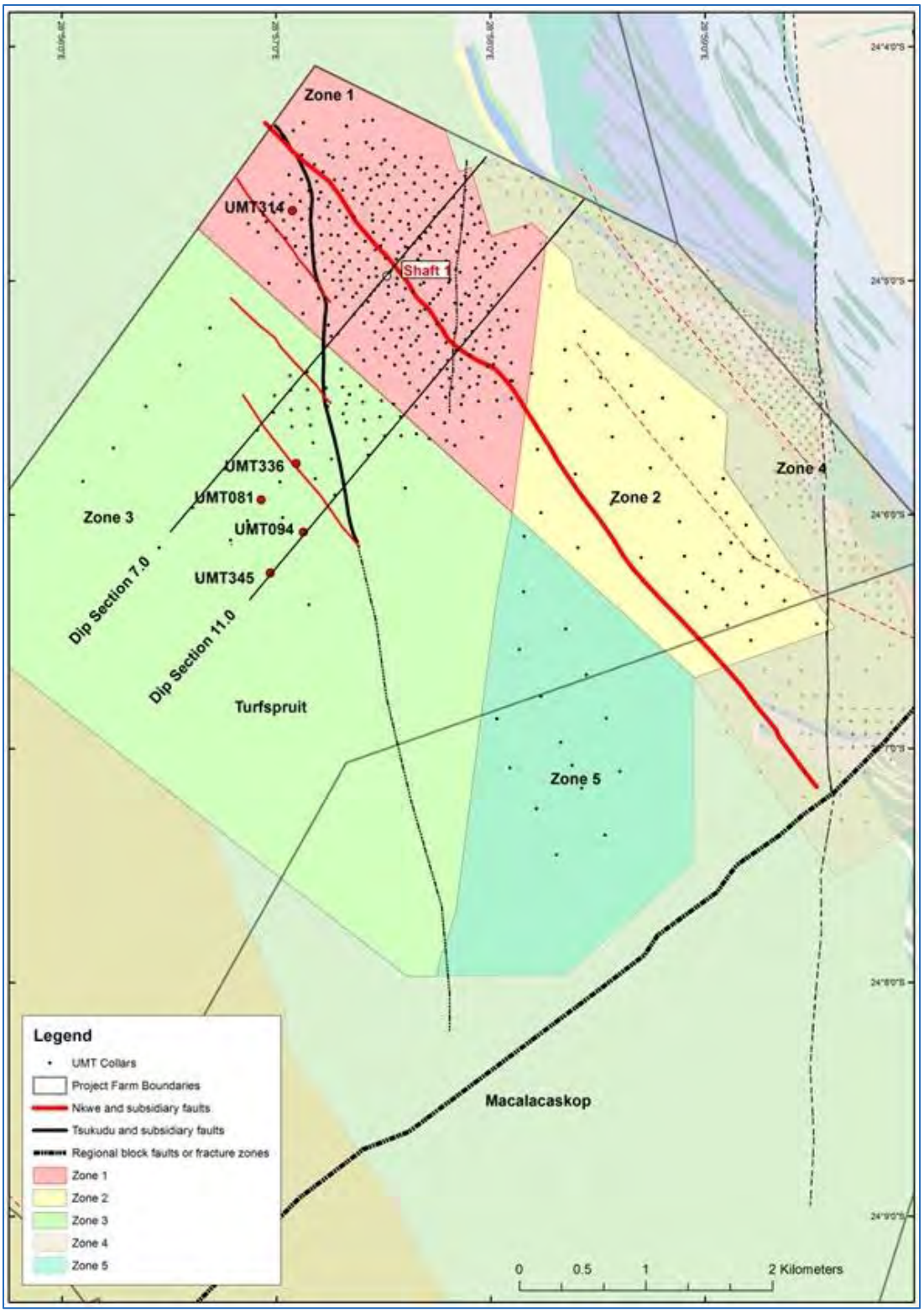
Courtesy Ivanhoe, 2012; modified after Kruger, 2005. Figure is schematic and not to scale. Section line illustrated is shown on Figure 7.1.

Figure 7.3 Project Geology Plan



Courtesy Ivanhoe 2016.

Figure 7.4 Project Exploration Zones Plan



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

7.2.1 Introduction

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

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

7.2.2 Stratigraphic Correlations and Nomenclature

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

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

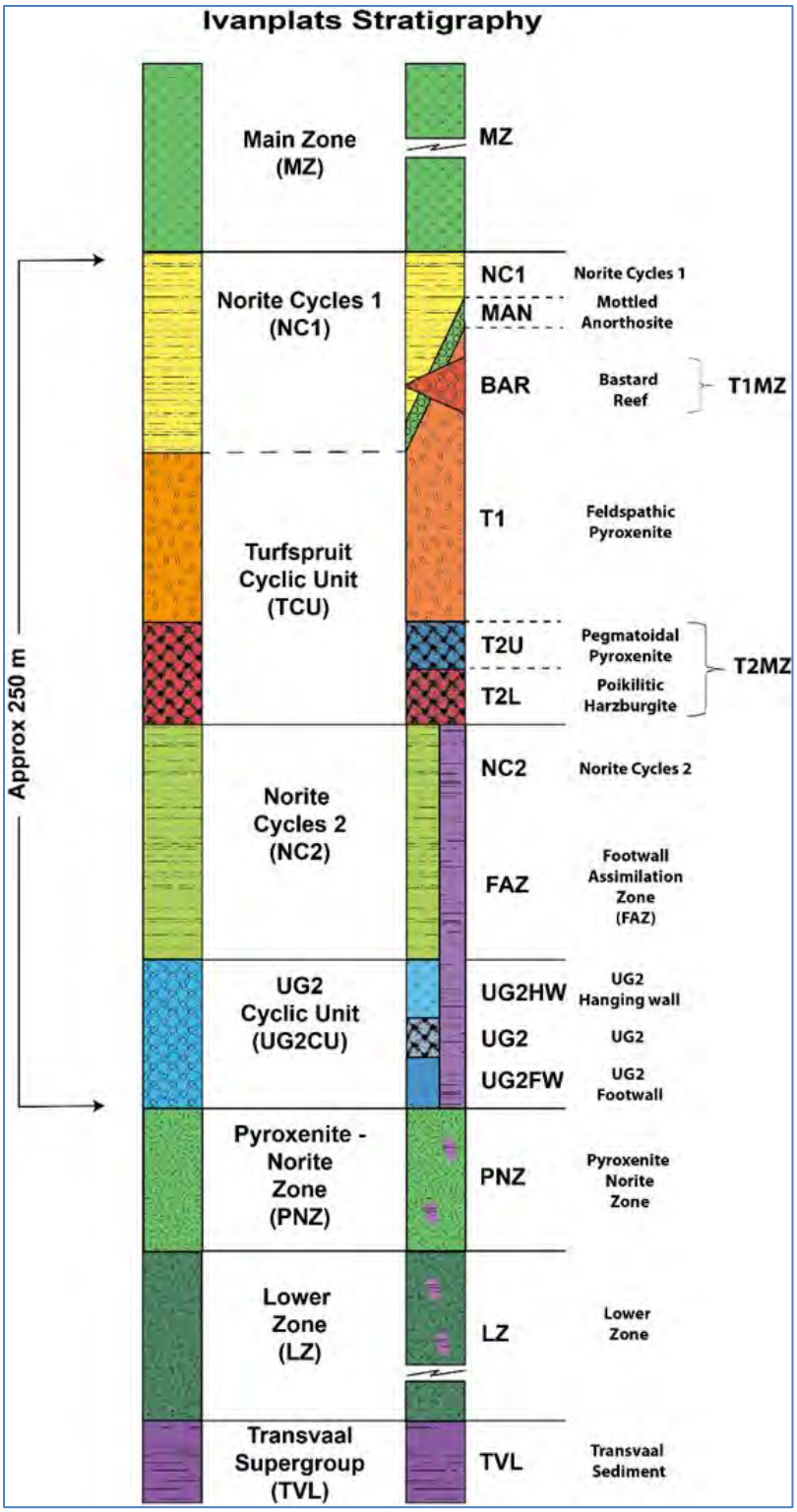
Figure 7.5 shows a comparison between a clean magmatic stratigraphy (left column) and a magmatic stratigraphy including significant sediment interaction (right column).

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).

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).

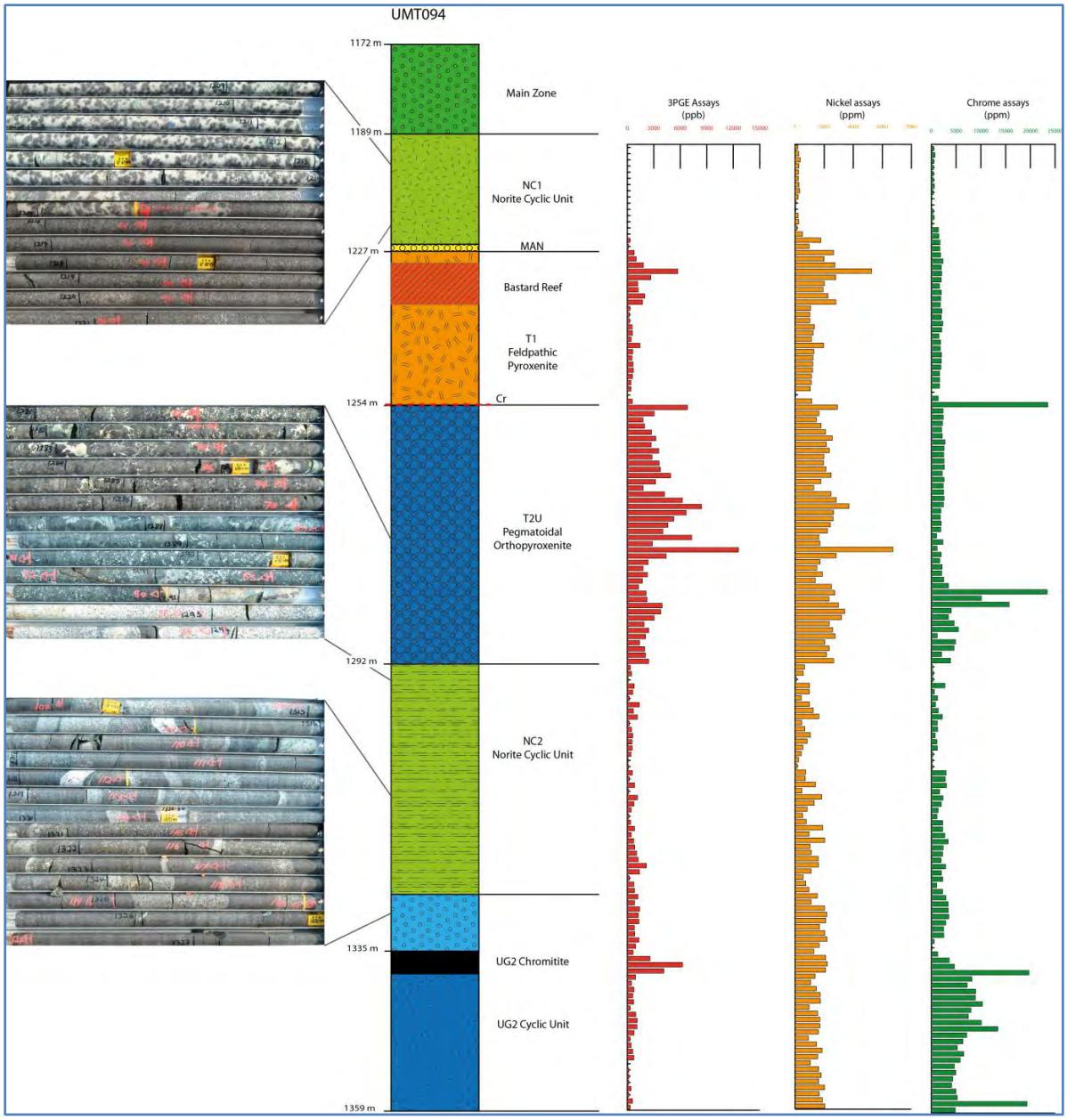
In addition to the above described magmatic stratigraphy, undifferentiated lithologies have also been recognised on the project. The major occurrence is the Footwall Assimilated Zone (FAZ) that occupies a similar stratigraphic position as the NC2 in the well drilled Zone 1 area. The FAZ is a zone of 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 Platreef Stratigraphic Column



Courtesy Ivanhoe, 2016.

Figure 7.6 Example of Magmatic Cyclic Units from UMT094



Courtesy Ivanhoe, 2016. See Figure 7.4 for location of UMT094; tickmarks are at 3000 ppb intervals for 3PGE, 2000 ppm for nickel, 5,000 ppm for chromium

7.2.3 Transvaal Supergroup

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

7.2.4 Lower Zone (LZ)

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

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

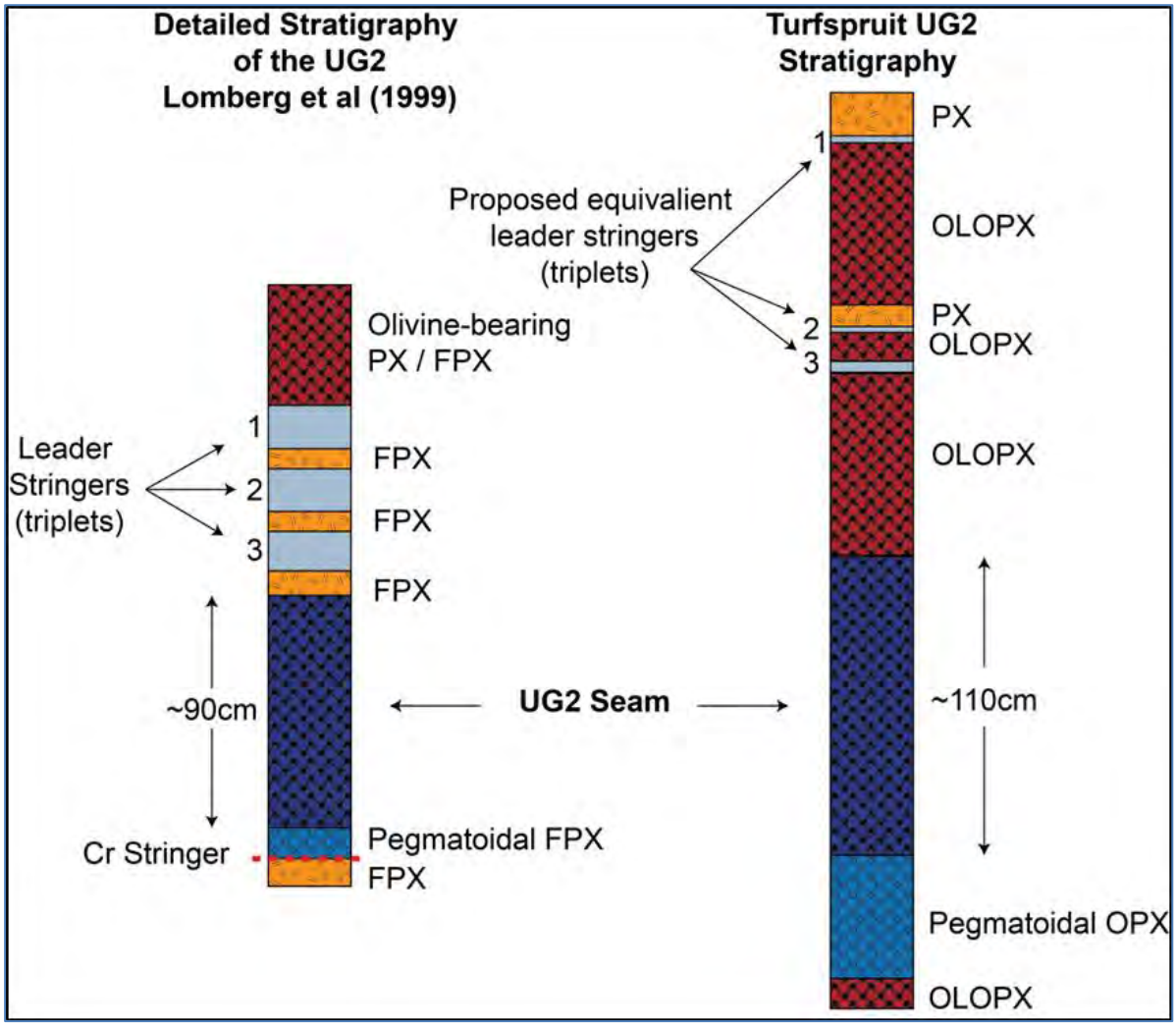
7.2.5 Pyroxenite-Norite-Zone (PNZ)

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

7.2.6 UG2 Cyclic Unit

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

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

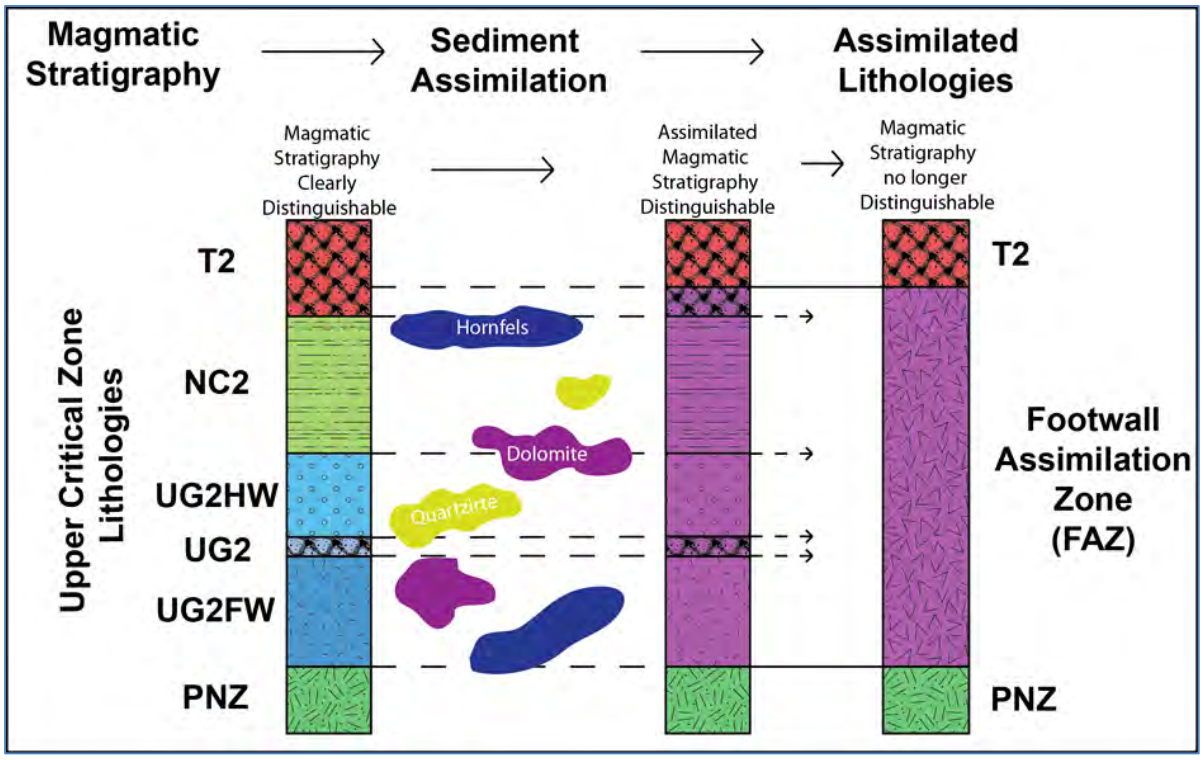


Courtesy, Ivanhoe 2016.

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)



Courtesy Ivanhoe modified after Nodder, 2015

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

7.2.8 Turfspruit Cyclic Unit (TCU)

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

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

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

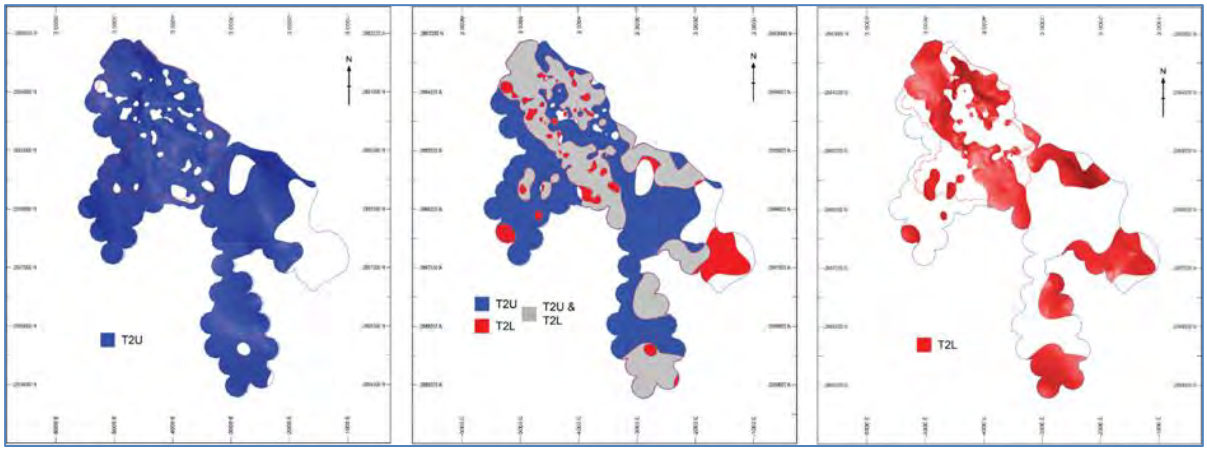
7.2.8.1 T2U and T2L

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

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

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

Figure 7.9 Distribution of the T2U and T2L



Courtesy 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 sulphides hosted within the typically equigranular feldspathic pyroxenite with local chromitite stringers. The T1MZ contact is gradational with adjacent weakly to un-mineralised T1 pyroxenite. The T1MZ is better developed where the T1 feldspathic pyroxenite is thickened. The average thickness of the T1MZ is 4.5 m using a 2 g/t 3PE+Au cut-off. Table 7.1 summarises the thicknesses of the T1MZ and T2MZ.

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

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

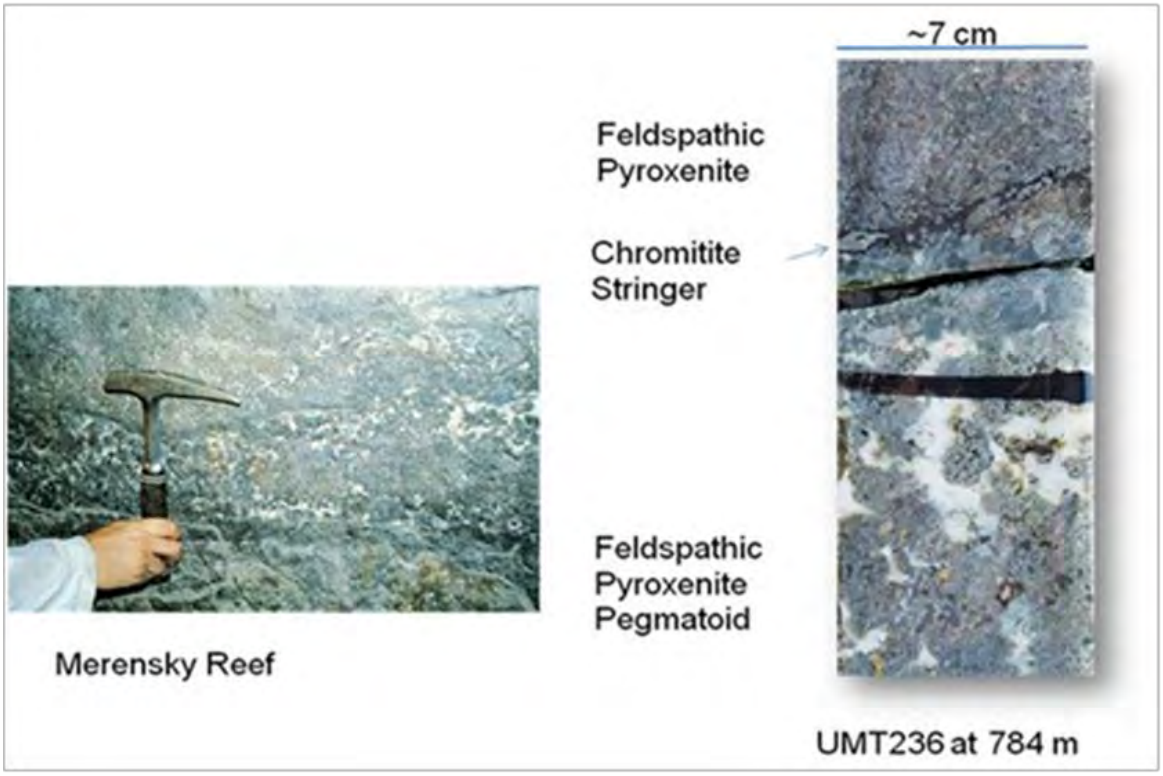
The T1MZ is therefore correlated with the Bastard Reef, and the mineralisation found above the top contact of the T2 pegmatoid, within the feldspathic pyroxenite, is correlated with the M1, as postulated by Davey (1992) and Lea (1996). This implies that the T2 pegmatoid correlates with the M2.

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

Table 7.1 True Thickness of Minzones

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

Figure 7.10 Comparison of Merensky Reef and the TCU



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

7.2.9 Norite Cyclic Unit 1 (NC1)

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

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

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

7.2.10 Main Zone (MZ)

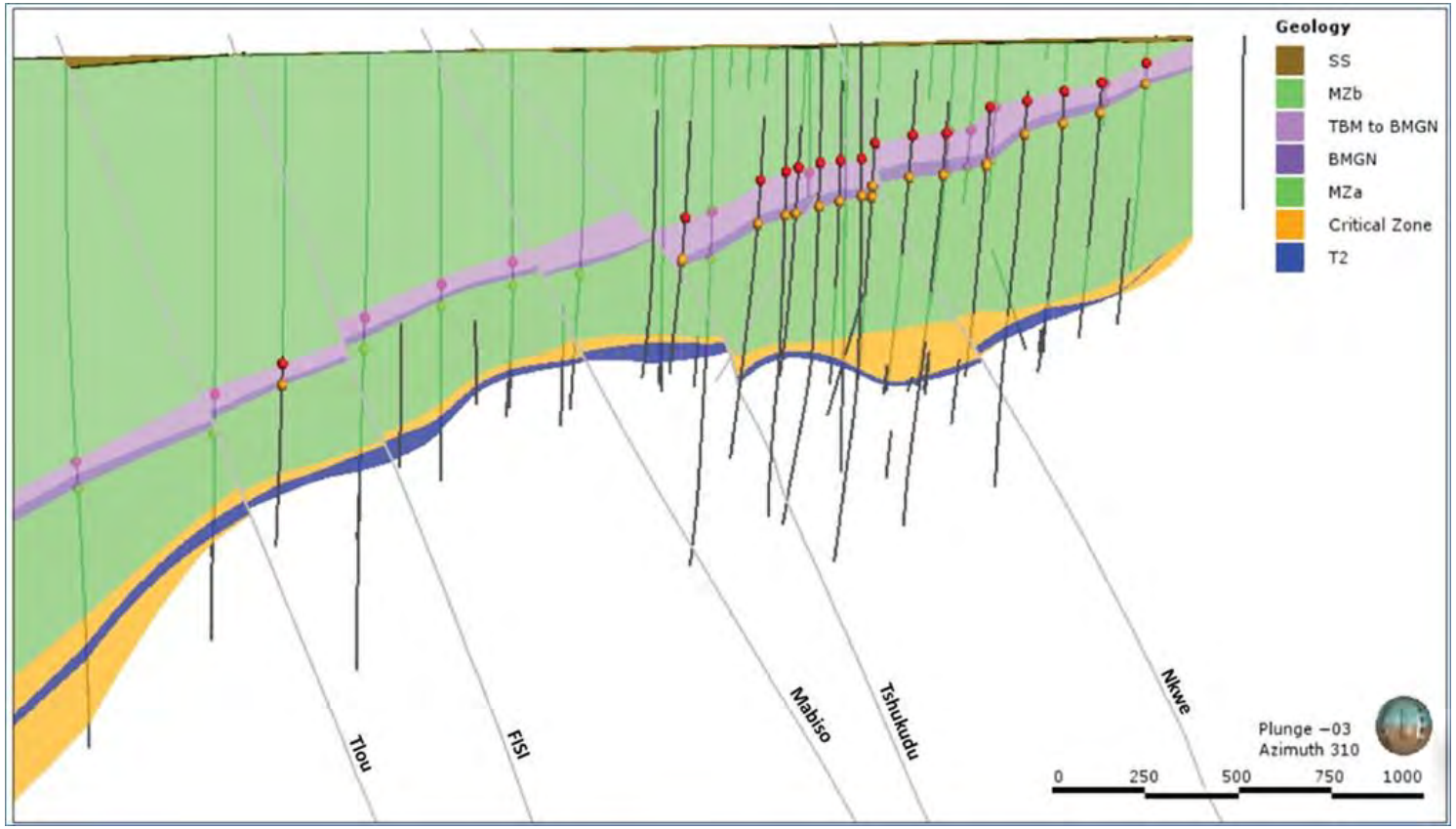
Overlying the magmatic rocks of the CZ is a succession of leucocratic to melanocratic norite and gabbronorite of the MZ. The MZ is the uppermost unit of the RLS observed in drillholes in the Platreef Project area, and MZ forms the hanging wall to the UCZ. Drilling has intersected the MZ up to a vertical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

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

- The interval (MZa) between the bottom of the base of the Main Zone and the base of the 'Basal Melagabbronorite' (BMGN) (bottom green layer in figure):
- The interval between the bottom and top of the BMGN (dark purple in figure);
- The interval between the top of the BMGM 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 7.11 Main Zone Layers (Section 7.0)



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

7.3 Metamorphism and Metasomatism

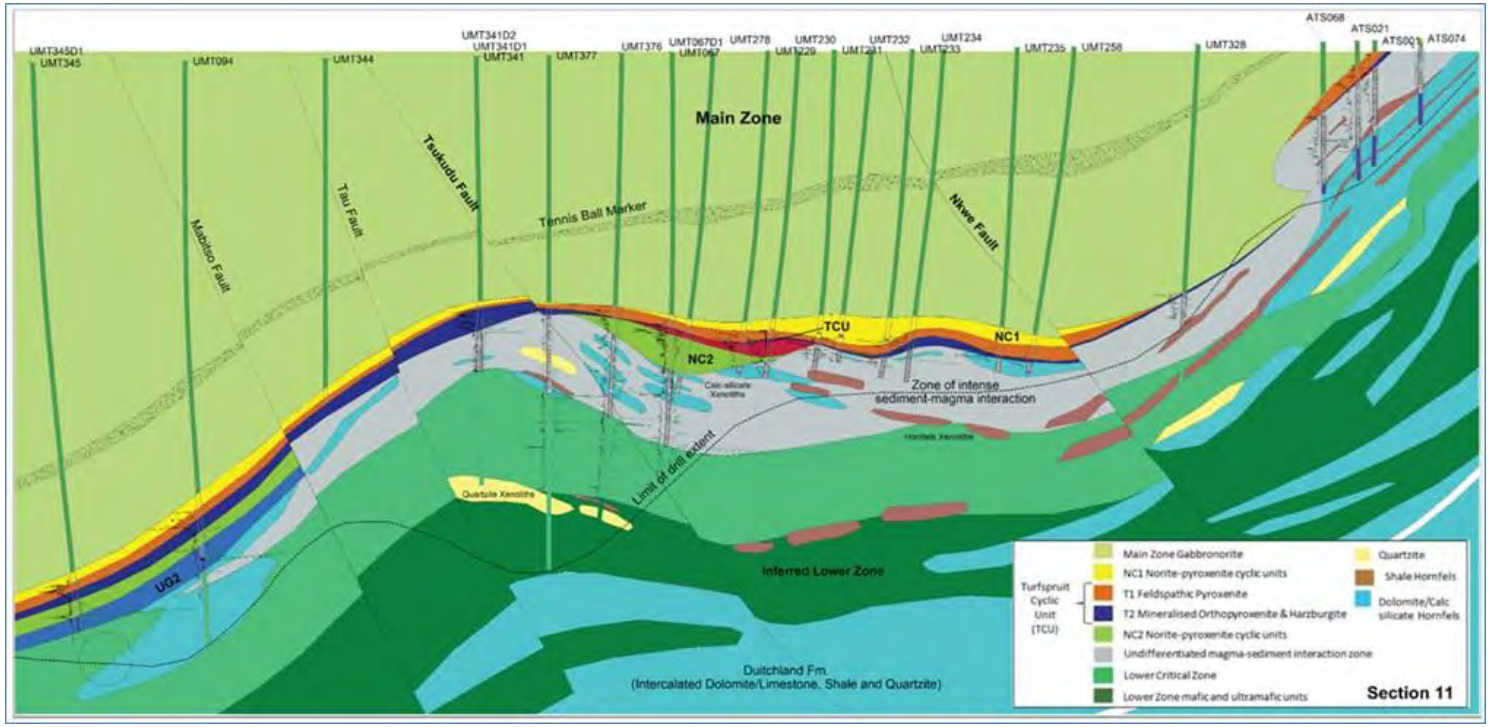
The stratigraphically lower and the up-dip magmatic units on the Turfspruit and Macalacaskop farms are characterised by interactions between Bushveld magma and the Transvaal Supergroup host sequence that is composed of diverse dolomite to alumino-silicate sedimentary rocks of the Duitschland Formation.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

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

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

Underlying the variably differentiated CZ units are layers of LZ ultramafic cumulates, that can be a thick as 800 m in some areas (Yudovskaya et al., 2013). The top of the LZ package appears transitional into plagioclase-rich lithologies. Rafts and xenoliths of pyroxene-cordierite hornfels are common and form part of a sequence containing various metasediments metamorphosed to granulite/pyroxene-hornfels facies. These very complex rocks have been variably brecciated and consist of a variety of Mg-skarn minerals. Other alteration products such as talc and serpentine can add local complexity (Figure 7.12). Table 7.2 summarises the mineralised intercepts for the drillholes in Figure 7.12.

Figure 7.12 Diagrammatic Dip Section



Courtesy Ivanhoe, 2016

Table 7.2 Intercepts Grading > 2 g/t and > 3 g/t 2PE+Au Located on Dip Section 11 (Figure 7.12)

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

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

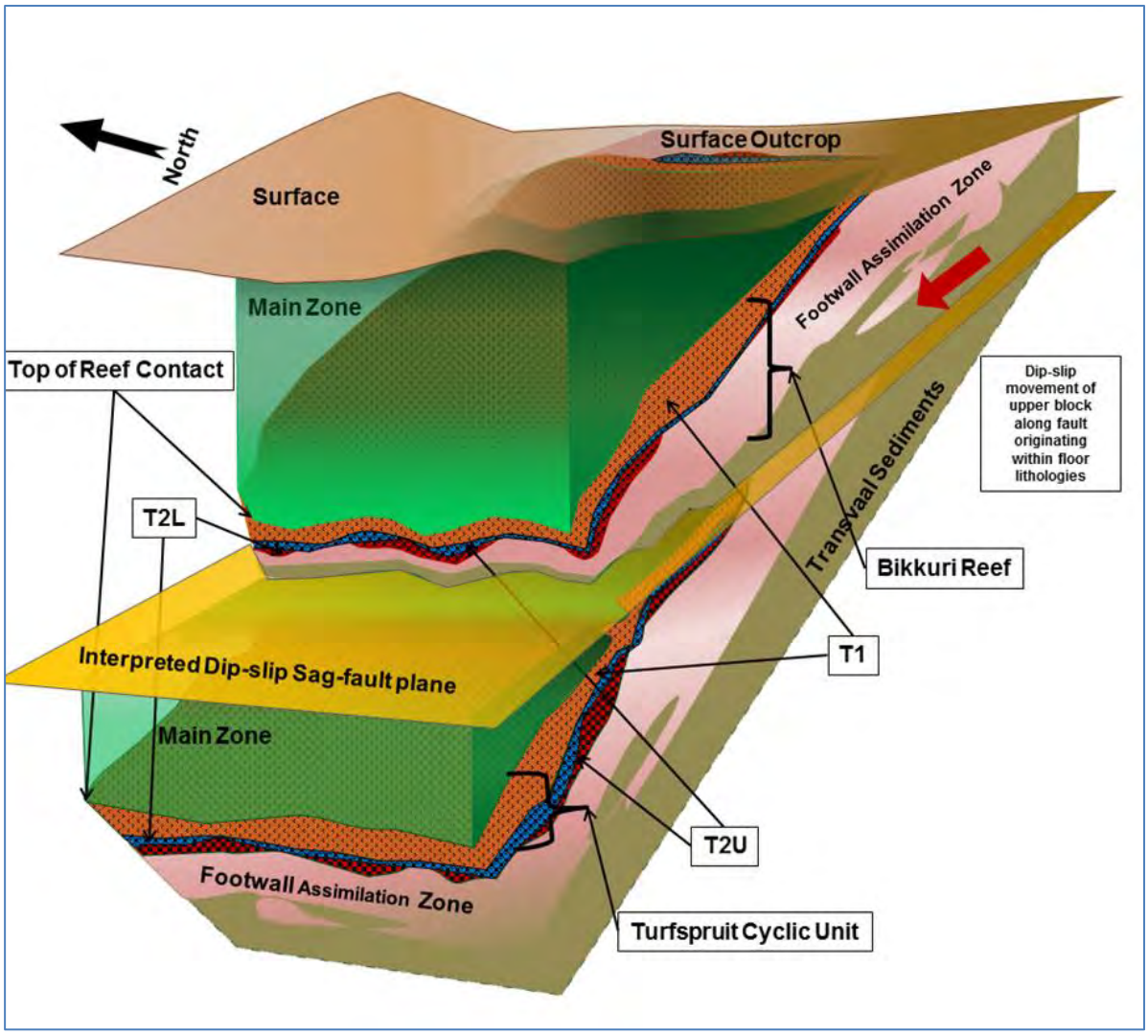
7.4 Bikkuri

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

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

The Bikkuri is interpreted to be the result of semi-consolidating magma that slumped back into the crystallising magma chamber (Grobler et al, 2013). Figure 7.13 shows a diagrammatic view for the Bikkuri emplacement.

Figure 7.13 Schematic Diagram of Preferred Emplacement Mechanism for Bikurri.



Courtesy Ivanhoe, 2016, After Grobler et al (2013)

7.5 Structure

7.5.1 Regional Structures

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

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

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

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

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

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

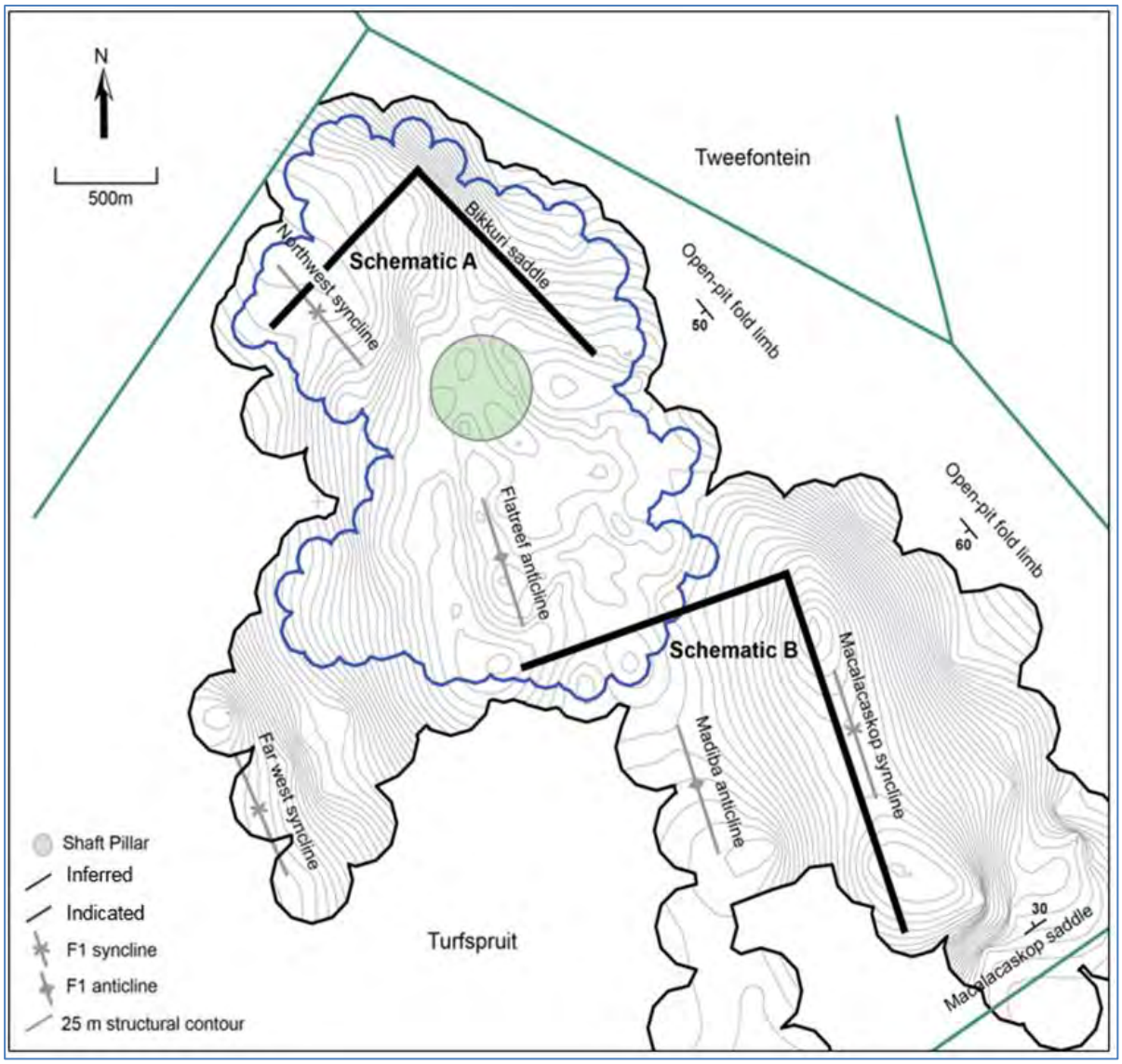
7.5.2 Project Folding

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

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

Figure 7.14 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.

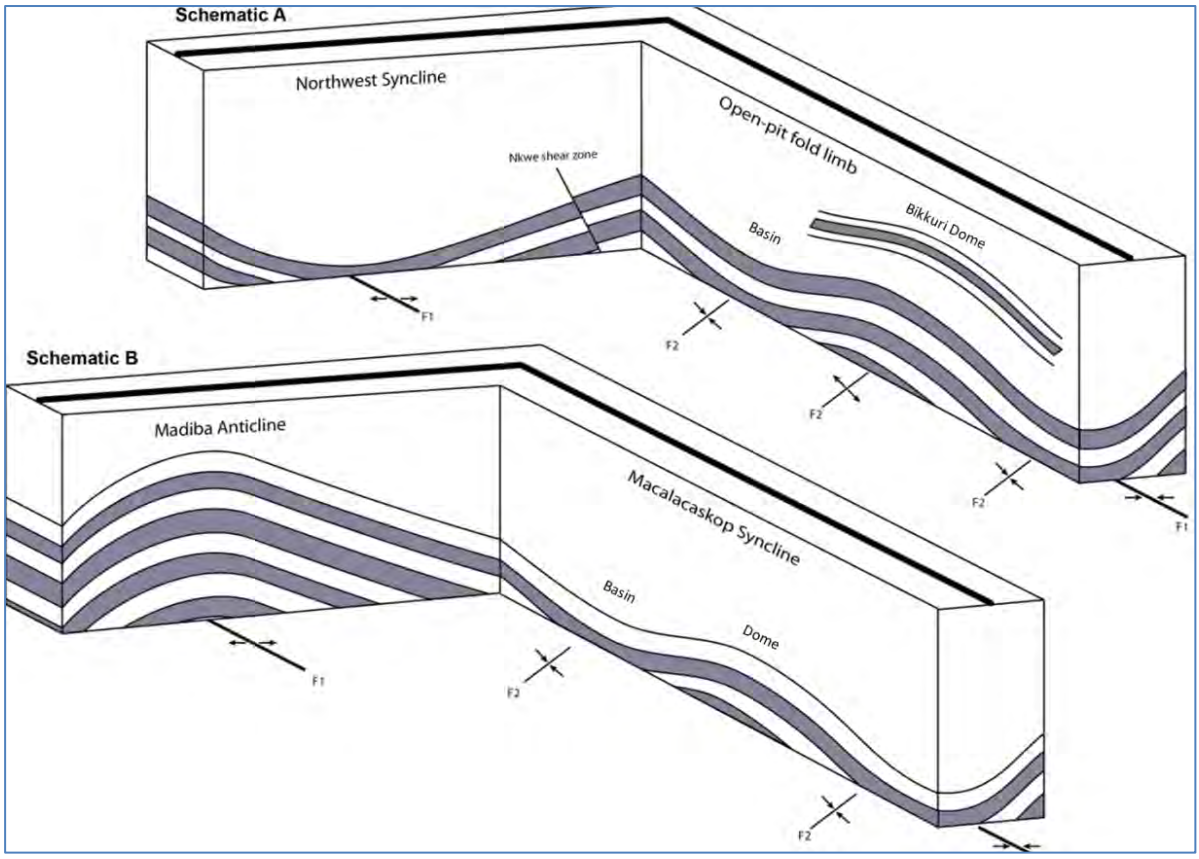
Figure 7.14 Plan View of Major F1 Fold Structures on Structural Contour Map



Courtesy Ivanhoe, 2016. Structural contours define the base of the T2. Section lines refer to the sections shown in Figure 7.13.

Figure 7.15 is a schematic of the interpreted folding derived from metasedimentary interlayers or "rafts".

Figure 7.15 Schematic View of Fold Structures in Zones 1 and 2



Note: Figure courtesy Ivanhoe, 2016. Oblique view from south to north of schematic folding derived from metasedimentary interlayer "rafts". Figure 7.14 shows the location of the fence lines.

7.5.3 Project Fault Structures

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

An updated 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 revised 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 refined 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.

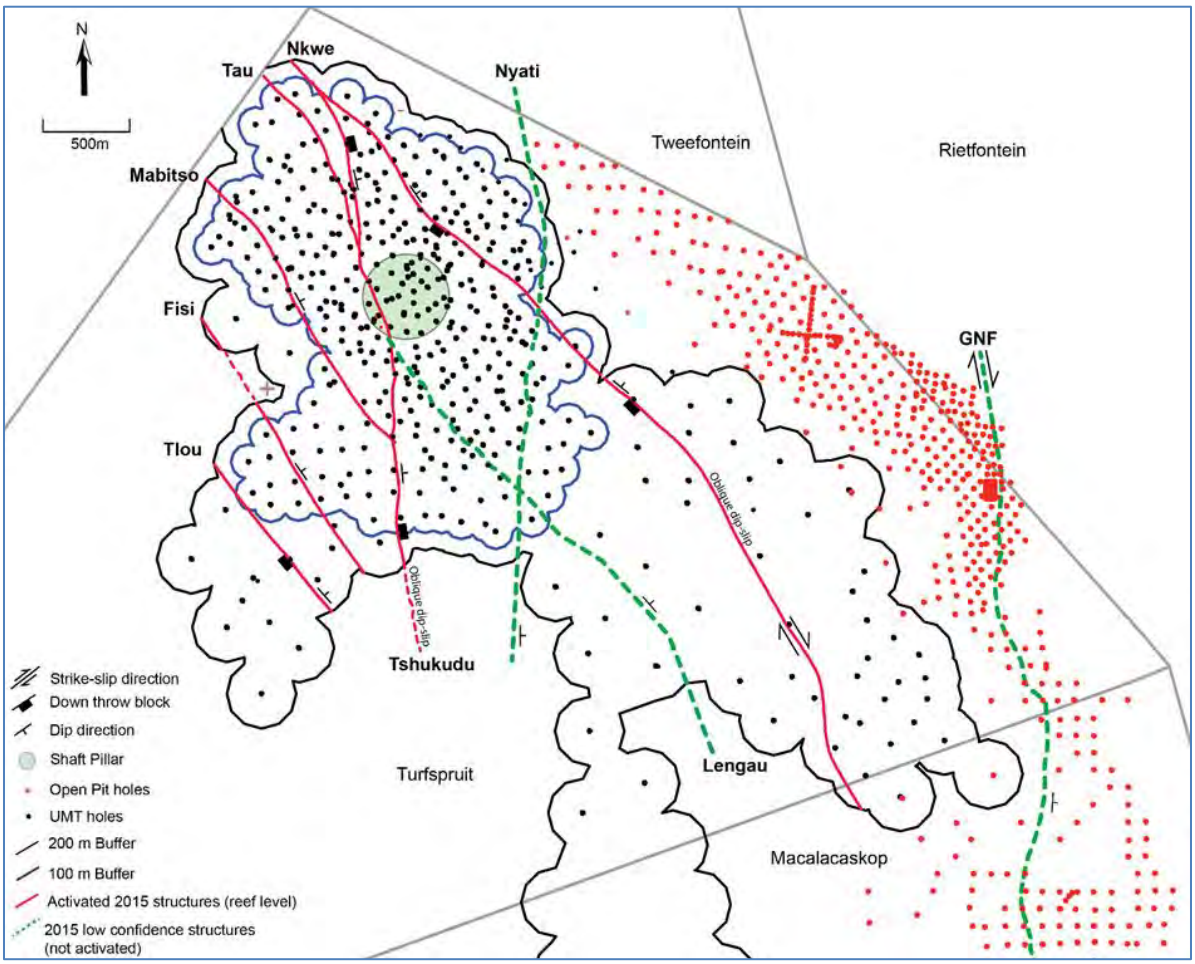
The major ductile fault structures currently recognised include:

- 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 1900 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 7.16 shows the faults included in the 2016 Mineral Resource model update. The dip direction and sense of movement are also shown.

Figure 7.16 Structure Interpretation for Model Domains.



Courtesy Ivanhoe, 2016.

Three primary structural trends are evident for the steep structures:

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

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

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

7.5.4 Granite Dykes

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

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

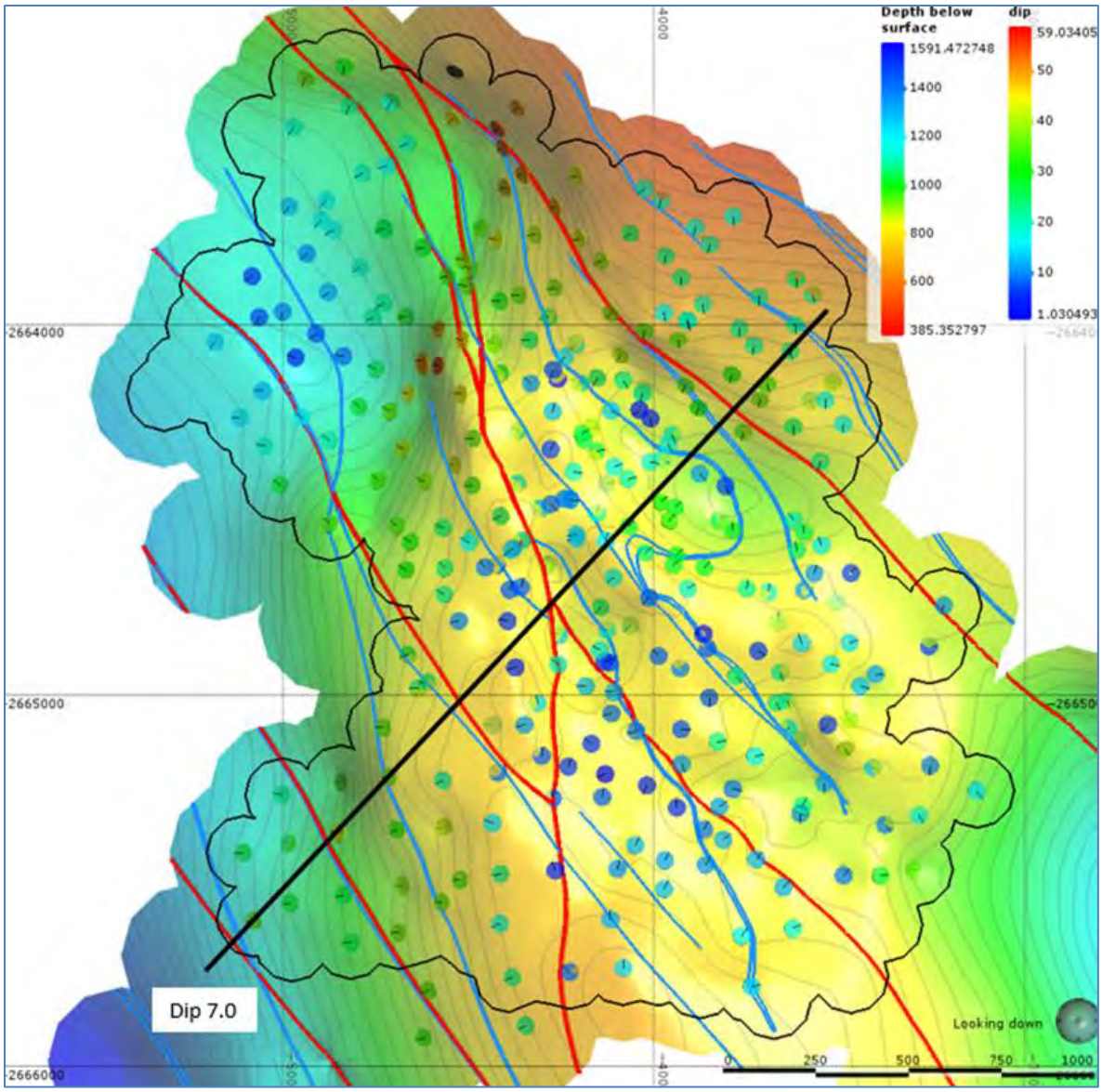
Figure 7.17 shows the locations of key dyke features in relation to the structural model at the level of the T2 horizon.

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

- Low Angle Granite Veins (LGVs): strike 335° and dip at 32° towards the northeast; a total of eight sub-parallel dykes have been modelled as continuous features named LGV10, LGV20, LGV30, LGV40, LGV50, LGV60 and LGV70;
- Steep Granite Veins (SGVs): strike 329° and dip at 68° towards the northeast; a total of 10 sub-parallel zones with increased dyke frequency of occurrence and widths have been modelled as continuous features and named SGV10 to SGV100. Numerous additional granite intersections in drill core (< 2 m thick) indicate that a significant number of narrow stockwork-type intrusions should be anticipated during underground development. This is particularly relevant between SGV10 and SGV20.

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

Figure 7.17 Structural Model Intersecting the Base of The T2 Surface



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

7.5.5 Other Considerations

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

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

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

7.5.6 Conclusions

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

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

7.6 Mineralisation

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

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

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

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

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

A PNZ domain includes predominantly disseminated sulphide mineralisation within homogeneous pyroxenite/norite lithologies. Locally, massive sulphides 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 sulphides hosted in feldspathic pyroxenite. BMS are predominantly chalcopyrite, pentlandite and pyrrhotite.
TCU	T1	Medium to coarse-grained magmatic sulphide grains hosted in feldspathic pyroxenite.
	T2 (Merensky reef analogue)	Very coarse-grained magmatic sulphides hosted in pegmatoidal orthopyroxenite and pegmatoidal poikilitic harzburgite. The top of the mineralised zone is commonly marked by a chromite stringer.
FAZ	UDCZ[1]	Medium- to coarse-grained magmatic sulphides 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 sulphides hosted in chromitite. Associated with high- grade PGEs
PNZ Mineralisation	Platreef contact style mineralisation	Fine-grained massive sulphide bodies hosted mainly in the pyroxenite and norite of the PNZ.

1; UDCZ=Undifferentiated Contaminated Zone

7.6.2 Platinum Group Minerals (PGM) and Base Metal Sulphides

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

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

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

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

7.6.2.1 Base Metal Sulphides

Within the Platreef Project, sulphide occurrence consists mainly of pyrrhotite, pentlandite, chalcopyrite and lesser pyrite. Sulphide distribution and concentrations vary and range from less than 1% to more than 25%. Rare sections of core may have massive sulphides 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 sulphide formation. An early phase is dominated by irregular blebs of disseminated pyrrhotite and pentlandite followed by a later phase where chalcopyrite is dominant. Figure 7.18 and Figure 7.19 shows sulphide 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-sulphides. The abundance of PGE- arsenides, antimonides, Bi-Te minerals and PGE-sulphide minerals that are associated with the upper part of the T2U geomet unit corresponds to the upper part of the Merensky Reef found elsewhere. The geometallurgical study has also shown that only a small fraction of PGE mineral assemblage are associated with BMS.

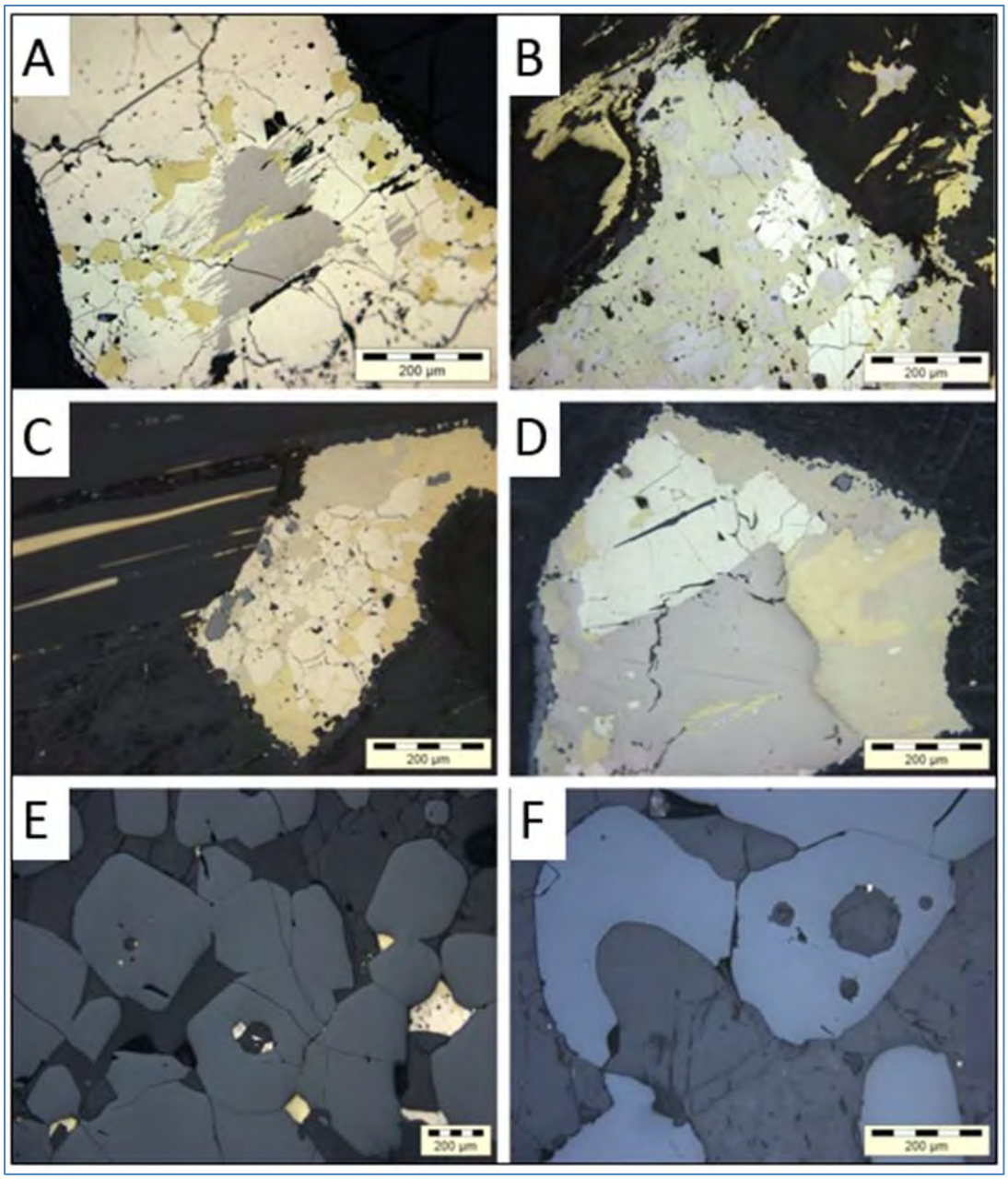
Yudovskaya (2015) determined that clear magmatic assemblages (Merensky-like trends) can be distinguished from an original assemblage influenced and overprinted by secondary effects. The zonation of PGM distribution favours in-situ crystallisation where modal PGE mineral assemblages are controlled by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

The Bastard and T2U reefs contain an association of high-temperature primary magmatic Pt sulphides and Pt alloys that often form eutectoid intergrowths with base metal sulphides. This is an indication of crystallisation at around 1,000ºC. Chromitite is the only lithology which contains laurite (RuS2). Figure 7.20 shows detailed SEM images with PGE mineral assemblage and textural relationships typical of the T2 reef.

Figure 7.18 Core Photograph from UMT083 at 1,323 M Depth, Illustrating Sulphide Mineralisation

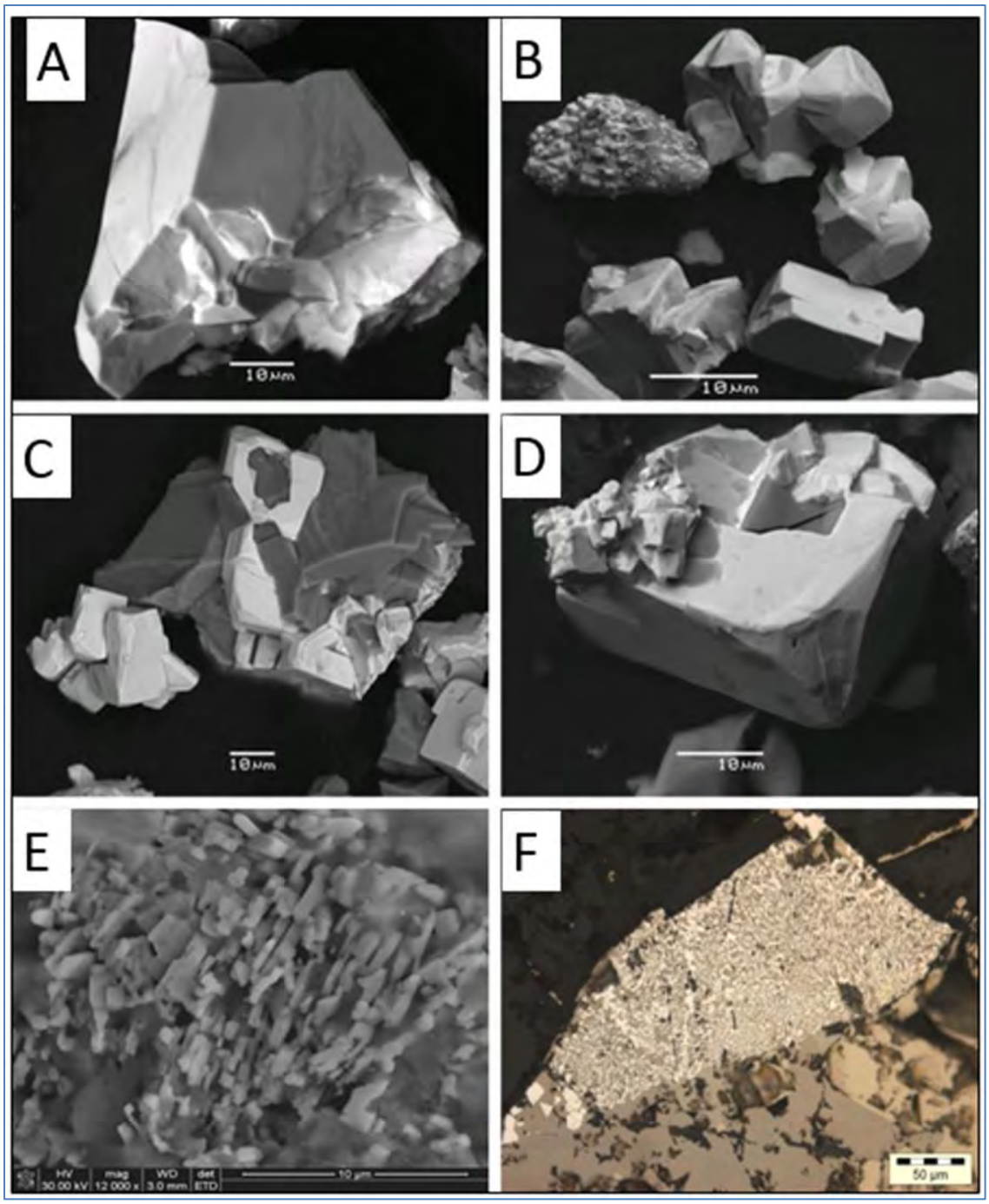
The yellowish mineral is chalcopyrite; the dull purplish mineral is pyrrhotite; the light cream mineral with higher reflectance and some cleavage is pentlandite. Core photographs courtesy Ivanhoe, 2011.

Figure 7.19 Mineral Assemblages Found in the T2U.



A-Flame-like pentlandite after pyrrhotite (UMT314 at 1135.5m). B – Relics of pyrrhotite in cubanite. Granular pentlandite is white, chalcopyrite is remobilised outside the massive sulphide intergrowth (UMT314 at 1136.7m); C – Pyrrhotite-cubanite-pentlandite assemblages is replaced by secondary silicates along margins (UMT314 at1136.7m); D – magmatic pyrrhotite-cubanite-pentlandite assemblages is rimmed by later magnetite rim (UMT314 at1136.7m); E – atoll-like and sieved chromite in the chromitite seam. Sulphides are seen as interstitial and inclusions in chromite (UMT314 at 1135.54m); F – embayed and atoll-like chromite of the lowermost chromitite seam (UMT314-1160) (courtesy of Yudovskaya (2015) unpublished internal correspondence). See Figure 7.4 for location of UMT314.

Figure 7.20 SEM Images of Typical Platinum Minerals from The T2 Reef



A: RuS2 intergrown with IrAsS (UMT314 at 1135.54); B: PtS and Pt3Fe crystals as well as fine-grained eutectoid intergrowth of Pt-Fe alloy and pyrrhotite (UMT314 at 1135.54); C: euhedral skeletal crystals of isoferroplatinum intergrown with pyrrhotite (UMT314 at1135.71); D: wide range of isoferroplatinum crystal sizes (UMT314 at1135.71); E: micron-sized crystals of isoferroplatinum in sulphides (UMT314 at1135.71); F: the same type as in E eutectoid intergrowth of Pt3Fe and pyrrhotite adjacent to coarser pyrrhotite and pentlandite under reflected light (UMT314 at1135.54). (Courtesy of Yudovskaya (2015) unpublished internal correspondence). See Figure 7.4 for location of UMT314.

7.7 Comments on Section 7

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

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

The data have permitted an update of the structural model and a better understanding of the magmatic stratigraphy on the Platreef Project.

8 DEPOSIT TYPES

Two main PGE deposit types occur within the Bushveld Complex:

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

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

- The Platreef remains a complex and enigmatic deposit.
- Stratigraphic relationships with other stratiform deposits such as the Merensky and UG2 reefs have been suggested.
- The extent to which the Northern Limb was connected to the rest of the complex across the Thabazimbi–Murchison Lineament (refer to Figure 7.1 where this is shown as the TML fault) remains to be established.
- The Platreef represents a complex of sills intruded into basement granite-gneiss, Transvaal Supergroup sediments or pre-Platreef Lower Zone intrusions.
- Intrusive relationships of the Main Zone gabbronorites, into solidified and deformed Platreef, removes the Main Zone as a source of metals for the Platreef.
- Mineral chemistry, bulk geochemistry, and Sr, Nd, and Os isotope geochemistry of the Platreef are most consistent with an ultramafic (Critical or Lower zone) component.
- Platreef εNd 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 sulphides 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 sulphides 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 sulphides were subsequently expelled as a crystal-sulphide 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.

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

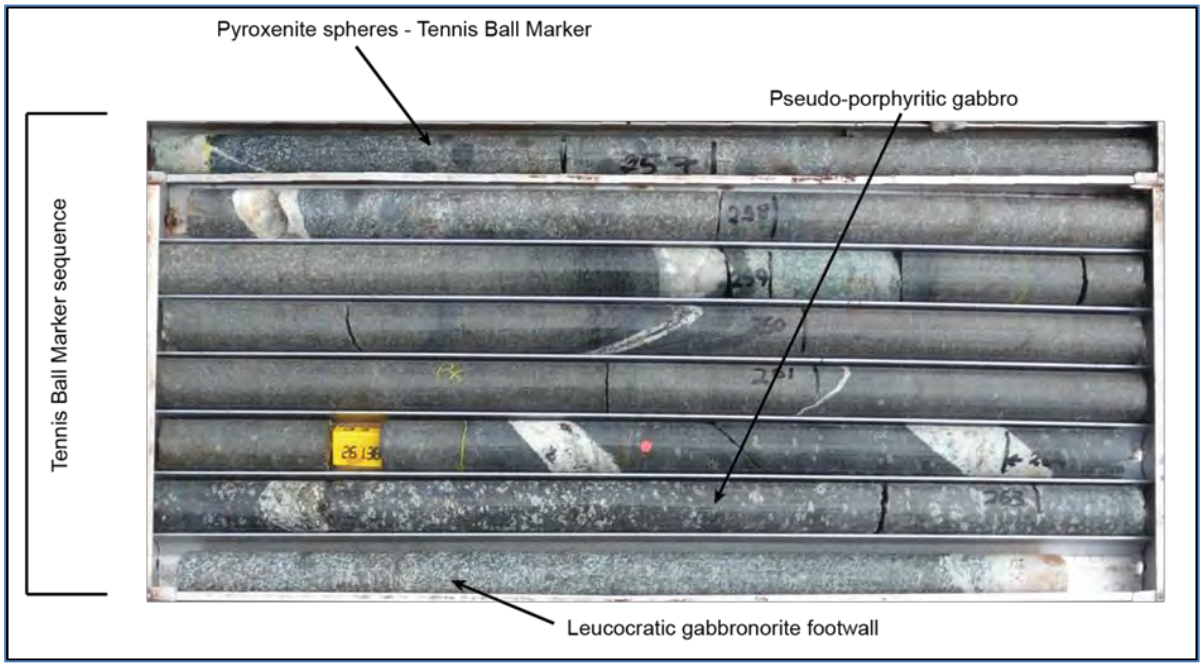
Systematic modelling of MZ and UZ lithologies or a model of the granite dykes had never been undertaken. Recent work has enabled Platreef geologists to confirm trends on the magnetics image.

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

Figure 9.1 Core Showing TBM

Courtesy Ivanhoe, 2016.

Figure 9.2 TBM Sequence in UMT070



Courtesy Ivanhoe, 2016.

9.3 Geochemistry

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

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

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

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

9.4 Geophysics

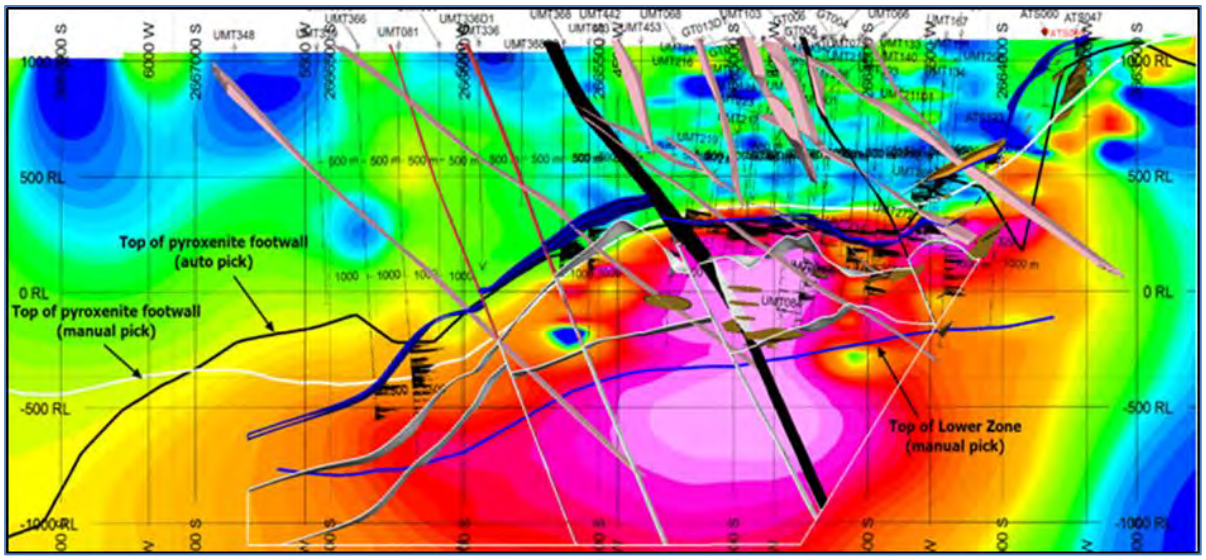
Geophysical survey methods at the Platreef Project have included aeromagnetics, gravity gradiometer and a number of downhole geophysical methods including caliper; self-potential (SP)/point resistance (PR); electrode-array-focussed resistivity (EAL); magnetic susceptibility (Msus); temperature/conductivity;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

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

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

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

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



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

Figure 9.4 Geologically Constrained Falcon Gravity Inversion Interpretation

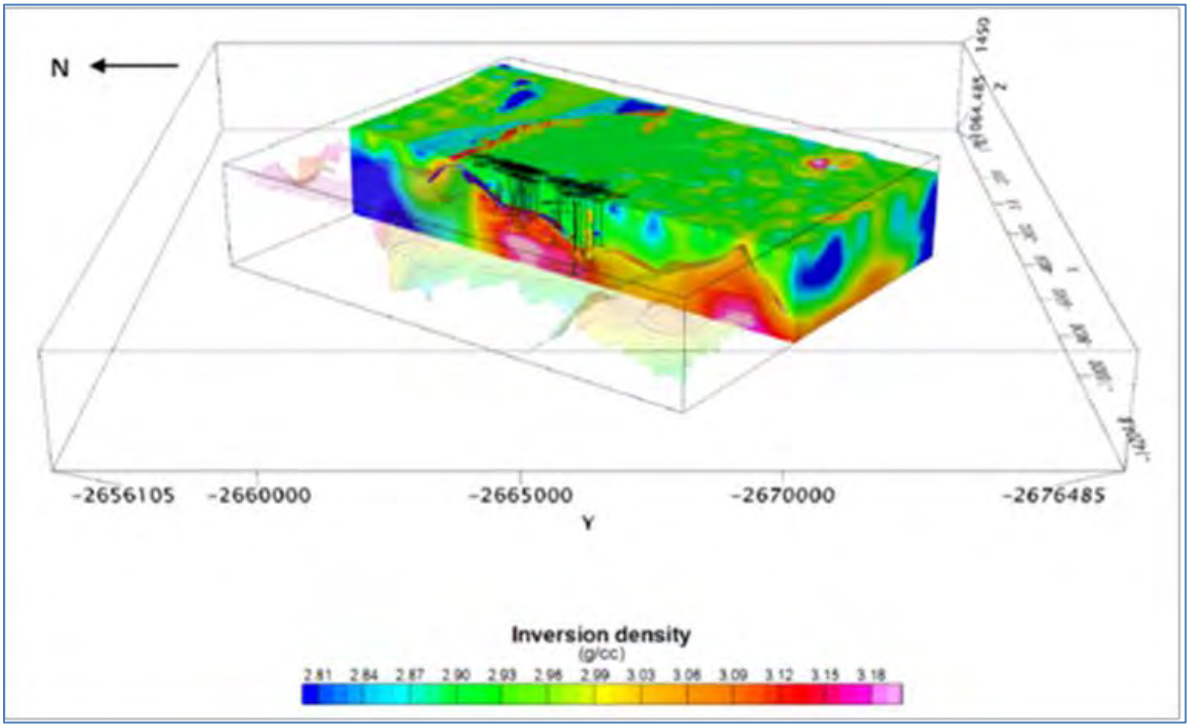
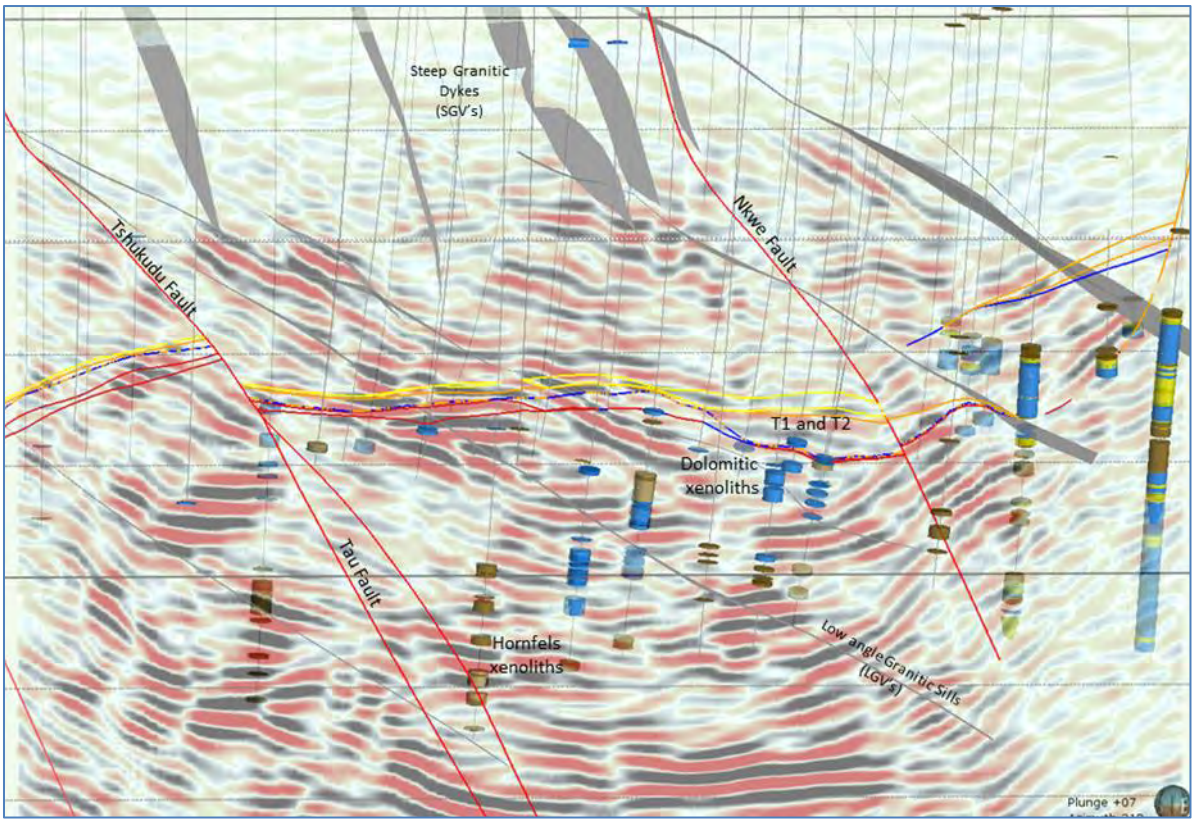


Figure courtesy Ivanhoe, and sourced from Williams (2012). Inversion sliced along a north-east oriented section. Image shows computed depth to >2.97 g/cm3 isosurface which maps the gabbronorite/pyroxenite contact and thereby depicts the approximate structure of the mineralised reef.

Figure 9.5 Dip Section 10.0 from Vlseis Depth Converted Data



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

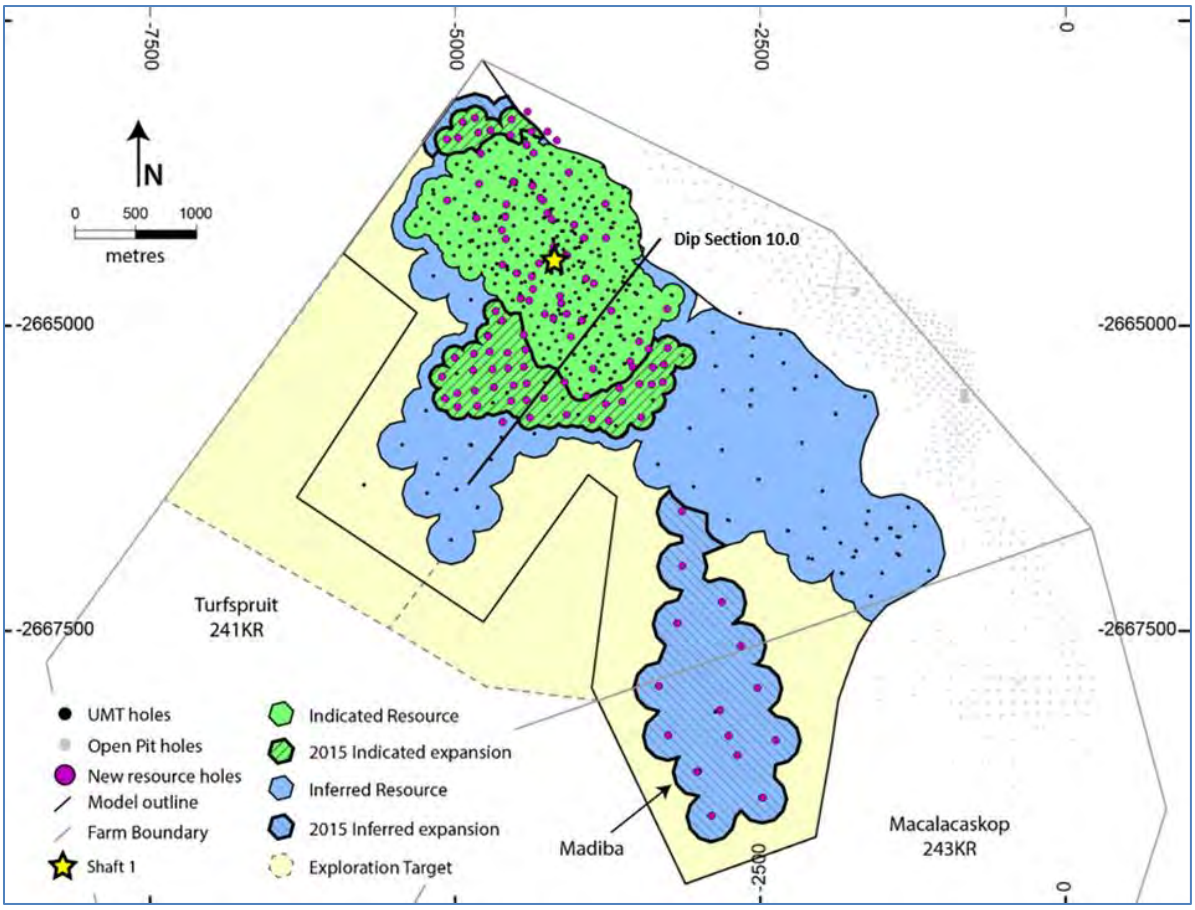
9.5 Petrology, Mineralogy, and Research Studies

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

9.6 Exploration Potential

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

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



Courtesy Ivanhoe, 2016.

9.7 Comments on Section 9

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

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

10 DRILLING

10.1 Drill Summary

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

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

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

A total of 3 (3,094 m) drillholes from Phase 1 were included in the resource model update for estimation purposes.

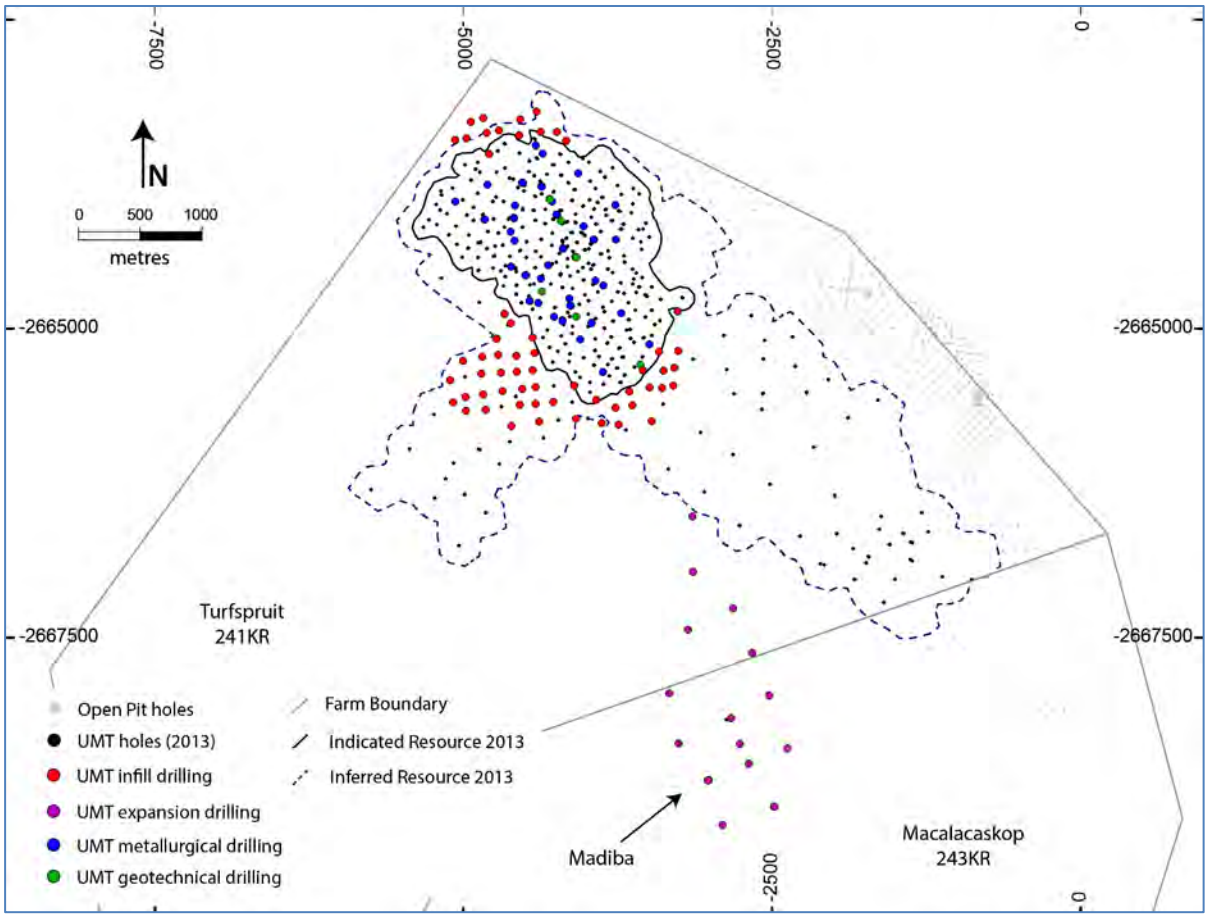
The database includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totalling 501,638 m completed by 11 February 2015. No drilling for resource estimation purposes has occurred between this date and the Report effective date. Depths for deflections are 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 drilling was completed 11 February 2015.

Figure 10.1 Drill Collar Location Plan with New Drilling Since 2013



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

10.1.1 Drilling Completed 2013 To 2015

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

Table 10.1 Drilling Completed Since 2013

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

10.1.2 Zone 4

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

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

10.1.3 Zones 1 To 3

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

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

10.1.4 Zone 5

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

10.2 Drill Methods

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

10.2.1 Zone 4

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

10.2.2 Zones 1 To 3 and Zone 5

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

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

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

10.3 Geological Logging

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

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

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

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

10.3.1 Zone 4

Geological core logging involved the recording of lithology; grain size; type and degree of alteration (low, medium, or high); type and visible percentage of sulphide (pentlandite, pyrrhotite, chalcopyrite, and pyrite); relative sulphide 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 sulphide 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 to 3 and Zone 5

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

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

10.4 Core Recovery

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

10.5 Collar Surveys

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

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

10.6 Downhole Surveys

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

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

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

10.7 Metallurgical Drilling

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

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

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

Figure 10.2 Metallurgical Drillhole Map

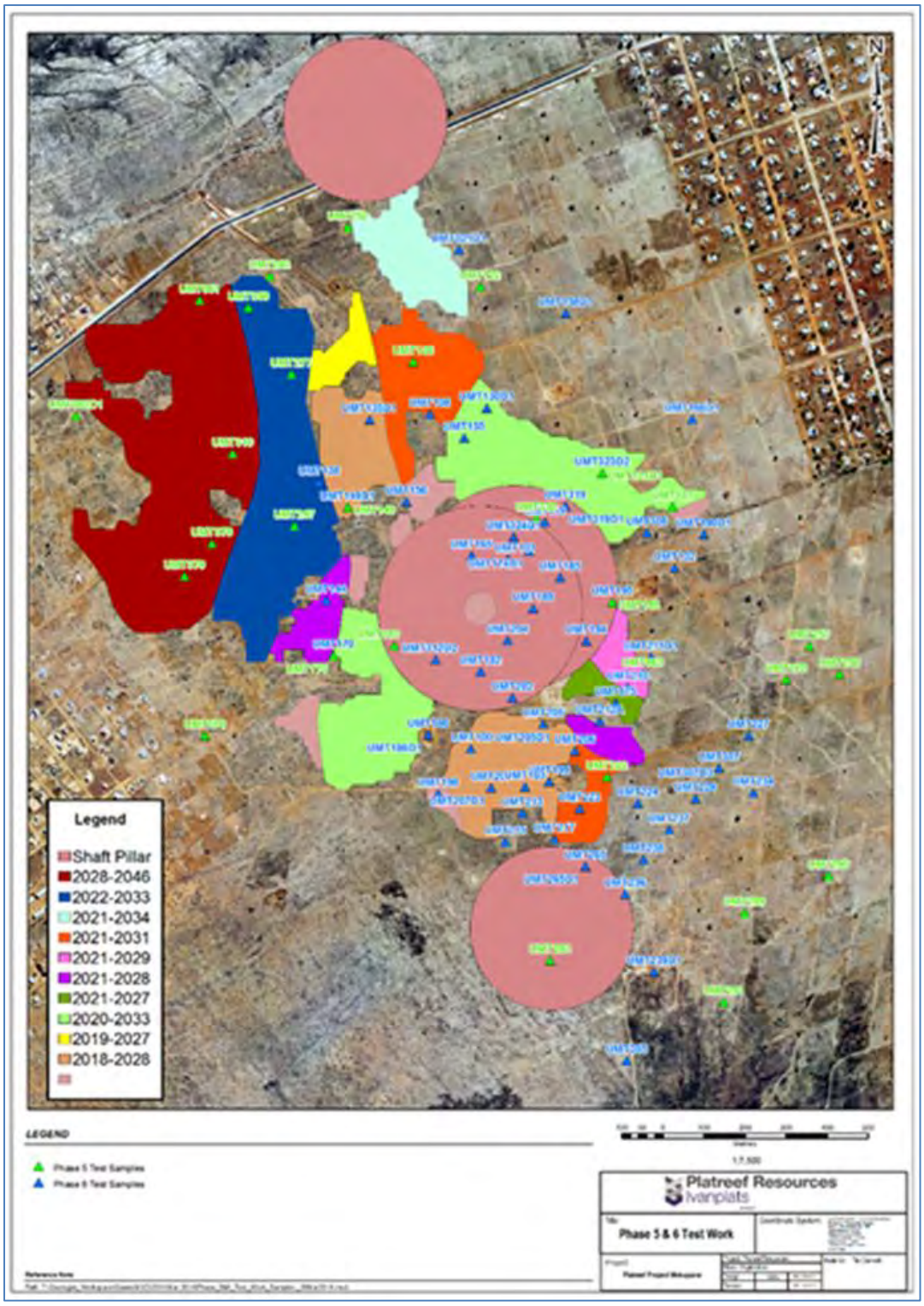


Figure by Ivanhoe, 2016

10.8 Summary of Drill Intercepts

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

Table 10.2 Drill Intercept Example Summary Table

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

10.9 Comparisons of Intercept Positions - Twin Hole Data

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

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

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

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

10.10 Comments on Section 10

10.10.1 Amec Foster Wheeler Comments

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

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

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

10.10.2 Metallurgical

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

10.10.3 Geotechnical

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

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

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

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

11.1 Sampling Methods

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

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

11.1.1 Assay Sampling

AMK and ATS Sampling

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

UMT Sampling

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

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

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

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

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

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

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

11.2 Density Determinations

11.2.1 Zone 4

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

11.2.2 UMT Bulk Density

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

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 Density by Stratigraphic Unit

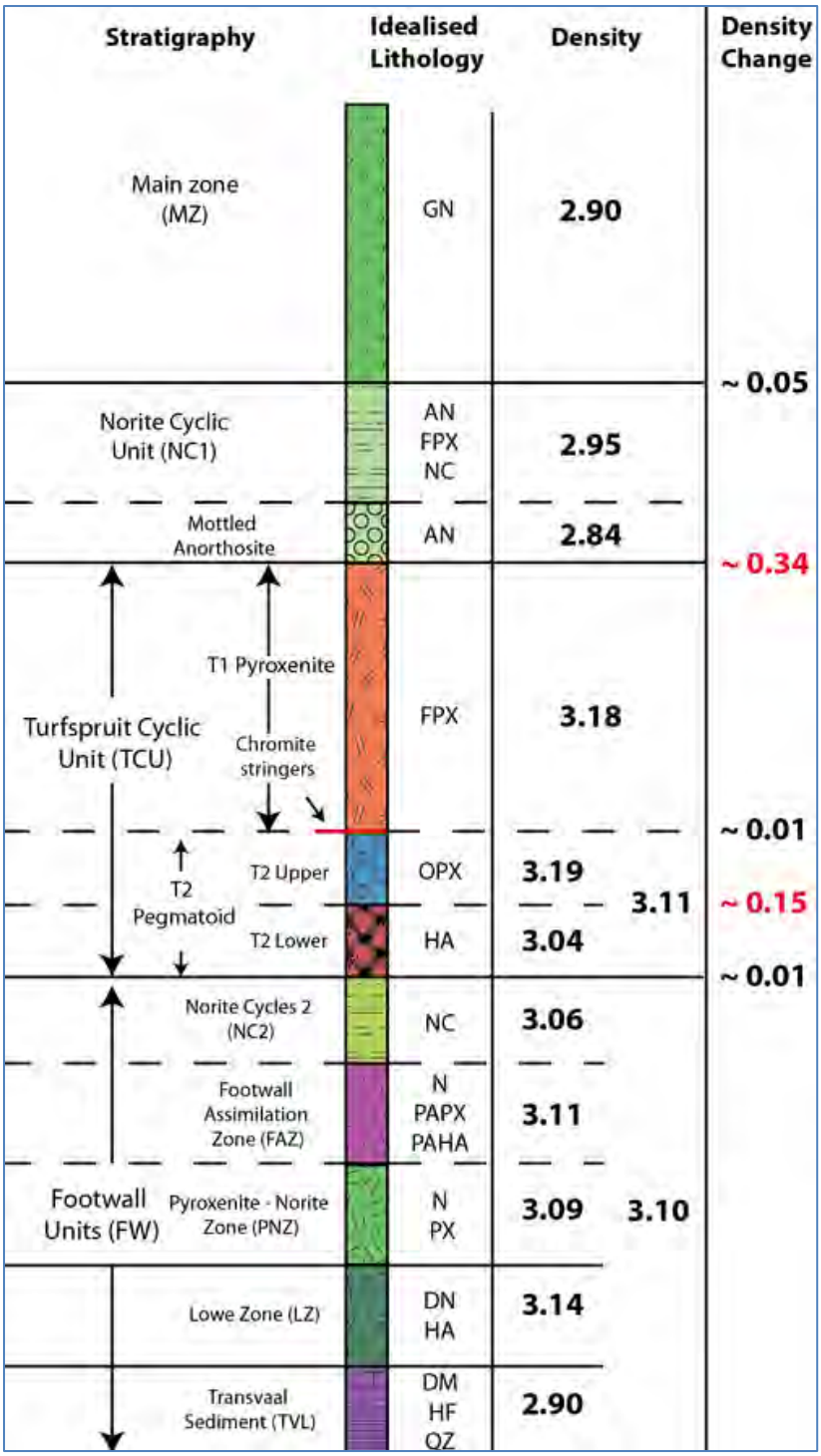
MSTRAT	Description	MSTRAT	MCODE	Number	Average	Minimum	Maximum	C.V.
Bikurri Hanging Wall	Main Zone	BKHW	10	3,772	2.91	2.55	3.30	0.03
	Bikkuri Norite Cycles 1	BKNC1	11	244	2.98	2.62	3.26	0.04
	Bikkuri Mottled Anorthosite	BKMAN	12	31	2.83	2.63	2.95	0.02
Bikurri	Bikkuri B1	B1	13	264	3.15	2.62	3.34	0.03
	Bikkuri B2	B2	14	59	3.13	2.84	3.37	0.04
Bikurri 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, Amec Foster Wheeler notes, with reference to Figure 11.1 that:

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

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

Figure 11.1 Idealised Density Strip Log



Courtesy Ivanhoe, 2016.

11.3 Analytical and Test Laboratories

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

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

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

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

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

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

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

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

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

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

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

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

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

11.4 Sample Preparation and Analysis

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

11.4.1 AMK and ATS Sample Preparation and Analysis

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

11.4.2 UMT Sample Preparation

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

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

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

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

11.4.3 UMT Sample Analysis

Ultra Trace is the main laboratory used to analyse samples and used a multi-acid digestion followed by inductively-coupled plasma-optical emission spectroscopy (ICP-OES) reading to determine total Ni, Cu, Cr, and sulphur. Some samples were also assayed for sulphur using a LECO furnace (controlled combustion of sample pulp with infrared reading of SO2 gas); the LECO and ICP sulphur 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 20th sample would also be assayed by fire assay with nickel sulphide collection followed by ICP-MS (FN001).

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

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

- Fire assay Pd collector followed by ICP-MS for Au, Pt and Rh (Code 415).
- NiS collection for Au, Pt, Pd, Rh, Ir, Ru and Os (Code 419).

Ultra Trace (now Bureau Veritas Minerals) used the following analytical methods:

- Fire assay lead collection followed by ICP-MS for Au, Pt and Pd (Doc 600, now FA003).
- Total acid digestion followed by ICP-OES for Cr, Cu, Ni and S (Doc 214, current code is MA101).
- Selected samples have been analysed for Rh, Pt, Pd and Au using fire assay lead/palladium collection followed by ICP MS (code FA004).
- A subset of these samples have been analysed for Rh by fire assay with nickel sulphide collection (code NSF001).
- Small-scale aqua regia digestion followed by ICP-OES for Cr, Cu, Ni and S (AR201). This method was used for check analysis only and not for primary samples.

Genalysis was used as an additional assay laboratory for portions of 2011 and 2012 and used the following analytical methods:

- Fire assay lead collection followed by ICP MS for Au, Pt and Pd (method code FA25/MS).
- Multi acid digestion followed by ICP-OES for Cu, Cr, Ni, and S (method code 4A/OM).
- Aqua regia digestion followed by ICP/OES for Cu, Cr, Ni, S (method code AR01/OM). This method was used for check analysis only and not for primary samples.

11.4.4 Check Sample Analysis

Genalysis in Perth used the following analytical methods (laboratory codes included in parentheses):

- Fire assay lead collection followed by ICP-MS for Au, Pt and Pd (FA25/MS).
- Multi acid digestion followed by ICP-OES for Cu, Cr, Ni and S (4A/OM).
- Sieve test as indicated by individual sample breakdown (SV02).
- Aqua regia digestion followed by ICP-OES for Cu, Cr, Ni, and S (AR01).

In contrast, the Johannesburg branch of Genalysis used the following methods on selected samples:

- NiS fire assay for Au, Pt, Pd, Rh, Ru, Os and Ir (NS25/MS).
- Pd Collector fire assay for Rh (FA25P/OE).

ACME used the following protocols:

- 3B03 - Lead fire assay followed by ICP MS for Au, Pt and Pd.
- Group 1E – Four-acid digestion followed by ICP OES (for Al, Ca, Cr, Cu, Fe, Mg, Ni and S).
- Group 1D01 - Aqua regia digestion followed by ICP OES (for Al, Ca, Cr, Cu, Fe, Mg, Ni and S).

No check samples have been submitted since October 2012.

11.5 Quality Assurance and Quality Control

11.5.1 AMK and ATS QA/QC

AMK and AST drilling was completed to support Mineral Resources amenable to open pit mining methods. Ivanplats is no longer considering the open-pit option. A detailed description of the AMK and ATS QA/QC is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted.

11.5.2 UMT QA/QC

Control Samples

As is prevalent throughout the industry, all laboratories employed by the Platreef Project use their own quality-control materials (blanks, pulp duplicates, standards) within each laboratory process batch. Laboratories routinely re-ran batches that failed their quality control requirements. Batches, which vary in size, typically include two duplicates, one or two blanks and a laboratory reference material. Results of laboratory quality controls are included in the laboratory reports. These results are informative because they show what the laboratory considers to be acceptable performance; batches showing inadequate performance are re-run, and the original assays are not part of laboratory final reports.

The Platreef Project inserted coarse reject duplicates, field blanks, and packets of certified reference materials (CRMs) in order to independently monitor laboratory performance.

Blanks

Blanks utilised locally sourced natural rock materials that have <10 ppb concentrations of Au, Pt, and Pd, but have copper concentrations of < 35 ppm and Ni concentrations of < 65 ppm. Blanks underwent preparation steps and therefore provide an upper limit on levels of contamination caused by preparation. One blank sample is inserted every 20th sample.

Coarse Reject Duplicates (CRDs)

Coarse reject duplicates were created by the preparation laboratory by routinely making a sample from the coarse reject of every 20th sample, and assigning it the same sample number as its duplicate pair, with the addition of a suffix CRD.

Certified Reference Materials (CRMs)

All sample submissions included packets of CRMs inserted every 20th sample. These CRMs were purchased from commercial African Mineral Standards (AMIS, Johannesburg), and/or in-house SRMs were used; the in-house SRMs were made from composites of drill sample coarse rejects that were prepared by SGS (Johannesburg), with best values assigned by Amec Foster Wheeler based upon round-robin results. Details are provided in Acuity (2015), Reid (2011, 2014) and Long (2013a). In-house SRMs were phased out as appropriate materials became available from AMIS.

As many as 15 CRMs and SRMs have been used extensively enough to compare Ultra Trace's mean results of each for comparison to best values. Currently, nine CRMs are in use. Excluding outliers that triggered follow-up investigation (for control insertion mix-ups) and in very rare cases remedial re-assaying of some laboratory batches, the average of the Ultra Trace results is within 10% of the certified value for the major elements of interest (Ni, Pt, Pd, Au, Cu) and in most cases for the added element, sulphur. Ultra Trace results for Cr are much lower than the AMIS certified values (based upon fusion or XRF pellet analysis), indicating that the multi-acid digestion method is not adequate for this element. This is a known problem with acid digestion for Cr.

Check Assays

Approximately 5% of drill sample pulps previously assayed by Ultra Trace were forwarded, along with blind CRMs and blanks, to Genalysis, Perth. Genalysis performed the same assay suite, plus aqua regia digestions for Ni and Cu. Agreement was usually adequate and, in all cases where it was not, samples were re-assayed by both laboratories to resolve the problems. The assay database was routinely updated where remedial assaying was performed.

In 2010, Genalysis began to exhibit some systematic errors in its acid digestion assays, likely attributable to introduction of new heating blocks. The problem was eventually resolved, but the decision was taken to suspend sending check assays to Genalysis. Sample pulps were instead submitted to Acme Laboratories, Vancouver.

Prior to suspending submissions to Genalysis, the Platreef Project used Genalysis aqua regia results to estimate, for each rock type, the fraction of total Ni likely to be in sulphide 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 sulphide collector fire assay to validate the conventional lead collector fire assay results for Pt and Pd, and to determine the grade of other PGEs, particularly Rh. NiS fire assays return lower Au results and are not regarded as reliable for Au. Pt and Pd results were on average slightly higher (about 5%) compared to the lead collector fire assays.

No check assays have been completed since October 2012.

11.6 Databases

The drillhole data were maintained in a Fusion database, created by Century Systems Technologies Inc. The Fusion database is maintained at the Platreef Project site. All available drillhole data including data from the AMT and ATS drill campaigns have been captured in the database.

The drillhole database was migrated from the Fusion database to an acQuire database on 1 May 2013.

11.6.1 AMT and ATS Data Entry

A description of the AMK and ATS data entry is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted.

11.6.2 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 current and planned 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.

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. Dr Parker and Mr Kuhl are 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 drill programmes were analysed by independent laboratories using industry-standard methods.
- Typically, Platreef drill programmes included insertion of blank, duplicate and SRM or CRM samples.
- Data that were collected were subject to validation, using in-built programme triggers that automatically checked data on upload to the database.
- Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar coordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards.
- Sample security has relied upon the fact that the samples were always attended or locked in the onsite sample preparation facility.
- Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.
- Current sample storage procedures and storage areas are consistent with industry standards.

12 DATA VERIFICATION

Several reviews of the database have been made since 2002. These include Amec Foster Wheeler reviews and those performed by independent consultants. Database audits were performed by Amec Foster Wheeler in 2007, 2010, 2012, 2014 and 2015 to ensure its suitability for resource estimation.

12.1 Amec Foster Wheeler Site Visits

12.1.1 Site Visits by QPs During UMT Drilling

In the April 2010 site visit (Kuhl, 2010), Amec Foster Wheeler completed a database audit and performed field checks of drill collars. No significant errors were noted that could affect Mineral Resource estimation.

Dr Harry Parker visited the Platreef Project site in March 2011 and reviewed geology logging of 12 drillholes (Parker, 2011).

Mr Kuhl also visited site in July–August 2011, and observed drilling operations and reviewed geology logging.

Mr. Kuhl completed a site visit between 25 January and 2 February 2012 and reviewed the TCU geological interpretation in cross-sections and drill core. Mr. Kuhl also visited drilling locations.

Dr. Parker next visited the site from 16 to 22 November 2012. Dr. Parker inspected core and geology logging in nine holes. He verified collar coordinates for 10 holes using a hand-held GPS unit and collected 20 witness samples from holes drilled since March 2011 and personally supervised their splitting and bagging for submission to sample preparation at Set Point (Mokopane). Dr. Parker also reviewed the structural interpretation.

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 retrieved 20 witness samples from Set Point (Mokopane) collected by Dr. Parker and supervised the packaging and shipment to the Ultra Trace Laboratory.

Dr. Parker visited the site from 26 to 27 August 2014. Dr Parker reviewed the progress of geological interpretations based on re-logging of drill core. Dr. Parker also reviewed data collection programs.

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.

Dr. Parker also visited the site between 6 and 10 July 2015. He reviewed the core logging and structural interpretation supporting the Mineral Resource Model.

12.1.2 Other Amec Foster Wheeler Site Visits

Mr Scott Long visited the site under the supervision of Dr Parker 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.

12.2 ATS and AMK Database Reviews

A description of the AMK and ATS database reviews is available in the September 2012 Technical Report (Parker et al., 2012). No issues that could affect Mineral Resource estimation were noted during the Fusion database reviews.

Given the problems identified while migrating the database from Fusion to AcQuire for the UMT data (see Section 12.4), Amec Foster Wheeler recommends a complete review of the assay database for the ATS and AMK drilling be completed against assay certificates prior to using ATS and AMK assay data for Mineral Resource estimations.

12.3 UMT Database Reviews

12.3.1 March 2010 Review

Amec Foster Wheeler completed a database review in April 2010. The review included collar and survey checks for 53 UMT drillholes completed after the 2007 database review of open pit data. All collars and surveys were checked against supporting documents. Lithology and density data were compared to supporting documents for five of the additional 53 drillholes. Assay data were checked for 5% of the assays from the additional 53 drillholes. No issues that could affect Mineral Resource estimation were noted.

12.3.2 August 2012 Review

Amec Foster Wheeler 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. Amec Foster Wheeler concluded the database was acceptable to support Mineral Resource estimation.

12.3.3 December 2012 Review

Amec Foster Wheeler completed a database review in December 2012 for drillholes completed after August 2012 (Yennamani, 2013). Amec Foster Wheeler compared the collar and downhole surveys, geology logs and assay data (Au, Pt, Pd, Ni, and Cu) against supporting documents. Minor errors were identified and corrected by Ivanhoe staff. The assay database was considered acceptable to support future Mineral Resource estimation.

12.3.4 February 2014 Data Review

Amec Foster Wheeler completed a database review in February 2014 for drillholes completed after December 2012. Amec Foster Wheeler compared collar surveys, downhole survey and geology logs against supporting documentation. Amec Foster Wheeler verified >95% of the assay results against original laboratory reports. Amec Foster Wheeler concluded that the drillhole database was acceptable to support Mineral Resource estimation.

12.3.5 October 2014 Data Review

Amec Foster Wheeler completed a database review in October 2014 for drillholes completed after February 2014. Amec Foster Wheeler 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. Amec Foster Wheeler concluded the database is acceptable for Mineral Resource estimation. A programme of assaying selected samples for Rh was proposed and initiated.

12.3.6 May 2015 Data Review

Amec Foster Wheeler completed a database review in May 2015 for drillholes completed after October 2014. Amec Foster Wheeler 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. Amec Foster Wheeler concluded the database was acceptable for Mineral Resource estimation.

12.4 Database Migration

Although data capture into the AcQuire database was initiated on 1 May 2013, the final database migration from Fusion to AcQuire was not completed until Q1 2014. Amec Foster Wheeler 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 Amec Foster Wheeler checking approximately 100% of the assay data for the UMT drillholes against laboratory certificates. A review of the assays from the AMK and AMT drillholes is pending.

12.5 Quality Assurance and Quality Control Results

Ivanhoe monitors QA/QC data (blanks, duplicates and CRMs) when results are received. If results are not within established limits, re-analysis of samples in the vicinity of the failing controls are requested. The data are not accepted unless re-assays produce acceptable results. Overall, a small number of reports have been rejected, and these have been remediated. Amec Foster Wheeler periodically reviewed QA/QC data.

12.5.1 UMT QA/QC (To March 2011)

Amec Foster Wheeler obtained and reviewed the available QA/QC data for the UMT drilling in March 2011. Amec Foster Wheeler noted:

- All Ultra Trace means on SRMs are within 5% of recommended values for the five major elements of economic interest (Pt, Pd, Au, Cu, Ni). Results are sufficiently accurate for Mineral Resource estimation for all five elements of economic interest.
- Generally, the results for Au, Pt, and Pd blanks were satisfactory. Significantly poorer performance was noted for Cu and even more so for Ni results. The apparent poor performance for Cu may be a consequence of a low bias in Set Point Cu assays (used to certify the blank material). Nickel values were of concern because approximately 80% of samples exceeded the 8 ppm value stated by Set Point.
- Genalysis results for Cu, Pt, Pd, and Au were in line with the SRMs, but Genalysis showed a low bias for Ni. Amec Foster Wheeler 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 sulphur 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.

Amec Foster Wheeler evaluates 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.

Amec Foster Wheeler noted that Ivanhoe were not submitting pulp duplicates as part of their QA/QC programme, and recommended that Ivanhoe use Ultra Trace's reported pulp-duplicate results to assess the precision of pulp duplicates.

12.5.2 UMT QA/QC March 2011 to June 2012

Amec Foster Wheeler obtained and reviewed the available QA/QC for the period between March 2011 to June 2012. Results were as follows:

- Approximately 3,100 blanks were passed through preparation and assay during the period. Three clusters of low-grade contamination were found in three different drillholes (UMT 146, 155, and 181), all assayed by Genalysis. Indications were that the contamination likely occurred during sample preparation. The level of contamination is too low to have any impact on the future use of the samples in Mineral Resource estimation.
- The Platreef Project's increased drilling rate necessitated using Genalysis and Set Point laboratories, in addition to Ultra Trace. Amec Foster Wheeler separated the results by laboratory and calculated each laboratory's median result for each element of interest for each AMIS CRM. Results showed acceptable agreement between the laboratories.
- Multi-acid digestion results show good accuracy by all laboratories for copper and nickel but pronounced low biases by Genalysis and Ultra Trace for Cr. Set Point does not report Cr results. The Cr assays are not accurate by multi-acid digestion. Reliable Cr results most likely would require a fusion followed by reading by XRF. The low bias seen here is consistent with that seen previously in Ultra Trace results.

Except for Cr, which is not used in the resource estimations, accuracy of these elements is sufficient by all laboratories for use in estimation of Mineral Resources.

In 2016, Amec Foster Wheeler (Reid, 2016a) reviewed the results of check samples submitted in 2011 and 2012.

Between 28 June 2011 and 25 October 2012, Ivanhoe submitted 20 batches of check samples (comprised of a 5% selection of available pulp material) to ACME Laboratories (Acme) in Vancouver, Canada. Although a review of the included CRMs showed poor performance with respect to the CRMs by ACME, the check assay results were generally within 5% of the primary assay laboratory results. Only Pd showed a slight bias outside of ± 5%; Pd results indicated the primary laboratory was 5.5% lower than ACME check results.

12.5.3 UMT QA/QC June 2012 To July 2014

Amec Foster Wheeler obtained and reviewed the available QA/QC for the period between June 2012 to July 2014. Results were as follows:

- Approximately 899 blanks were passed through preparation and assay during the period. The average results for Pt, Pd and Au were less than 5 ppb, and the average results for Ni and Cu were less than 20 ppm. This is comparable to previous results. The level of contamination observed is too low to have any impact on the use of the samples in Mineral Resource estimation.
- Eight CRMs were submitted for analysis. The overall relative bias for the CRMs is within 5%, and the assay accuracy is sufficient for resource estimation.
- Duplicate results from coarse reject material indicate acceptable precision is obtained by Ultra Trace.

Ivanhoe did not submit any samples from this period for check assays.

Based on the above results, Amec Foster Wheeler is of the opinion the Pt, Pd, Au, Ni, and Cu assay results for this period have sufficient accuracy and precision to support resource estimation.

In October 2012 Ivanhoe submitted three batches to Genalysis Laboratory Services Pty Ltd (Genalysis) in Perth, Australia and one batch to Ultra Trace Assay Labs (Ultra Trace) in Perth, Australia. A review of the included CRMs showed poor performance with respect to the CRMs by Genalysis while there were too few results from Ultra Trace to express an opinion on assay accuracy.

The check assay results from Geanlysis 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 checks results from Ultra Trace were within 5% of the primary assay laboratory results with the exception of Au, which showed a 10.9% negative bias. This indicates the primary assay laboratory results are lower than the check assay (Ultra Trace) results.

Due to the low-grade nature of the Au check samples, these biases are not considered to be material.

Ivanhoe has not submitted additional samples for check assay since October 2012.

12.5.4 UMT QA/QC July 2012 To July 2015 (Acuity Geoscience)

Mr. Dale Sketchley of Acuity Geoscience Ltd. (Acuity) completed a review on the QA/QC data available for drilling completed between July 2012 and July 2015 (Acuity, 2015). Results include:

- Results for Au, Pt and Pd from 920 blank samples indicated only two samples above the 40 ppb threshold for Pt and Pd. Copper and Ni results from 846 samples indicated two samples with Cu results and three samples with Ni above the 100 ppm threshold. Approximately 899 blanks were passed through preparation and assay during the period. The average results for Pt, Pd and Au were less than 5 ppb, and the average results for Ni and Cu were less than 20 ppm. This is comparable to previous results. The level of contamination observed is too low to have any impact on the use of the samples in Mineral Resource estimation.
- Acuity's review of results from the nine CRMs submitted for analysis determined that the overall relative bias for the CRMs is within 5% and concluded the assay (with exception of Au results below 75 ppb) accuracy is sufficient for resource estimation.
- Duplicate results from coarse reject material indicated acceptable precision with the exception of gold as obtained by Ultra Trace. Acceptable precision was defined as having an absolute relative difference at the 90th percentile of 20%. Gold was observed to have a difference of 31-37%. Based on limited tests, Acuity has recommended finer grinding (85-90% passing 75 µm).

Check assay data were not reviewed in Acuity's memorandum.

Based on the above results, Acuity was of the opinion the Pt, Pd, Au (with the exception of values less than 75 ppb), Ni, and Cu assay results for this period have sufficient accuracy and precision to support resource estimation. Initially Acuity stated Au results below 50 ppb should be excluded from resource estimation; however, subsequent discussions between Amec Foster Wheeler 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%. Amec Foster Wheeler reviewed the impact of the Au grade reduction (Amec Foster Wheeler, 2016) and found the impact to be negligible. The recommended correction was not made.

12.6 Amec Foster Wheeler Witness Samples

Four groups of witness samples have been collected at Platreef by Amec Foster Wheeler, in April 2010, February 2011, November 2012 and May 2015. The purpose of collecting these samples was to confirm the presence of mineralisation.

12.6.1 2010 Witness Samples

Amec Foster Wheeler 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. Amec Foster Wheeler (Long and Parker, 2011) concluded a larger number of samples were required in order to achieve a reliable verification of the original assays or if large differences were found, showing them to be statistically significant.

12.6.2 2011 Witness Samples

Amec Foster Wheeler collected a second group of 260 witness samples (Long, 2011a)

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 sulphides in the original (half core) samples.

Pt and Pd returned lower average results in the quarter-core sampling compared to the original sampling. However, the results of the inserted CRMs indicated that the Pt and Pd results had a low (but within the acceptable range) bias for Pt and Pd, or around 5%, and the CRMs associated with the original results for these samples did not.

After applying a correction to the Pt results for the low bias shown by CRM results, the difference between the original and new results was no longer statistically significant. However, the correction applied for a low bias shown by CRMs for Pd is smaller, and the data have less variance; consequently, the difference between the original and re-assay results remains statistically significant after applying a correction.

12.6.3 2012 Witness Samples

A third set of witness samples were taken in November 2012, and assay results were received in January 2013. Original and witness assay values were compared for Pt, Pd, Au, Ni, Cu, Cr, and S. Comparison of means of witness samples to means of original results show agreement within 5% for base metals, sulphur, and Pd. The original Au mean is 19% lower than the witness sample mean, and the original Pt mean is 14% higher than the witness sample mean.

Further investigation of Au and Pt results showed the percentage of occurrences where the original result of a pair was less than the witness sample result was not statistically significantly different from the expected 50–50 distribution expected. In the case of Pt, nine out of 20 pairs had a lower Pt result for the original assay.

12.6.4 2015 Witness Samples

Amec Foster Wheeler obtained a fourth set of 20 witness samples from recent drillholes in May 2015, and assay results were received in June 2015. The samples were collected from quarter-split core contained in intervals within the 1+2+3g/t 3PE+Au grade shells in the TCU stratigraphy. Original and witness assay values were compared for Pt, Pd, Au, Rh, Ni, Cu, Cr, and S and graphed. The resulting charts do not suggest any obvious sample mix-ups or outliers that are not a consequence of variation in grade. Amec Foster Wheeler concluded no bias is present between the assay values in the database and the values obtained from the witness samples.

12.7 Verification of Grind–Assay Function

Amec Foster Wheeler selected 92 pulp samples of pyroxenite and harzburgite for screening at 75 µm, because metallurgical test data available in 2011 indicated that there may be enhanced 2PE+Au grades related to the grinding of pulps, particularly for harzburgite. XPS recommended a grind of 80% passing -75 µm. Long (2011b) concluded that over 90% of harzburgite sample pulps are likely to achieve the recommended grind quality. Hence no modification of the grind protocol was recommended, nor was remedial work or further investigation considered warranted.

12.8 Gold Variability Test Work

Acuity (Acuity, 2015a) conducted a number of tests to investigate the high variability of gold lead collector fire assays and to recommend improvements for sample preparation and assaying work. QA/QC monitoring work has shown that gold CRM and duplicate assays typically show high variability for all grade ranges, whereas companion platinum and palladium assays demonstrate high variability above approximately 1,000 ppb. The test work comprised varying the sample preparation grind size and litharge content of lead collector fire assay flux. Several important observations were noted, which have a bearing on the quality of data available for resource estimation.

- The high variability of gold appears to be at least partly related to pulverising grain size and flux composition. Test work showed that the 106 µ pulverising size is not optimal, resulting in lower gold values with higher variability, whereas the 75 µ pulverising size is closer to optimal, resulting in higher values with lower variability. The grade increase is noticeable at higher gold values, and the variability increase is noticeable at lower gold values. Moreover, the variability of gold generally decreased with increasing flux litharge content.
- Platinum and palladium grade patterns are not as well developed as for gold, but where there are grade differences of more than several percent, the finer grind size samples returned higher grades. Variability is lower for all of the finer grind size samples compared to the coarser grind size samples.
- Previous metallurgical test work 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 test work 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 test work. Laser ablation test work would provide additional information on mineralogy and grain size relationships. Additionally, sample preparation protocols need to be revised to reduce the routine grinding size to 75 µm from 106 µm.

Although note (Section 12.7) that Long (2011b) thought that harzburgite samples were already being ground to 75 µm.

12.9 Rhodium Analysis Using a Palladium Spike

A separate evaluation of the Pd-spike method for Rh analysis was performed on a subset of 22 samples (plus three duplicate samples). This comparison showed that the addition of the Pd to the conventional fire assay did not affect the Au and Pt results, with means agreeing within 3%. A comparison of a much smaller subset where there were original fire assays by NiS fusion covered five samples plus two duplicate samples. The mean of the Pd spike method was about 4% lower than the NiS fusion result.

The number of pairs is too few for a meaningful statistical test, but the agreement in means suggests this method is likely working sufficiently well for estimating Rh content in Platreef samples. Additional data from sample pulps assayed by both methods are needed to further substantiate this interpretation.

Platreef routinely checks 5% of samples with elevated PGEs using NiS fusion fire assay. These data are consistent with the initial finding that Rh by Pd spike produces a slightly lower (3 to 5%) value than that obtained by NiS fire assay. Pt and Pd are also slightly higher by a similar amount by NiS fusion compared with Pb fusion fire assay. Gold however, is slightly lower by the NiS fusion method.

As a result of the reviews of the check data, Dr Parker and Mr Kuhl concluded that the check data validate the original Rh assays.

12.10 Comparison of Ultra Trace and Mintek Assays

In 2013, Amec Foster Wheeler 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 test work at Mintek.

Mintek's assays included fire assay results for Au, Pt, Pd, and some Rh (only on samples with elevated PGEs); Leco total sulphur; 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 test work.

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, Amec Foster Wheeler informed Mintek of their poor Ni accuracy and requested re-assay using an atomic absorption instrument. Amec Foster Wheeler 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, Amec Foster Wheeler 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 Amec Foster Wheeler'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. Amec Foster Wheeler investigation of Mintek internal quality controls revealed that they were relying upon two SRMs that had not been assayed by any other laboratory.

Later in March 2013, Platreef QC Manager Annelien Parsons obtained all available pulp rejects from Mintek test work, 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):

Adjusted Ni = 0.87*Mintek ICP Ni + 207 ppm.

This equation shows essentially no adjustment is required for low nickel values, such as around 1,000 ppm, which is the tails assay for nickel; however, the head assays for metallurgical samples (>2,000 ppm Ni) will be affected.

Mintek's stated best values on its two in-house SRMs were also found to overestimate Ni by a similar amount, indicating a long-standing high bias in Mintek Ni results that covers all the Mintek metallurgical test work performed on Platreef samples in Q4'12 (Long, 2013b).

12.11 Comments on Section 12

Amec Foster Wheeler has been involved in the Platreef Project since 2001 and has conducted continuous monitoring of data collection and data entry. Minor problems have been identified and resolved by improving procedures at the site. In the opinion of Dr Parker, Mr Kuhl and their QA/QC and database specialist Mr. Reid, sufficient verification has been conducted to provide support that the data collected are suitable for use as a basis for Mineral Resource estimation.

Amec Foster Wheeler completed an audit of the UMT drillholes migrated to the Fusion database to the AcQuire database. Data checks included collar survey table, deviation survey table, geology tables, assay tables and specific gravity table. Errors were identified and corrected (King, 2015 and Reid, 2016b). Checks of the migrated assay table for the ATS and AMK drilling are still required.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

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

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

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

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

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

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

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

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

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

A three stage crushing (primary crushing within mining scope) and ball mill circuit has been identified as the preferred option, with lowest associated technical risk.

Bench-scale, batch open circuit flotation work was performed to derive the optimal flowsheet. This development work, indicated that a MF1 flowsheet, using the targeted copper-collector reagent suite and a split cleaner flotation circuit configuration in which the fast-floating fraction is treated in a separate cleaner to the medium and slow floating fractions resulted in optimal 3PE+Au, Cu and Ni recoveries for the targeted concentrate grades.

Once the optimum flowsheet had been derived, bench scale, batch open circuit flotation variability test work was performed.

In addition to the open circuit variability testing, locked cycle test work was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore schedule, with the focus placed on testing of an ore blend representing the first 5 years of mining. This locked cycle flotation testing achieved recoveries (3PE+Au) in the range 83.1%-88.7% at a saleable final concentrate of approximately 60g/t-95g/t (3PE+Au). Locked cycle testing on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries (3PE+Au) in the range 77.3%-85.5% with a final concentrate ranging approximately 80 g/t-120 g/t (3PE+Au).

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

This relationship between 3PE+Au head grade, mass pull and final concentrate grade was used to derive recovery algorithms, which express 3PE+Au recovery as a function of both head grade and target concentrate grade as summarised in Figure 13.1.

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

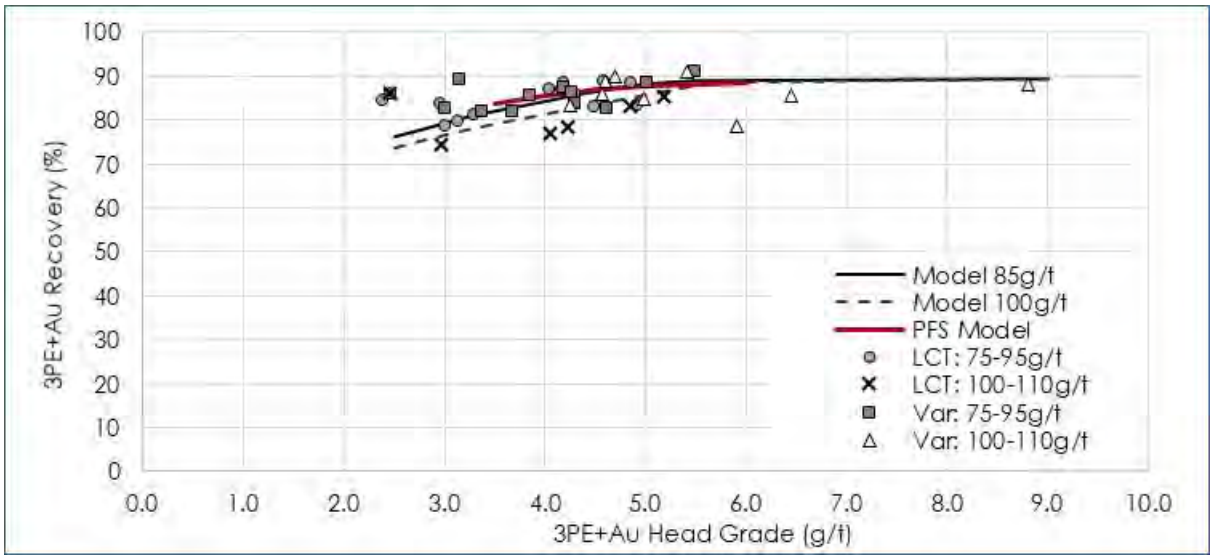


Figure by DRA, 2017.

13.2 Previous Metallurgical Test Work

Various metallurgical test work campaigns have been conducted using core samples from the Platreef deposit since October 2001. Prior to 2006, testing was conducted on predominantly lower-grade material from the potentially large open-pit area. In 2008, a deep drilling exploratory programme was launched, and the resource was updated to include deeper higher-grade material.

Between 2010 and November 2014, a series of metallurgical test work campaigns were carried out on the Platreef mineralised material as summarised in Table 13.1.

Table 13.1 Summary of Previous Metallurgical Test Work

Test Work 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 sulphides 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 flowsheet, commonly referred to as MF2 in South African processing terms was developed based on results from grind optimisation test work and reagent dosage testing. The development test work at XPS was unable to produce a concentrate grade of 80 g/t -100 g/t and PGE recovery to final concentrate was approximately 60%.
Phase 3 (2012) Managed by Ivanplats and Amec	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 flowsheet development and reagent scouting tests.	A single stage milling and flotation circuit, commonly referred to as MF1 in South African processing terms was developed. A reagent suite that included oxalic acid and thiourea addition with extensive conditioning time in the mill, prior to flotation, indicated that a PGE recovery of 83% could be achieved at a concentrate PGE (3PE) grade of 123 g/t, based on samples tested. High chrome media was used as grinding media in the laboratory mill.
Phase 4 (2012 – 2013) Managed by Ivanplats and Amec	Mintek	Drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and a composite containing 15%T1, 42.5% T2U and 42.5% T2L	Bench scale flotation testing included grind optimisation and flotation circuit development testing aimed at reproducing the SGS Phase 3 results.	The findings of the Mintek Phase 4 testing were largely in agreement with the SGS Phase 3 findings. A reagent suite that included oxalic acid and thiourea addition with conditioning in the mill prior to flotation indicated that a PGE recovery of 85% could be achieved at a concentrate PGE (4E) grade of 120 g/t. The type of grinding media used was also determined to be critical, with stainless steel and high chrome media consistently returning superior results compared to tests using carbon steel media. Locked cycle tests were conducted with an oxalic acid and thiourea in-mill conditioning time of 109 minutes which would not be possible to during commercial operation.

Test Work Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 5 (2013) Managed by Ivanplats and DRA	SGS Lakefield	Drill core samples representing T1, T2 upper (T2U) and T2 Lower (T2L) and footwall/contaminated zone (CZ). Testing was conducted on domain composites as well as ore blend composites representing the expected mine blend ratios.	Determination of comminution parameters (JK data, BRWi, BBWi, Ai) Bench scale flotation testing included grind optimisation and flotation circuit development testing aimed at obtaining optimised flotation conditions and expected performance for a reduced oxalic acid and thiourea in-mill conditioning time of 5 minutes. Dewatering and rheology characterisation test work.	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 5 minutes, indicated that a PGE recovery of 85% could be achieved at a concentrate 3PE+Au grade of 85 g/t. High chrome 440C grinding media was used in the laboratory mill. Phase 5 and Phase 6A test results formed the basis of the Platreef 2014 PEA flowsheet.
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 5 minutes. Modal mineralogy and PGE investigations on flotation concentrates and tailings. Dewatering and rheology characterisation test work.	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 test work indicated that a PGE recovery of 87.8% could be achieved at a concentrate PGE (3PE+Au) grade of 93 g/t. Mineralogical studies on tailings indicated that PGE losses to rougher tailings were comprised predominantly of PGE tellurides (43%) with lesser PGE arsenides (26%), gold (17%) and alloys (14%). Liberated PGE's in the rougher tailings were all in the <10 µm size fraction, which are regarded as non-recoverable by flotation. Phase 5 and Phase 6A test results formed the basis of the Platreef 2014 PEA flowsheet.

Test Work Description	Laboratory	Samples	Scope	Summary of Key Findings
Phase 6B (2014) Managed by Ivanplats and DRA	Mintek	Comminution variability test work 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 test work (Grindmill, UCS, CWi, Ai, BRWi, BBWi) Bench scale flotation testing aimed at optimizing the PEA flowsheet and reagent suite. The test work 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 PEA in Phase 5 and Phase 6A.	Bond ball work index test data and grindmill results confirmed previous competency test work highlighting that the ore can be classified as hard. Evaluation of alternate flowsheets and reagent suites indicated, that a standard flotation reagent suite, similar to that employed by platinum flotation operations in the vicinity of the Platreef deposit, provided comparable results to the oxalic acid and thiourea reagent suite when using an MF2 (mill-float followed by mill-float) circuit configuration. Preliminary MF1 (single stage mill and float) testing of an alternate reagent suite containing a targeted copper collector was able to achieve similar PGE, copper and nickel recovery and grades to those achieved for tests using an oxalic acid and thiourea reagent suite. In Phase 6B locked cycle test work using the MF1 flowsheet 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 g/t-96 g/t. A locked cycle test conducted on a footwall (contaminated zone) composite achieved 77% PGE recovery at a concentrate PGE (3PE+Au) grade of 57 g/t.

The findings from the Phase 1 – Phase 6B testing were presented in the Platreef 2015 PFS.

The Platreef 2015 PFS flowsheet was based on a single stage milling circuit followed by flotation (MF1) using an oxalic acid and thiourea reagent suite with the inclusion of a post mill reagent conditioning stage.

At the time of the Platreef 2015 PFS, MF1 testing of an alternate reagent suite using a targeted copper collector was able to achieve similar PGE, copper and nickel recovery and final concentrate grades to those achieved during test work using an oxalic acid and thiourea reagent suite. However, at the time, testing of this alternative copper collector reagent suite was ongoing, and further development of this flowsheet was incorporated into the Platreef 2017 FS metallurgical test work scope.

13.3 Platreef 2017 FS Test Work Samples and Scope

This Section summarises the metallurgical test work carried out between January 2015 and October 2016 at Mintek, called Phase 7, under the management of technical teams from Ivanplats and DRA.

These test work results were used in conjunction with the Platreef 2015 PFS test work results, from Phase 5 and Phase 6, to further quantify the flotation response, support flowsheet development, evaluate the degree of variability and derive metal recovery estimates.

The Phase 7 test work at Mintek included initial flowsheet development and optimisation test work as well as ore variability assessments. Comminution variability testing was conducted on approximately 1,286 kg of HQ drill core representing samples from the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ). Flotation testing was conducted on approximately 1,140 kg of quarter PQ drill core samples representing samples from the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ). Subsequent to the selection of Platreef 2017 FS samples, the CZ has been re-named in the latest geological 3 model, more appropriately, to the Footwall Assimilation Zone (FAZ). It is however still referred to as CZ in this report and within the Mintek test reports.

The Phase 7 flotation test work was aimed to further evaluate the alternative copper collector reagent suite identified in the Platreef 2015 PFS with the aim of deriving an optimal flowsheet for flotation variability testing. In addition to the bench scale flotation test work, comminution variability test work was conducted on the remaining variability drill core samples that were not tested in Phase 6B.

13.3.1 Platreef 2017 FS Sample Selection

13.3.1.1 Comminution Variability Test Samples

Comminution test work was conducted using samples representing the following:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example T1 from drill core hole TMT009.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example a blend of T1, T2U, T2L, CZ and HW from drill core hole TMT009.
- Domain composites representing blends of individual geometallurgical units from various drill core samples. For example a blend of T1 from drill core holes TMT009 – TMT016.

Table 13.2 Comminution Variability Samples

Core ID	Total Mass (kg)	T1	T2U	T2L	CZ	HW
TMT009	256.4	X	X		X	X
TMT010	149.2	X		X	X	X
TMT011	257.7	X		X	X	X
TMT012	180.0	X	X	X		X
TMT013	85.5	X	X			
TMT014	102.0	X		X		X
TMT016	255.3	X	X	X		X
Total	1,286.1	7	4	5	3	3*

*Three hanging wall composites representing T1, T2U and T2L hanging wall were tested

13.3.1.2 Flotation Test Work Samples

Ivanplats delivered quarter PQ drill core samples representing geometallurgical units T1, T2 upper (T2U), T2 Lower (T2L), hanging wall (HW) and footwall (CZ) to Mintek in Johannesburg. The Platreef 2017 FS flotation test work samples are listed in Table 13.3 below, and the locality of the drill core samples is shown in Figure 13.2.

13.3.1.3 Test Work Sample QA/QC

During both the PFS and FS metallurgical test campaigns, geological core log interval assay data from Set Point was compared to Mintek and SGS assay data to assess if any major bias or analytical deviation existed. No bias was noted between the PGE, Cu, Ni and S assay values in comparison to the core log data. In addition, 90% of all PGE, Cu, Ni and S analysis data (80 tests) was found to fall within a 10% variance between the assay data as compared to the core log data. Mintek's QA/QC reports were also reviewed and found to be satisfactory.

Flotation development testing was conducted on samples representing the following:

- Blend composites representing blends of geometallurgical units from various drill core samples. For example a blend of 14% T1, 51% T2U, 30% T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

Flotation variability testing was conducted on samples representing the following:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example T1 from drill core hole UMT104D1.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example a blend of T1, T2U, T2L and CZ from drill core hole UMT166D1.
- Blend composites representing blends of geometallurgical units from various drill core samples. For example a blend of 14% T1, 51%T2U, 30% T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

Table 13.3 Flotation Variability Samples

Core ID	Total Mass (kg)	T1	T2U	T2L	CZ	Point Sample Blend Composite
UMT104D1	12.0	X			X	X
UMT130D1	83.7	X	X		X	X
UMT135D1	104.0	X	X		X	X
UMT149D1	56.7	X		X	X	X
UMT158D1	54.0	X		X	X	X
UMT166D1	76.8	X	X	X	X	X
UMT186D1	57.6	X		X	X	X
UMT190D1	59.5	X	X		X	X
UMT205D1	42.0		X		X	X
UMT207D1	43.2		X	X		X
UMT211D1	35.9	X	X			X
UMT239D1	36.0	X		X		X
UMT265D1	66.3	X	X	X	X	X
UMT286D3	42.4	X	X		X	X
UMT295D1	17.3				X	X
UMT307D3	56.0	X	X		X	X
UMT312D2	41.9		X		X	X
UMT319D1	63.0	X		X	X	X
UMT321D1	82.2	X	X	X		X
UMT323D2	62.0	X	X	X		X
UMT318	5.2		X	X	X	X
UMT362	11.5	X		X	X	X
UMT373	6.7		X	X	X	
UMT374	12.6	X	X			X
UMT379	4.6	X	X			X
UMT384	7.4		X	X	X	
Total	1140.0	19	18	14	19	24

Figure 13.2 Variability Sample Drill Core Location

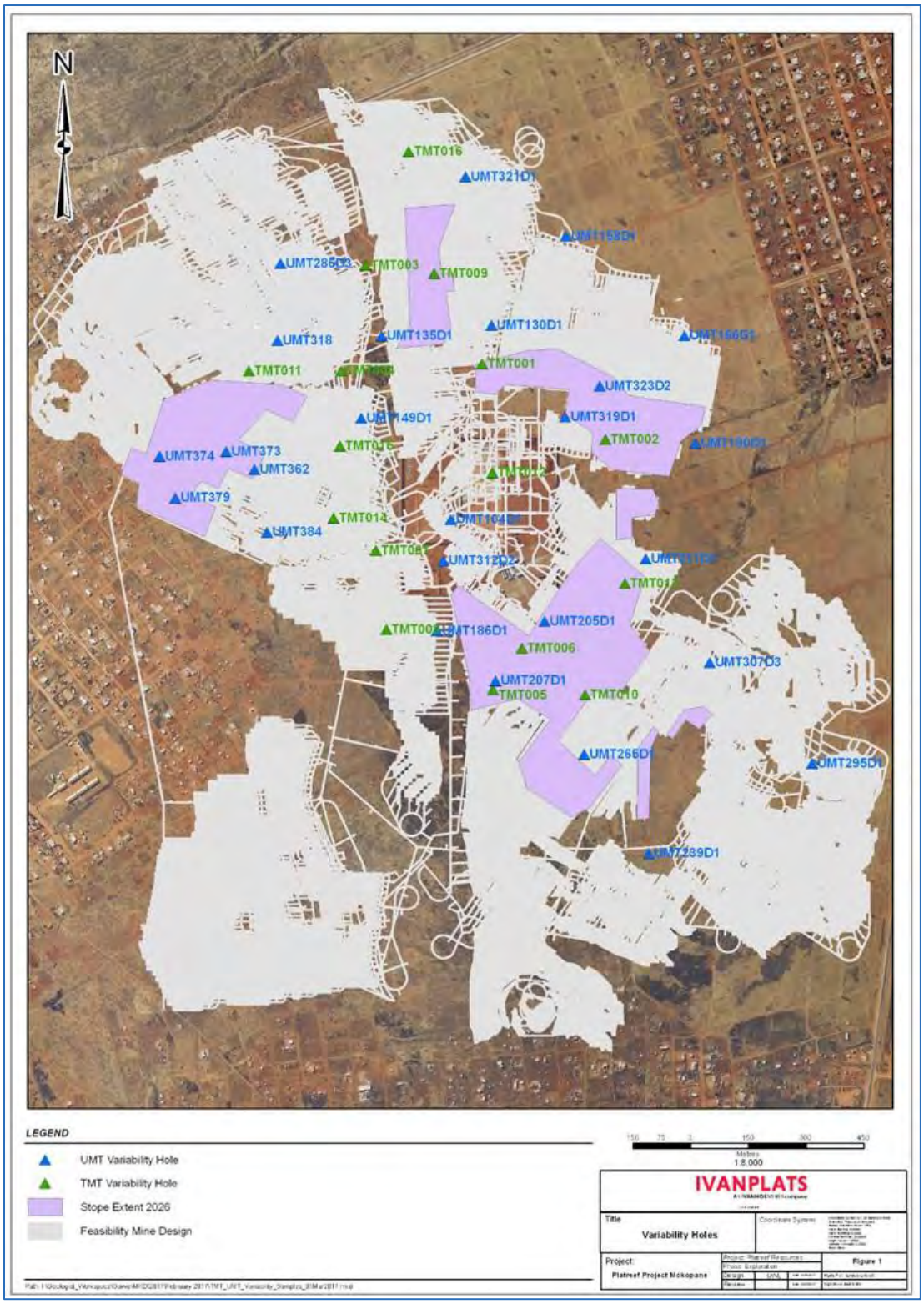


Figure by Ivanplats, 2017.

13.3.2 FS Test Work Scope

The Platreef 2017 FS test work scope is summarised in Table 13.4.

Table 13.4 Results of MF1 Copper Collector Flowsheet Development and Optimisation Testing

Test Work Description	Laboratory	Samples	Scope
Comminution Variability Testing	Mintek	7 x T1 4 x T2U 5 x T2L 3 x HW 1 x CZ	Bond crushability work index (CWI) Bond abrasion index (AI) Bond rod work index (BRWI) Bond ball work index (BBWI)
Chemical Composition and Mineralogy	Mintek	6 x T1 7 X T2U 3 x T2L 6 X CZ	Bulk mineralogy (x-ray diffraction (XRD)) Base metal sulphide (BMS) characterisation Platinum group metals (PGM) search
Open Circuit Flotation Development and Optimisation Test Work	Mintek	Platreef 2015 PFS Composite Development Composite 1 Development Composite 2 Development Composite 3 Development Composite 4 Development Composite 5	MF1 copper collector flowsheet development Sulphur optimisation Grade optimisation Flowsheet validation testing
Open Circuit Flotation Variability Test Work	Mintek	16 x T1 13 x T2U 10 x T2L 15 x CZ 26 x Blend Composites	Open circuit variability tests using optimised Platreef 2017 FS flowsheet
Locked Cycle Testing	Mintek	Development Composites 1,3,4 and 5 4 x Year 1-5 Blend 1 x Life of Mine (LOM) Blend	Locked cycle testing of interim flowsheets; Locked cycle variability testing of optimised Platreef 2017 FS flowsheet
Dewatering Test Work	Vietti Slurry Technology	Blend of FS Composites 1–4	Thickening test work

13.4 Ore Mineralogy

22 run-of-mine (ROM) samples obtained from individual drill cores, representing the geometallurgical units T1, T2U, T2L and Contaminated Zone (CZ), were sent for mineralogical analyses and characterisation. The objective of the mineralogical investigation was to identify the minerals present, the platinum group elements (PGEs), the base metal sulphides (BMS), grain sizes, liberation characteristics, mineral associations and to describe their mode of occurrence.

A maximum of 20 unsized polished sections of one portion were prepared for PGE and BMS search analyses by automated scanning electron microscope (AutoSEM). A second sub-sample was pulverised and submitted for X-Ray diffraction analysis (XRD). The XRD results were used to verify the minerals identified during by AutoSEM modal analysis.

The mineralogical study indicated that pyroxene is the predominant mineral Phase present throughout all 22 samples, followed by minor to trace amounts of olivine, dolomite, magnetite, serpentine, clay minerals, quartz, amphibole, chlorite, mica, plagioclase, talc, pentlandite, chalcopyrite and pyrrhotite.

13.4.1 PGE Relative Abundance and Liberation

A summary of the PGE relative abundance, liberation index and floatability is presented in Figure 13.3, to Figure 13.5.

Figure 13.3 Variability Sample PGE Relative Abundance

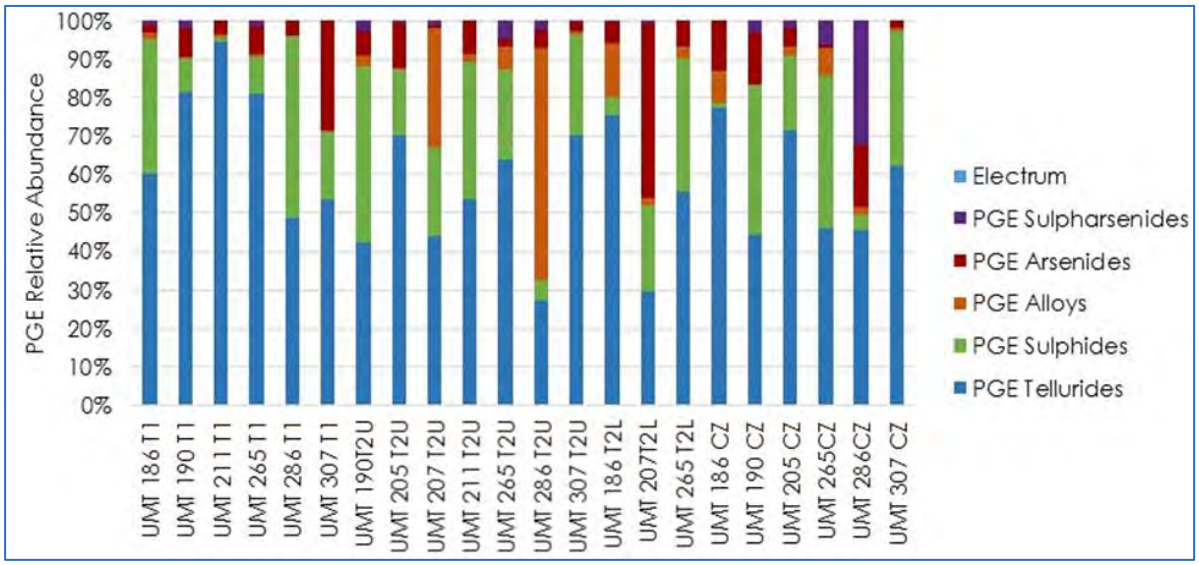


Figure by DRA, 2017.

Figure 13.4 Variability Sample PGM Liberation Index

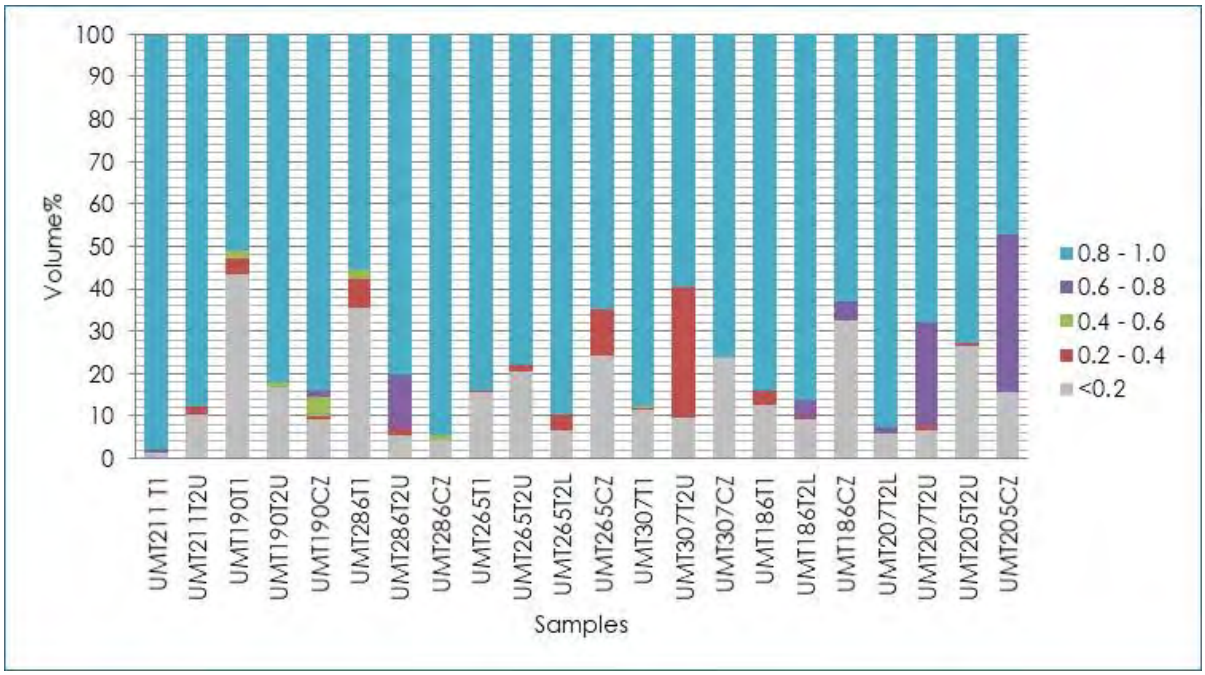


Figure by DRA, 2017.

Figure 13.5 Variability Sample PGE Floatability Index

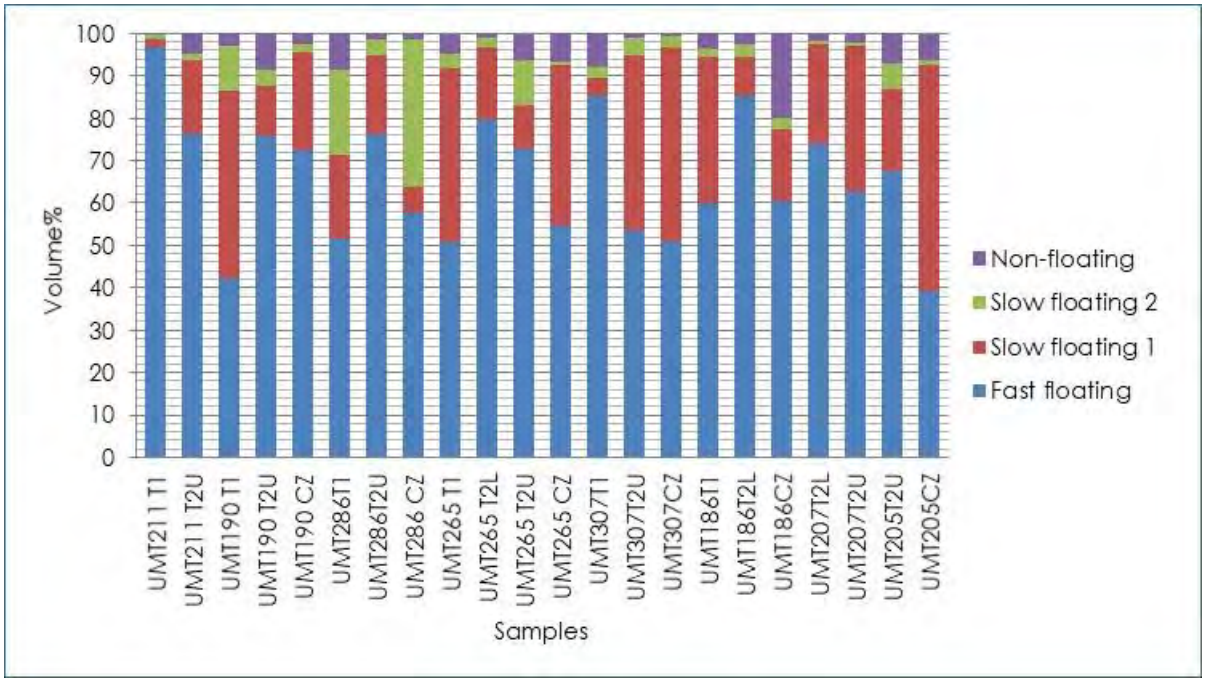


Figure by DRA, 2017.

Mineralogy indicated the following with regard to PGE speciation and liberation:

- PGE-tellurides are the dominant PGM species, averaging 59% by volume in the samples tested, followed by lower amounts of PGE-arsenides, PGE-sulphides and PGE-others. Laurite is present in minor to trace amounts in only a few samples.
- The majority of PGMs identified in the samples, reported to the 0-15 µm size class and were found to be well liberated, however, more than 20% (by volume) of PGM grains observed in the UMT190 T1, UMT265T2U, UMT265 CZ, UMT307 CZ, UMT186 CZ and UMT205 T2U samples were associated with relatively large gangue particles (<0.2 liberation index) compared to the other samples where low liberation indices occurred in minor to trace amounts.
- The PGM fast floating fraction varied in the range 40%-97% (by volume) and averaged 66%. A significant portion of slow floating PGMs were identified (3%-55% (by volume)).

13.4.2 Base Metal Sulphide Mineralisation and Liberation

A summary of the base metal sulphide mineralisation is presented in Figure 13.6.

Mineralogy indicated the following with regard to BMS mineralogy and Liberation:

- Pentlandite and pyrrhotite are the dominant sulphide mineral phases present, with minor to trace amounts of chalcopyrite and trace amounts of pyrite. The majority of pentlandite, pyrrhotite and chalcopyrite were found to be associated with free surfaces, and minor to trace amounts are associated with gangue and other BMS mineral phases.
- Pentlandite, pyrrhotite and chalcopyrite grain size was found to be variable. The majority of pyrite grains however, occur in the <10 µm size class.
- Pentlandite displays a similar high cumulative liberation in all samples except UMT265 CZ, where it is poorly liberated.
- Chalcopyrite displays a similar cumulative liberation in all samples except UMT211 T1, UMT190 T1, UMT265 T1 and UMT307 T1, where it is poorly liberated.
- Pyrite was found to be poorly liberated, while pyrrhotite was found to be well liberated in all samples except samples UMT186 T2L and UMT207 T2U.

Figure 13.6 Variability Sample Base Metal Mineralisation

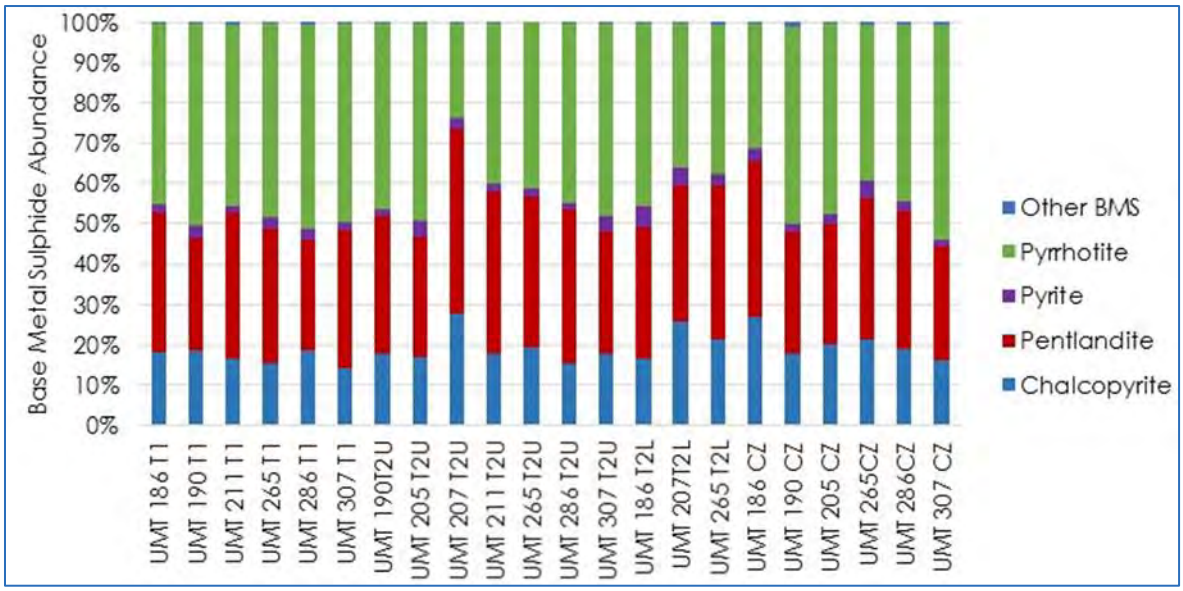


Figure by DRA, 2017.

13.5 Comminution Variability Test Work

In addition to the Phase 6B comminution variability test work that was conducted as part of the Platreef 2015 PFS, additional comminution variability testing, was conducted as part of the Platreef 2017 FS test work programme. In total 22 variability samples were tested as part of the Phase 7 comminution test work campaign.

13.5.1 Comminution Variability Test Results

A summary of all the comminution variability test results is presented in Table 13.5 to Table 13.7.

Table 13.5 Summary of Comminution Results for Year 1-5 Samples

Description		BRWi	BBWi	CWi (Avg)	Ai (Avg)
		kWh/t	kWh/t	kWh/t	g
Domain Point Samples					
TMT004	T1	-	-	12.5	0.40
TMT006	T1	-	-	13.4	0.27
TMT007	T1	-	-	14.1	0.34
TMT009	T1	13.7	21.3	16.8	0.47
TMT011	T1	14.2	21.7	21.1	0.34
TMT014	T1	15.7	20.7	20.9	0.37
TMT002	T2U	14.6	21.6	16.2	0.45
TMT004	T2U	-	-	12.4	0.38
TMT005	T2U	-	-	14.0	0.42
TMT006	T2U	-	-	12.9	0.33
TMT007	T2U	-	-	13.6	0.31
TMT009	T2U	15.6	22.2	23.4	0.41
TMT005	T2L	-	-	8.4	0.06
TMT011	T2L	19.1	23.6	15.3	0.12
TMT014	T2L	19.8	22.2	13.8	0.11
TMT004	CZ	16.7	19.0	12.4	0.41
TMT006	CZ	20.9	23.7	17.5	0.10
TMT007	CZ	16.0	20.7	10.6	0.27
Point Sample Blend Composites					
TMT 004	45%T1:55%T2U	16.2	21.6	-	-
TMT 005	20.%T2U:80%T2L	18.7	22.9	-	-
TMT 006	47%T1:53%T2U	16.8	20.0	-	-
TMT 007	70%T1:30%T2U	19.2	23.6	-	-

Table 13.6 Summary of Comminution Results for Year 6+ Samples

Description		BRWi	BBWi	CWi (Avg)	Ai (Avg)
		kWh/t	kWh/t	kWh/t	g
Domain Point Samples					
TMT001	T1	-	-	12.4	0.39
TMT003	T1	-	-	14.9	0.39
TMT008	T1	-	-	13.4	0.32
TMT010	T1	17.2	18.9	19.2	0.22
TMT012	T1	17.0	20.3	21.0	0.34
TMT013	T1	15.0	22.4	15.3	0.39
TMT016	T1	16.6	20.9	18.7	0.26
TMT001	T2U	-	-	16.2	0.45
TMT003	T2U	-	-	12.4	0.38
TMT008	T2U	-	-	14.0	0.42
TMT012	T2U	12.0	18.3	12.9	0.33
TMT013	T2U	19.6	23.6	13.6	0.31
TMT016	T2U	17.1	23.7	23.4	0.41
TMT003	T2L	-	-	15.5	0.20
TMT008	T2L	-	-	14.3	0.11
TMT010	T2L	19.2	23.4	14.2	0.12
TMT012	T2L	20.0	23.2	17.1	0.24
TMT016	T2L	23.0	25.4	21.1	0.25
TMT001	CZ	19.7	22.9	10.4	0.26
TMT003	CZ	17.1	21.3	12.5	0.44
TMT008	CZ	15.2	21.9	17.6	0.40
Point Sample Blend Composites					
TMT 001	41.% T1:59% T2U	16.6	20.7	-	-
TMT 003	25% T1:11%T2U:64% T2L	18.6	19.9	-	-
TMT 008	24%T1:3%T2U:73%T2L	18.4	21.9	-	-

Table 13.7 Summary of Comminution Results for the Domain Composite Sample

Description	BRWi	BBWi	CWi (Avg)	Ai (Avg)
	kWh/t	kWh/t	kWh/t	g
Domain Composites				
T1 (TMT 001 - 008)	-	20.5	17.4	0.35
T2U (TMT 001 -008)	-	22.2	26.0	0.38
T2L (TMT 001 - 008)	-	23.4	14.4	0.14
CZ (TMT 001 - 008)	-	22.4	16.7	0.35
CZ (TMT 009 - 016)	17.2	24.4	18.0	0.33
T1-HW (TMT 001 - 008)	-	20.3	22.1	0.27
T1-HW (TMT 00 1- 008)	16.6	20.8	13.0	0.36
T1-HW (TMT 009 - 016)	16.6	20.2	21.7	0.36
T2-HW (TMT 001 - 008)	-	21.2	11.8	0.40
T2-HW (TMT 001 - 008)	16.2	20.4	15.9	0.36
T2U - HW (TMT 009-016)	16.0	21.3	17.4	0.36
T2L - HW (TMT 009 - 016)	16.2	21.4	20.1	0.40

The frequency distributions obtained using all comminution results conducted in Phase 6B and Phase7 were analysed for each ore type and are presented in Figure 13.7 to Figure 13.10.

Figure 13.7 Probability Distribution of CWI Data

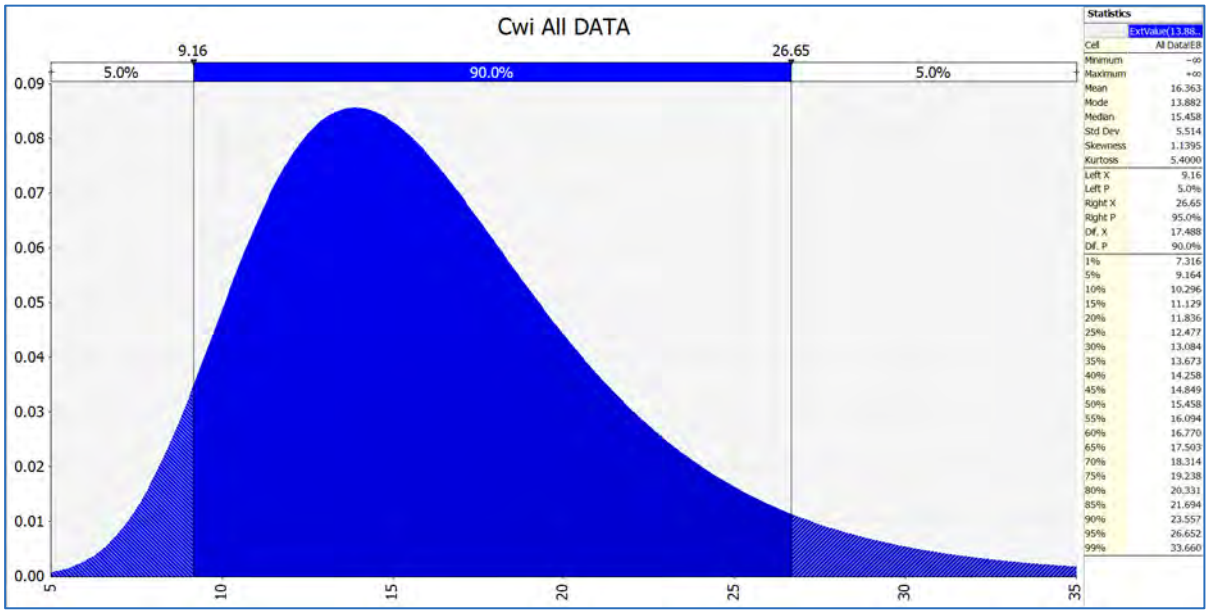


Figure by DRA, 2017.

Figure 13.8 Distribution of BRWI Variability Data

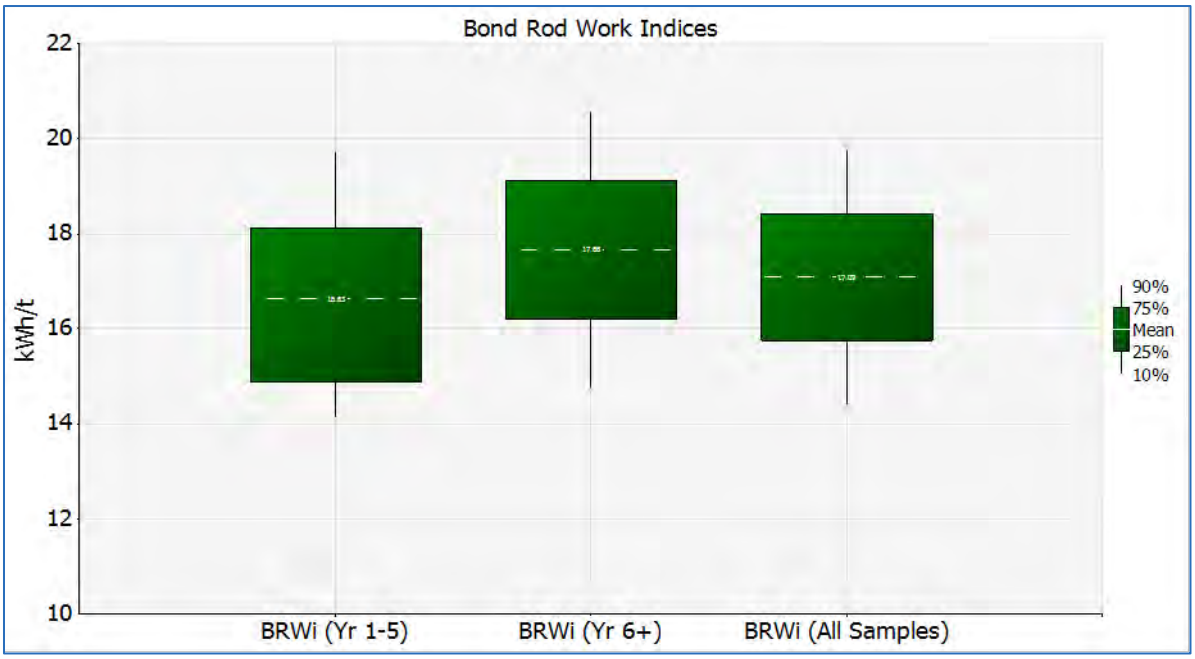


Figure by DRA, 2017.

Figure 13.9 Distribution of BBWI Variability Data

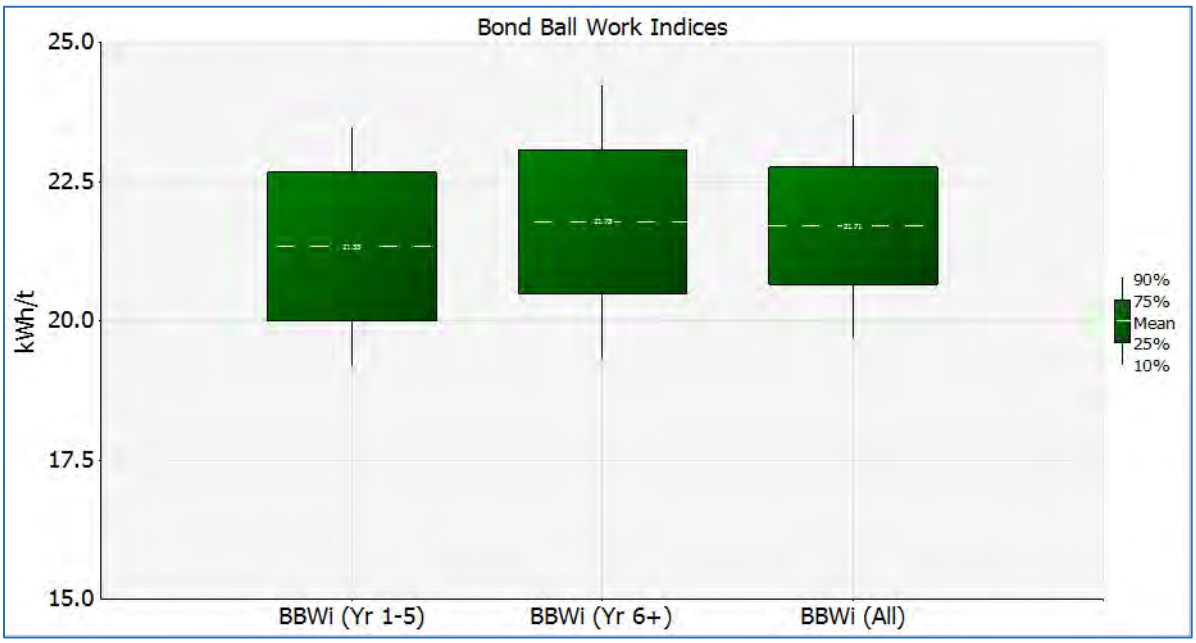


Figure by DRA, 2017.

Figure 13.10 Distribution of Ai Variability Data

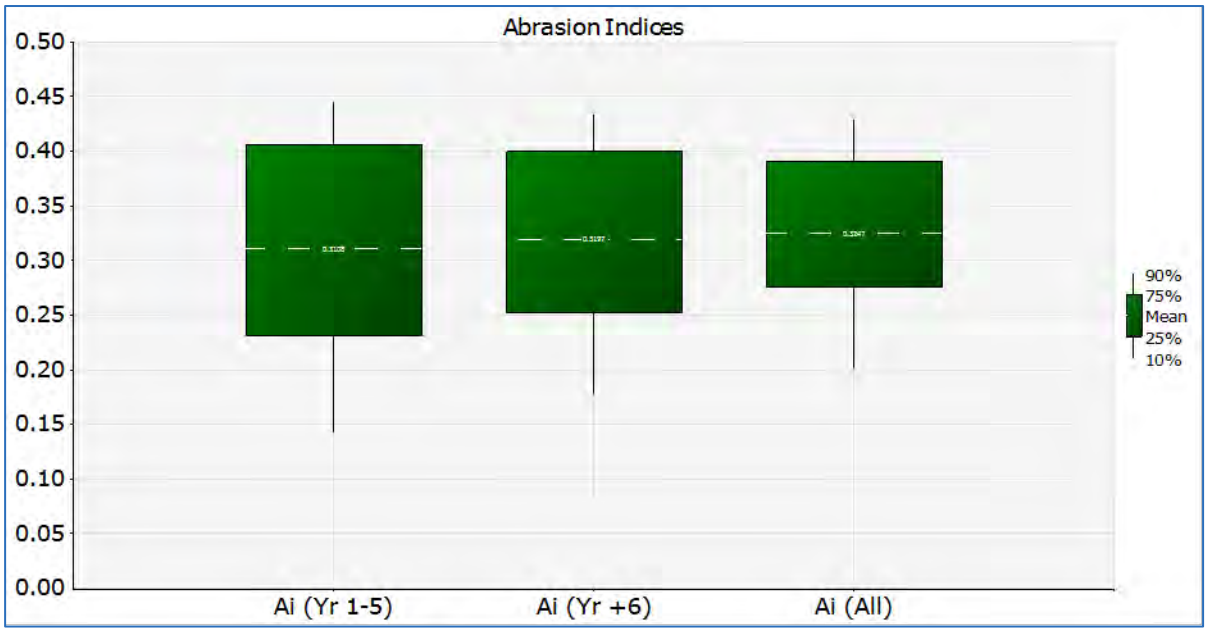


Figure by DRA, 2017.

Based on the crusher work index data the ore tested can be classified as being variable with medium-hard to very hard classification. The T2L and CZ ore types were noted as having a lower crusher work index as compared to the other ores and were classified as medium-hard to hard.

For the 90% confidence interval, Bond rod work indices were in the range 14.2 kWh/t -19.5 kWh/t in the first five years of mining and 15.0 kWh/t – 20.5 kWh/t in the later production years. The ore tested can be classified as medium-hard to very hard. The T2L domain samples indicated a higher resistance with respect to rod milling, with hardness classification of hard to very hard. The Bond rod work index data has indicated an approximate 1.0 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1-5 samples.

For the 90% confidence interval, Bond ball work indices were in the range 19.0 kWh/t - 23.0 kWh/t in the first five years of mining and 19.4 kWh/t – 24.2 kWh/t in the later production years. The samples were competent with respect to ball milling with hardness classification of hard to very hard. The T2L domain samples indicated a higher resistance with respect to ball milling. The Bond ball work index data has indicated an approximate 0.5 kWh/t increase in the mean ore hardness for the samples tested from the later years of mining as compared to the Year 1-5 samples.

The abrasion index was found to fall within a similar range of approximately 0.25 g -0.50 g, for all ore types, with the exception of T2L, with an abrasion index of 0.06 g – 0.25 g. The ore can be classified as having a medium abrasion tendency.

The comminution variability test work indicated no significant difference in the mean competency between the ore from the early production years (Year 1-5) and the later production years. Average Bond ball work indices indicate that the ore from the later production years was approximately 5% harder than ore from the first five years of mining.

There was a minor variation in hardness for the T1 and T2U domain samples as compared to the T2L and CZ domain samples as summarised in Table 13.8.

Table 13.8 Summary of Comminution Classification per Domain

Domain	CWi (kWh/t)			BRWi (kWh/t)			BBWi (kWh/t)		
	P_{50}	P_{85}	Class	Min	Max	Class	Min	Max	Class
T1	16.0	21.6	Medium - Hard	13.7	18.9	Medium - Hard	17.2	22.4	Hard - Very Hard
T2U	15.9	24.2	Medium - Very Hard	12.0	18.3	Medium - Hard	19.6	23.7	Hard - Very Hard
T2L	13.5	18.8	Medium - Hard	19.1	22.2	Hard - Very Hard	23.0	25.4	Very Hard
CZ	14.3	18.8	Medium - Hard	15.2	19.0	Hard	20.9	23.7	Very Hard
T1-HW	17.5	26.1	Hard - Very Hard	-	16.6	Hard	20.3	20.8	Very Hard
T2-HW	14.7	22.3	Medium – Very Hard	16.0	16.2	Hard	20.4	21.4	Very Hard

The 85th percentile test data was used as the design basis for the comminution circuit. The crusher circuit was sized based on the 85th percentile crusher work index results. The grinding circuit was sizing was based on 85th percentile, Bond work index results, which were used in combination with particle breakage rates derived from grindmill testing. The average ore hardness and abrasion index data has been used to derive the operating cost estimates, whilst effect of variability is also quantified.

13.6 Flotation Circuit Development Test Work

As part of the Platreef 2017 FS, open circuit flowsheet development and optimisation flotation test work was conducted. All test work was conducted at bench scale using industry standard testing techniques. The aim of this testing was as follows:

- Further development and optimisation of the MF1 flowsheet using the copper collector reagent suite. This testing included evaluation of optimal grind, circuit configuration, depressant addition and evaluating flotation circuit solution pH.
- Compare the performance of the optimised MF1 flowsheet using the copper collector reagent suite to the MF1 flowsheet using the oxalic and thiourea reagent suite presented in the Platreef 2015 PFS
- Evaluate the effect of rougher solids concentration
- Reduce pyrrhotite (sulphur) recovery to final concentrate
- Optimise final concentrate PGE (3PE+Au) grade

- Evaluate the effect of sample composition and head grade
- Finalise the flotation flowsheet for open circuit variability and locked cycle testing

13.6.1 Development Sample Assays

The measured head assay data for the development testing composites is presented in Table 13.9. The development Composites 1-4 represent an ore blend of 12%T1, 32%T2U, 50%T2L and 6% CZ, and the Platreef 2015 PFS Composite (PFS Composite) represents an ore blend of 15%T1, 42.5%T2U and 42.5%T2L.

Table 13.9 Development Sample Measured Head Assays

Test ID	Grade									
	3PE +Au	Pt	Pd	Rh	Au	Cu	Ni	Total S	MgO	Fe
	(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)
PFS Composite	3.99	1.79	1.87	0.10	0.23	0.21	0.39	1.00	24.36	10.13
Development Composite 1	3.25	1.34	1.62	0.06	0.23	0.20	0.39	1.28	23.02	9.15
Development Composite 2	4.11	1.83	1.94	0.10	0.24	0.21	0.40	1.04	26.36	10.13
Development Composite 3	3.48	1.53	1.61	0.10	0.24	0.18	0.36	0.81	27.56	10.15
Development Composite 4	5.71	2.49	2.70	0.19	0.33	0.21	0.46	0.97	27.06	10.40
Development Composite 5	5.39	2.24	2.70	0.15	0.30	0.21	0.46	0.97	25.07	9.77

Composites 1 and 2 were selected using the final Platreef 2015 PFS mine blend and represent a low 3PE+Au grade of 3.25 g/t (Composite 1) and the average expected PFS LOM 3PE+Au grade of approximately 4.1 g/t (Composite 2), while Composite 3 was selected to specifically target a lower sulphur grade in feed of 0.8% as per the expected sulphur grade in the early years of mining as compared to the >1.0% sulphur for Composites 1 and 2.

Composites 4 and 5 were selected using the final Platreef 2015 PFS mine blend and represent the higher grade 3PE+Au feed grade of approximately 5.50 g/t that will be targeted in the early years of mining.

13.6.2 Summary of Results

13.6.2.1 Development Testing of the MF1 Copper Collector Reagent Suite

Development testing of the MF1 copper collector flowsheet was conducted on the remaining PFS Composite sample, to evaluate the effect of grind, pH and depressant addition for the MF1 copper collector reagent suite. The remaining Platreef 2015 PFS sample was used in order to allow for a direct comparison to the Platreef 2015 PFS test results for the oxalic acid and thiourea reagent suite (PFS Test 69) and MF1 copper collector reagent suite (PFS Test 54).

A summary of the flowsheets used in this testing is presented in Figure 13.11and Figure 13.12. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.10.

Figure 13.11 Oxalic Acid and Thiourea Reagent Flowsheet (Platreef 2015 PFS)

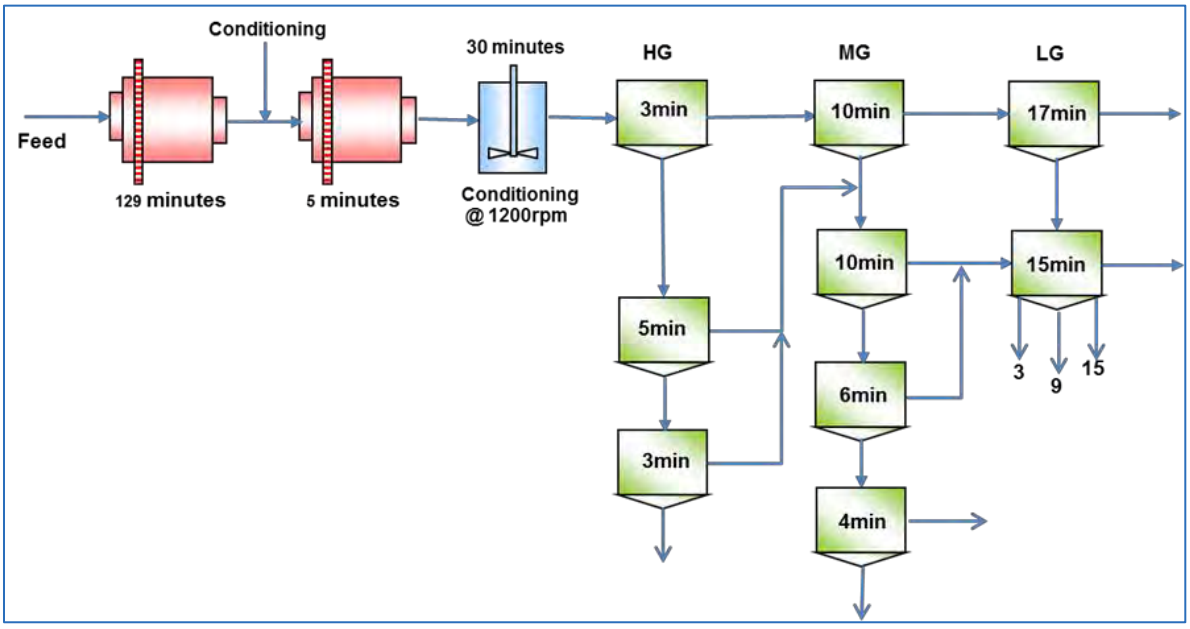


Figure by DRA, 2017.

Figure 13.12 Copper Collector Reagent Flowsheet (Platreef 2017 FS)

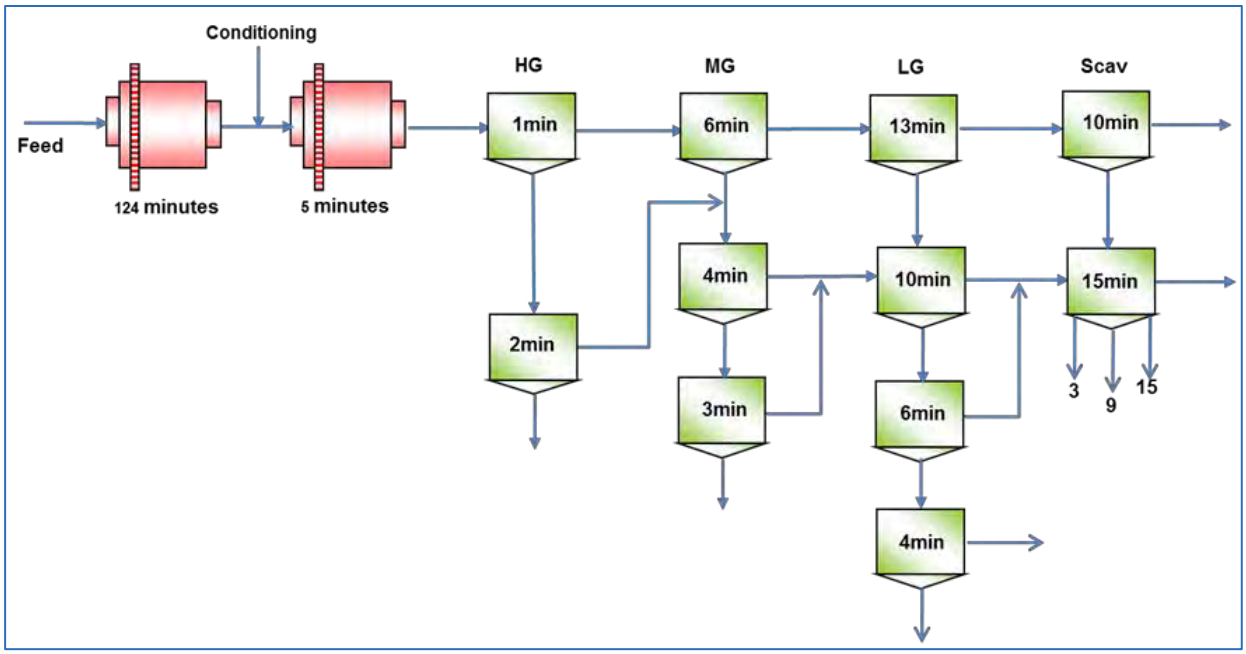


Figure by DRA, 2017.

Table 13.10 Results of MF1 Copper Collector Flowsheet Development and Optimisation Testing

Test No.	Sample	Reagent Suite	Grind P_{80}	Depressant		pH	Mass	Grade					Recovery				
				Rougher	Cleaner		(%)	3PE+Au	Cu	Ni	Fe	S	3PE+Au	Cu	Ni	Fe	S
				(g/t)	(g/t)			(g/t)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
69 PFS	PFS Comp	Oxalic & Thiourea	75	-	93	Natural	4.10	75.88	4.22	7.06	23.90	17.66	78.04	84.30	68.06	9.78	65.81
54 PFS	PFS Comp	Copper Collector	75	-	85	Natural	3.23	89.68	5.04	8.46	20.86	16.99	74.20	83.00	64.51	6.87	51.25
79	PFS Comp	Copper Collector	75	150	90	Natural	3.27	88.52	5.43	7.72	26.02	19.32	75.85	86.43	64.41	8.48	61.92
80	PFS Comp	Copper Collector	106	150	90	Natural	3.40	81.04	4.67	6.82	23.84	12.90	68.86	80.81	58.42	7.98	48.58
81	PFS Comp	Copper Collector	150	150	90	Natural	3.56	78.51	4.58	6.23	21.23	15.36	70.26	81.60	62.19	7.55	58.33
88	PFS Comp	Copper Collector	75	-	85	Natural	2.63	89.46	6.03	7.20	23.23	17.68	63.92	77.16	48.53	6.13	43.62
89	PFS Comp	Copper Collector	75	10	185	pH ~ 11 in Rougher	1.85	113.4	9.11	4.61	20.87	16.95	54.08	82.83	22.23	3.85	30.53
90	PFS Comp	Copper Collector	75	10	185	pH ~11 in Cleaner	3.29	81.50	5.36	7.04	24.90	19.79	69.86	85.91	57.95	8.08	60.68
91	PFS Comp	Copper Collector	75	-	190	Natural	3.05	87.21	5.96	8.31	24.57	18.77	71.64	79.14	61.85	7.58	53.58

A series of optimisation tests (Tests 79, 88 and 91) were conducted on the remaining PFS Composite using the MF1 copper collector flowsheet. This testing found that increased depressant addition of 150 g/t in the roughers and 90 g/t in the cleaners, with the inclusion of a separate low-grade and scavenger cleaner circuit as presented in Figure 13.13, resulted in superior metallurgical response (Test 79). Test 54 represents the results of the initial copper collector testing conducted as part of the Platreef 2015 PFS using a flowsheet as presented in Figure 13.11 with the exclusion of the conditioning prior to flotation.

Tests 80 and 81 were conducted using the optimal conditions of Test 79, but at a coarser grind of 106 µm and 150 µm respectively. The coarser grind in these tests was found to result in reduced 3PE+Au recovery of 7.0% for the 106 µm grind and 5.6% for the 150 µm grind. The finer grind requires additional milling power, however the additional 3PE+Au recovery results in improved overall project economics as compared to a lower 3PE+Au recovery with reduced mill power consumption. This provided confirmation that the target of 80% passing 75 µm grind, as selected in the Platreef 2015 PFS, was optimal for the MF1 copper collector flowsheet.

Additional testing with lime addition to achieve a pH of ~11 in the roughers (Test 89) and cleaners (Test 90) with increased depressant in the cleaners was conducted. These tests were deemed unsuccessful. Test 89 indicated that operation at pH 11 in the roughers, resulted in reduced sulphur recovery and improved 3PE+Au grade, however this reduced 3PE+Au recovery due to increased losses to the rougher tails. Test 90 indicated that operation at pH 11 in the cleaners, resulted in similar sulphur recovery as achieved in Test 79 but at reduced first pass 3PE+Au and Ni recovery due to increased losses to the cleaner tails.

The results of the MF1 circuit optimisation Test 79, using a targeted copper collector indicated that the optimised flowsheet (Figure 13.12) could achieve a similar performance to Test 69 using, the oxalic acid and thiourea flowsheet as presented in the Platreef 2015 PFS (Figure 13.11). This comparison of performance is presented in Figure 13.13.

Figure 13.13 PGE Grade Recovery Curve for the Optimised MF1 Copper Collector Flowsheet

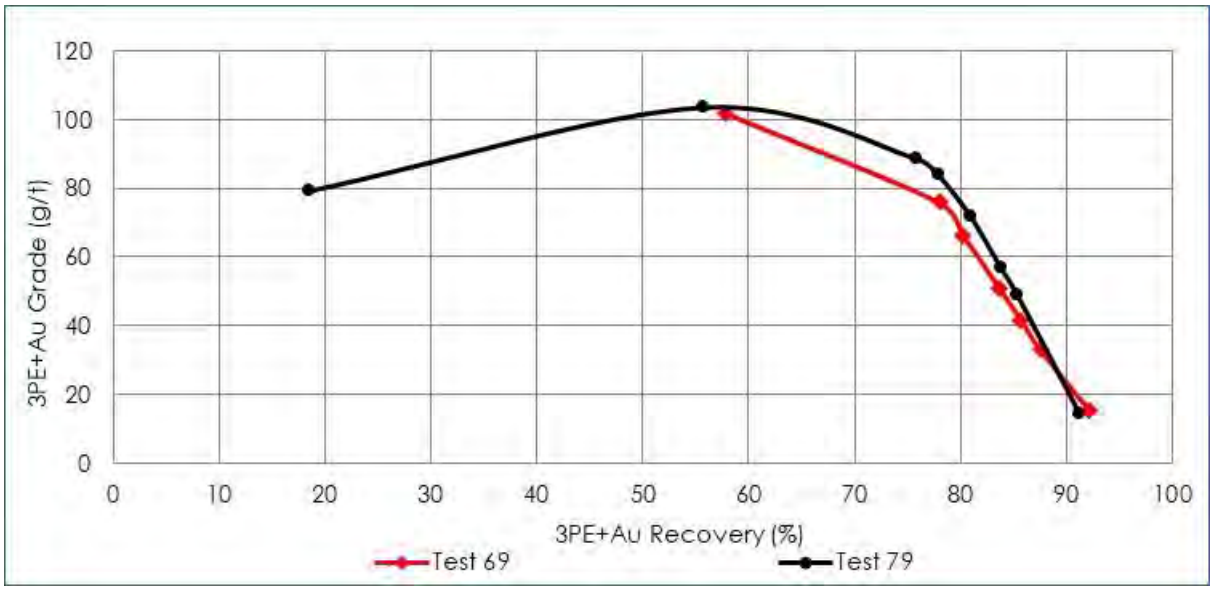


Figure by DRA, 2017.

Based on the confirmation of similar metallurgical performance for the MF1 flowsheet using the copper collector reagent suite, compared to the MF1 oxalic acid and thiourea flowsheet, it was decided to base the remainder of the Platreef 2017 FS testing on the MF1 copper collector flowsheet.

The MF1 copper collector flowsheet is preferable to the oxalic acid and thiourea reagent suite for the following reasons:

- The oxalic acid and thiourea reagent suite requires extended conditioning time for optimum effectiveness, resulting in scale-up difficulties and increased operating cost and technical risk. The conditioning time is not required for the copper collector flowsheet.
- The oxalic acid and thiourea reagent suite has not been used in the South African platinum industry, and use of the MF1 copper collector flowsheet mitigates the technical risk associated with the supply and use of this novel reagent suite.
- The MF1 circuit, using a reagent suite containing a targeted copper collector, has demonstrated in test work to have the potential to achieve lower first pass sulphur recovery to final concentrate. This may result in a more marketable concentrate that is likely to attract improved third party treatment terms.
- The copper collector reagent suite, during laboratory scale test work, was noted to result in more stable cleaner circuit froth that allows for more stable operation and improved mass pull control.

13.6.2.2 Sulphur Optimisation Testing

Sulphur optimisation testing was conducted on Platreef 2017 FS Composite 1 (FS Composite) (1.2% sulphur) and Composite 2 (1.0% sulphur), to evaluate the effect of sulphur feed grade, cleaner solids concentration and depressant addition on sulphur recovery to final concentrate.

Reducing sulphur (pyrrhotite) recovery to final concentrate would result in reduced concentrate mass pull, and lower concentrate sulphur content, to produce a more marketable concentrate product that is likely to attract improved third party treatment terms.

A summary of the flowsheets used in this testing is presented in Figure 13.12 (refer to Section 13.6.2.1). The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.11.

Table 13.11 Summary Results of Sulphur Optimisation Testing

Test No.	Sample	Rougher Solids %(w/w)	Scav. Cleaner Solids %(w/w)	Depressant		Mass	Grade					Recovery				
				Rougher	Cleaner	(%)	3PE+Au	Cu	Ni	Fe	S	3PE+Au	Cu	Ni	Fe	S
				(g/t)	(g/t)		(g/t)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
82	Composite 1	35	As Floated	150	90	3.66	61.04	3.97	6.88	23.68	19.57	68.47	77.58	63.21	9.04	59.70
82 Rpt	Composite 1	35	As Floated	150	90	3.00	71.47	6.03	7.69	22.24	15.14	63.54	83.55	56.06	6.92	37.90
84	Composite 1	35	As Floated	150	170	2.91	84.14	6.44	7.83	20.08	16.53	74.34	85.60	59.34	6.18	42.83
93	Composite 1	30	10	150	179	3.27	74.15	5.40	7.73	28.50	22.44	73.82	81.66	61.08	9.59	57.98
94	Composite 1	30	10	150	170	3.36	72.58	5.13	7.02	24.45	17.35	75.11	84.82	59.93	8.46	47.85
83	Composite 2	35	As Floated	150	90	3.69	79.60	5.28	5.87	19.79	17.24	74.49	84.92	55.61	7.59	58.56
85	Composite 2	35	As Floated	150	170	2.93	97.21	6.29	8.59	24.53	14.84	73.03	89.74	62.72	7.48	42.46

A series of optimisation tests (Tests 82, 84, 92, 93 and 94) were conducted on Composite 1 (1.2% sulphur) using the MF1 copper collector flowsheet. These tests indicated that increasing cleaner depressant addition from 90 g/t (Test 82) to 170 g/t (Test 84) resulted in a 16% reduction in first pass sulphur recovery to final concentrate. A repeat of Test 82, where a lower rougher and cleaner mass pull was targeted by adjusting flotation scrape rates, also resulted in reduced sulphur recovery. Test 82 (Rpt) resulted in significantly lower 3PE+Au recoveries, where the first pass 3PE+Au recovery to final concentrate was 10.8% lower than for Test 84, however this was as a result of increased rougher tails losses for Test 82 which would not be influenced by the cleaner depressant regime.

Bench-scale tests, conducted at a lower solids concentration of 30% (by mass) in the roughers and 10% (by mass) in the scavenger cleaner (Tests 93 and 94) did not result in improved 3PE+Au recovery or sulphur rejection relative to Test 84. Test 93, which considered the addition of a scavenger re-cleaner stage, was found to result in increased sulphur recovery.

Tests 83 and 85 were conducted on Composite 2, with a lower sulphur content of 1.0%. These tests confirmed the finding that increasing cleaner depressant addition from 90 g/t (Test 83) to 170 g/t (Test 85) resulted in a 16% reduction in first pass sulphur recovery to final concentrate.

Sulphur recovery in the sulphur optimisation testing is presented relative to 3PE+Au recovery in Figure 13.14.

Figure 13.14 Sulphur Recovery as Function of PGE Recovery in Sulphur Optimisation Testing

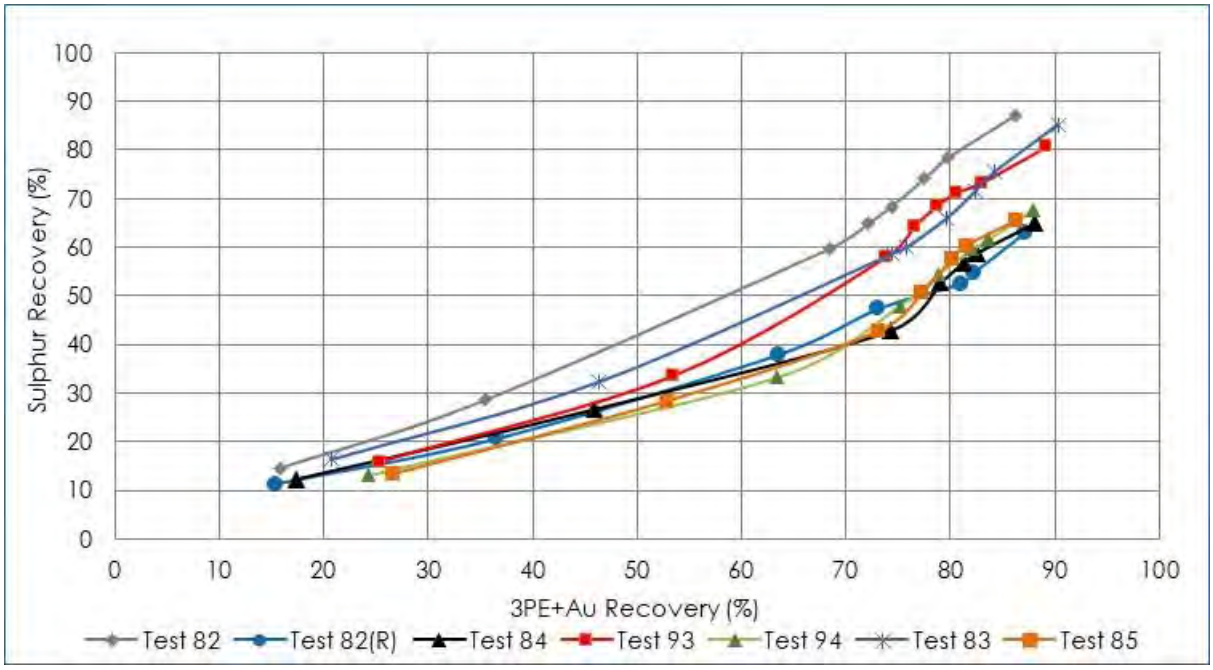


Figure by DRA, 2017.

As part of the sulphur optimisation testing, a locked cycle test was conducted on Composite 1 using the reagent regime with higher cleaner depressant addition as per open circuit Test 84. In this locked cycle test, depressant addition was adjusted based on the visual performance of the float, and a cleaner depressant addition rate of 130 g/t was achieved. This test achieved an overall 3PE+Au recovery of 81% and a sulphur recovery of 60%.

This lower than expected 3PE+Au recovery was attributed to high depressant addition in the cleaners, and further grade optimisation testing was initiated. This testing focused on optimising rougher depressant addition and evaluating the benefit of additional cleaning stages.

13.6.2.3 Concentrate Grade Optimisation

Concentrate grade optimisation test work was conducted on the Platreef 2017 FS Composite 3 (3.46 g/t 3PE+Au), with the aim of increasing final concentrate 3PE+Au grade. This test work evaluated the benefit of additional cleaning stages and optimisation of rougher depressant addition. After completion of the sulphur optimisation testing, Composite 1 was depleted. Further grade optimisation testing was thus conducted on Composite 3 with a similar 3PE+Au grade as Composite 1, however Composite 3 was selected to specifically target a lower sulphur grade in feed of 0.8%, that at the time of testing represented the expected sulphur grade in the early years of mining.

A summary of the flowsheets used in this testing is presented in Figure 13.15 and Figure 13.16. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.12.

Figure 13.15 Copper Collector Reagent Flowsheet with HG Re-Cleaner

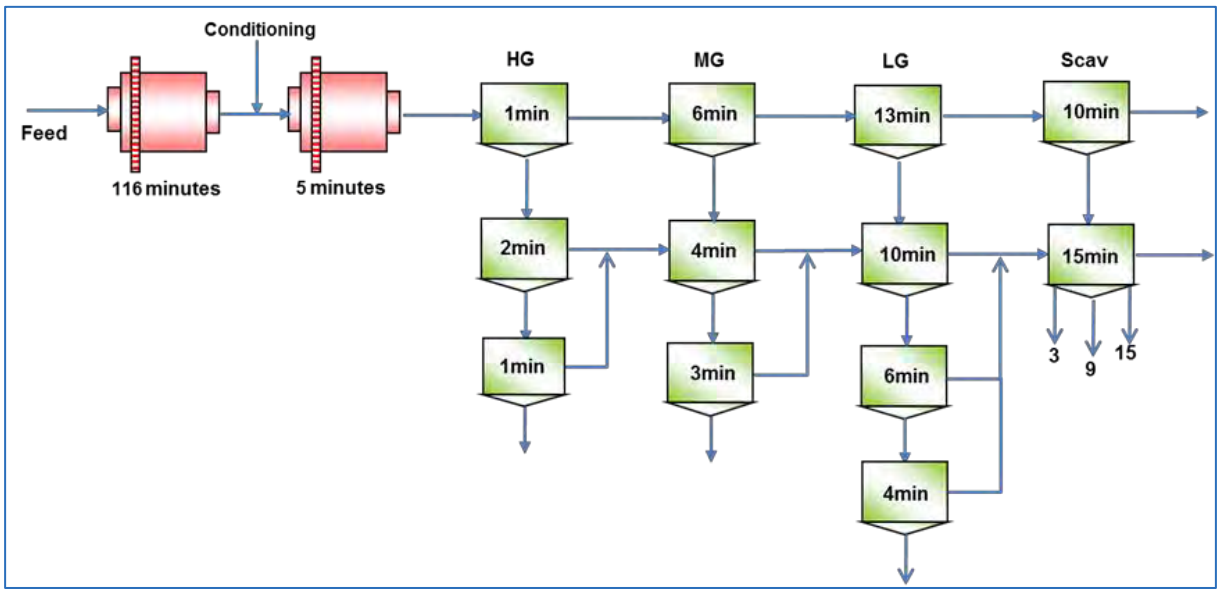


Figure by DRA, 2017.

Figure 13.16 Copper Collector Reagent Flowsheet with HG Re-Cleaner and MG/LG Column

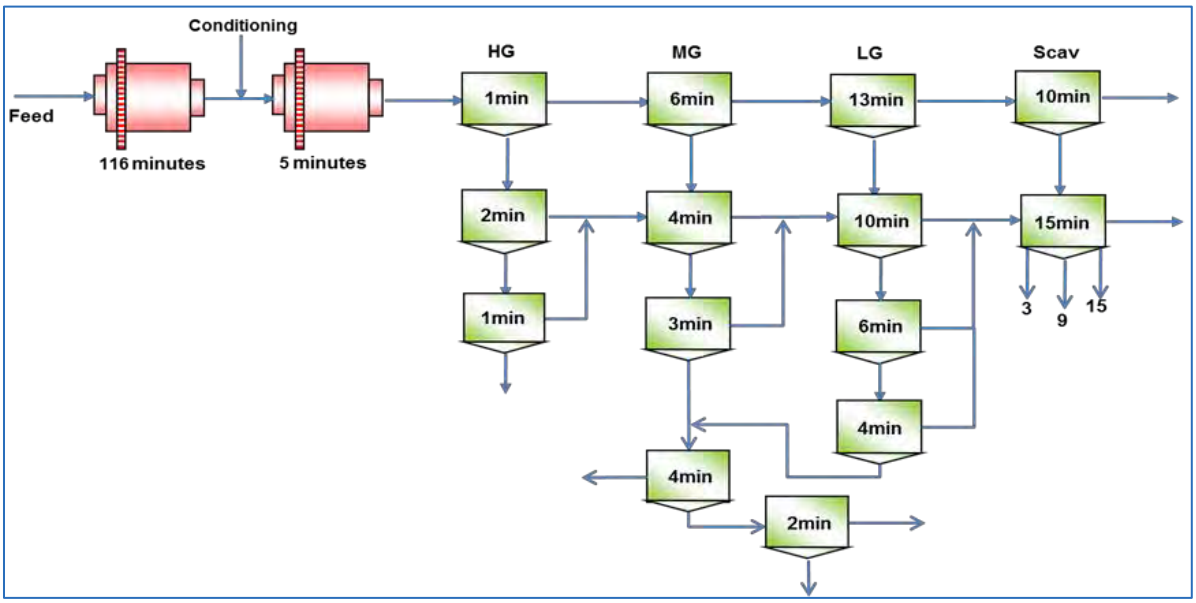


Figure by DRA, 2017.

Table 13.12 Summary Results of Concentrate Grade Optimisation Testing

Test No.	Sample	Depressant			Mass	Grade					Recovery				
		Rougher	Cleaner	Type	(%)	3PE+Au	Cu	Ni	Fe	S	3PE+Au	Cu	Ni	Fe	S
		(g/t)	(g/t)			(g/t)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
95	Composite 3	196	59	CMC	2.42	97.50	6.99	9.26	24.48	19.49	73.70	85.25	63.22	6.03	55.72
96	Composite 3	150	74	CMC	2.29	113.31	7.04	9.43	23.62	20.44	74.25	84.33	60.67	5.65	54.72
97	Composite 3	146	59	CMC	3.15	81.30	4.59	6.74	20.25	15.86	76.66	83.00	64.50	6.46	60.30
102	Composite 3	150	74	CMC	2.79	90.38	5.70	8.01	21.15	16.70	72.86	87.40	59.56	6.02	54.22
103	Composite 3	150	89	CMC	1.93	131.45	7.52	10.88	24.59	20.04	73.48	85.31	55.78	4.90	47.26
104	Composite 3	150	84	HC	3.88	62.45	4.07	4.12	13.95	8.65	66.37	86.08	46.72	5.39	41.93

This testing indicated that final concentrate grade could be improved with the addition of a high-grade (HG) concentrate re-cleaner stage (Tests 95 and 96), compared to testing with a single HG cleaner stage (Test 102). In Test 97, dilute two-stage cleaning of the combined medium-grade (MG) and low-grade (LG) concentrate was conducted in order to provide an indication of the expected performance of a column cell. This dilute two-stage cleaning, was found to allow for an increase in the final concentrate grade achieved in the combined MG/LG circuit. A column cell was thus included in the Platreef 2017 FS flowsheet, all further testing was conducted without the addition of this dilute two-stage cleaning, column simulation. The bench-scale column cell simulation is regarded as indicative only, and the true benefit associated with the inclusion of the column cell can only be accurately determined from pilot-scale testing.

Test 104, using a hemicellulose (HC) depressant, achieved lower PGE grades and recovery when compared to the results obtained with a carboxymethyl cellulose (CMC) depressant in the other tests.

Test 103, which considered the inclusion of a HG concentrate re-cleaner, CMC depressant dosage of 240 g/t and a target final mass pull of <2.0%, resulted in the optimal PGE upgrade profile, however recoveries were lower than for Test 96, within the target grade range. This is presented in Figure 13.17.

Figure 13.17 Concentrate Grade Optimisation Test Results

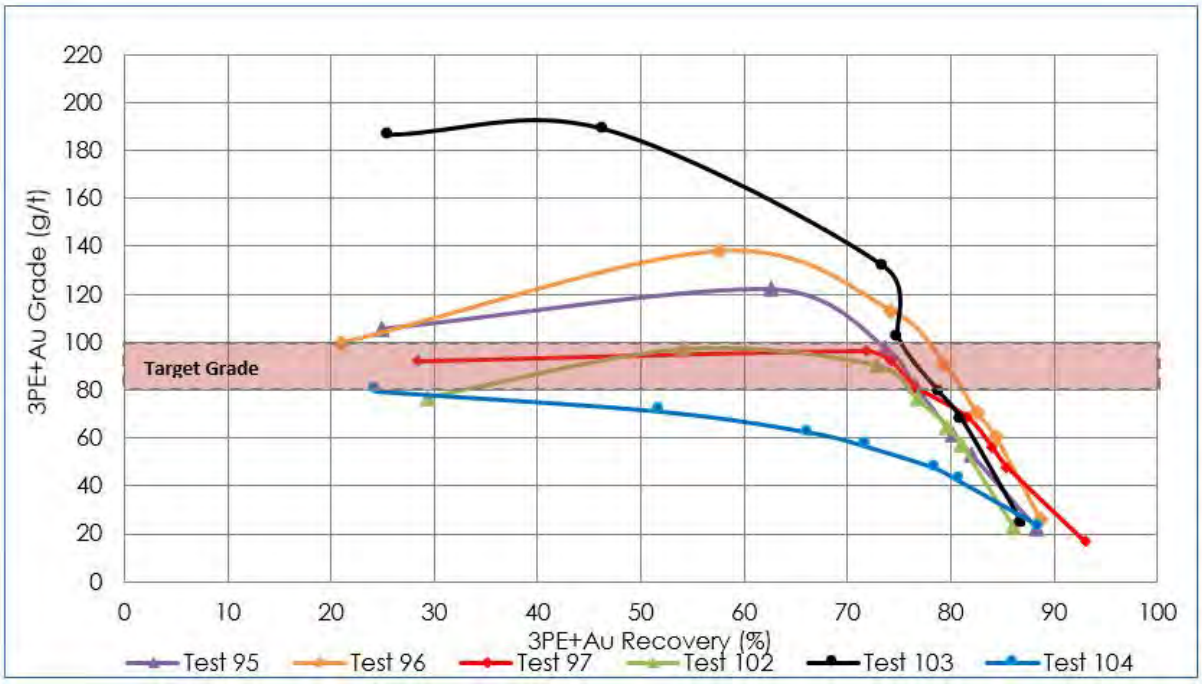


Figure by DRA, 2017.

13.6.2.4 Confirmation of Optimised Conditions for a High-Grade Sample

Further optimisation test work was conducted on a high-grade composite (5.39 g/t 3PE+Au), which represented at the time of testing, the expected PGE feed grade during the first five years of production. The aim of this phase of testing was to evaluate the effect of solids concentration, depressant addition and to evaluate any further opportunity for concentrate recovery and grade improvement.

The effect of rougher feed solids concertation had previously been evaluated on Composite 1, where no improvement in 3PE+Au recovery or concentrate grade was demonstrated, however this was possibly due to the increased cleaner depressant addition in this testing. Mintek also indicated that at the time of the initial testing, control of the rougher operating conditions in a 5L batch cell, at the lower solids concentration, had not been optimised.

A summary of the flowsheet used in this testing is presented in Figure 13.18. The test parameters and recovery and grades achieved for the final combined concentrate in open circuit testing are presented in Table 13.13.

Figure 13.18 Flowsheet to Confirm Optimal Conditions for a High-Grade Sample

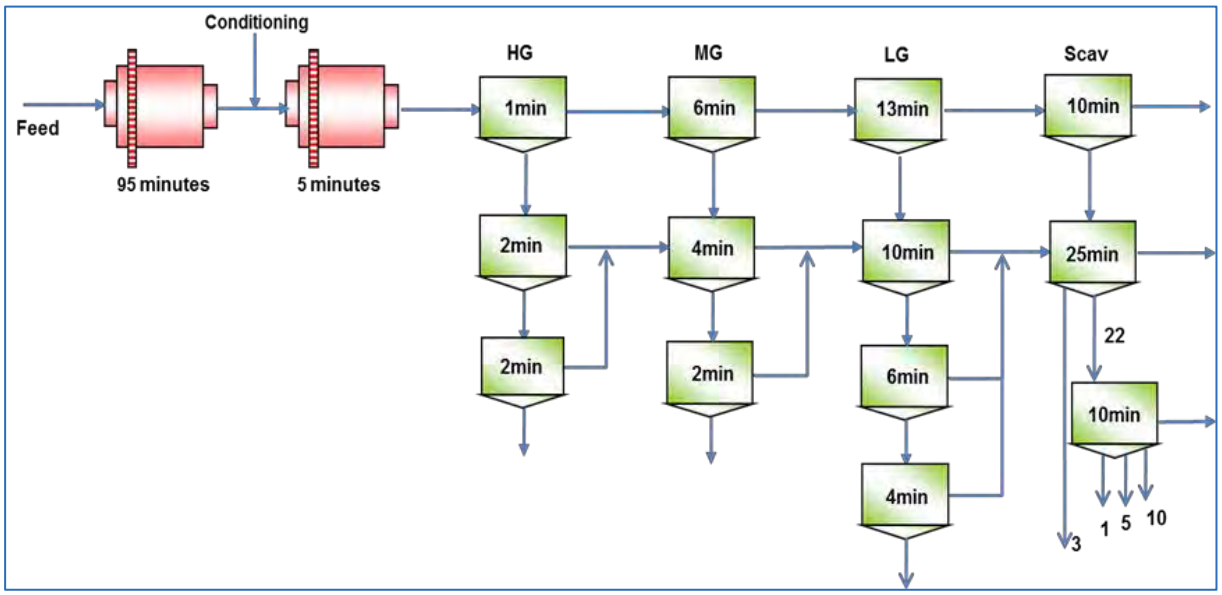


Figure by DRA, 2017.

Table 13.13 Summary Results for Optimisation Testing on a High-Grade Sample

Test No.	Sample	Rougher Solids %(w/w)	Depressant		Scavenger Cleaner Configuration	Mass	Grade					Recovery				
			Rougher	Cleaner		(%)	3PE+Au	Cu	Ni	Fe	S	3PE+Au	Cu	Ni	Fe	S
			(g/t)	(g/t)			(g/t)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
106	Composite 5	35	176	106	15min Cleaner	3.11	120.2	5.91	7.82	28.72	20.24	73.43	85.21	57.10	8.79	65.64
107	Composite 5	35	150	190	15min Cleaner	3.15	123.6	5.40	8.03	29.81	20.00	74.01	81.98	59.52	8.99	66.65
111	Composite 5	28	150	90	15min Cleaner	3.43	111.2	4.83	8.38	26.74	20.07	75.90	80.62	61.93	9.46	70.73
112	Composite 5	28	250	90	15min Cleaner	3.52	113.6	-	-	-	17.99	77.11	-	-	-	65.73
113	Composite 5	28	250	90	25min Cleaner	3.55	112.2	5.07	7.39	26.37	18.65	80.16	85.15	59.53	9.30	68.26
114	Composite 5	28	250	96	25min Cleaner 10min Re-Cleaner	3.68	121.1	4.86	7.30	25.16	18.60	79.52	85.21	62.16	8.81	68.20

*There was insufficient high-grade re-cleaner concentrate mass to allow for Cu, Ni and Fe assays in Test 112.

The test work indicated that final concentrate PGE grades of 110 g/t – 120 g/t could be achieved at a PGE head grade of 5.4 g/t.

Test 111 conducted at 28% solids (by mass) showed evidence of minor improvement in the grade recovery profile within the target operating range as compared to Test 106 with a similar depressant addition regime but conducted at 35% solids (by mass) as presented in Figure 13.19.

Test 113 was conducted with a 25 minute residence time in the scavenger cleaner, this test indicated that the first concentrate produced after 3 minutes in the scavenger cleaner was of comparable grade (~20 g/t) to the final low grade concentrate (~23 g/t). It was subsequently decided that this concentrate should be routed to the final low grade re-re-cleaner feed in locked cycle testing.

A further test (Test 114) was then conducted to evaluate if the scavenger concentrate produced after the first 3 minutes could be upgraded to the low grade concentrate grade with the inclusion of a re-cleaner stage. This proved unsuccessful, and a grade of ~11 g/t was achieved.

Figure 13.19 Evaluation of the Effect of Rougher Feed Solids Concentration

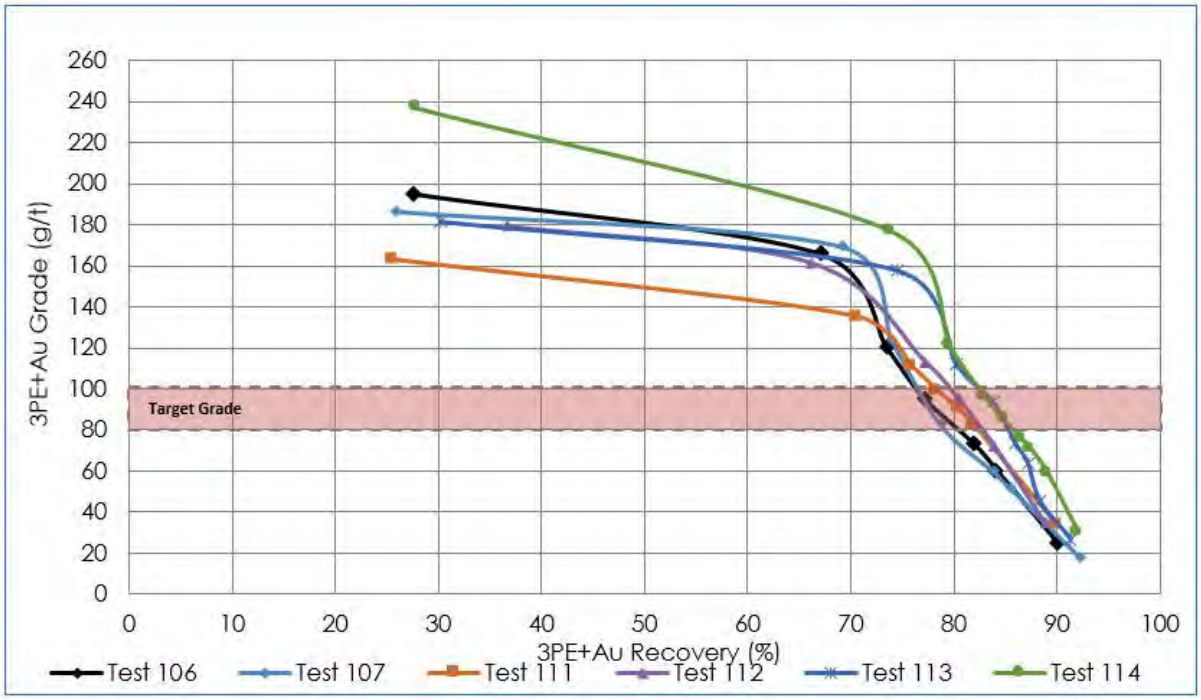


Figure by DRA, 2017.

The test conditions and flowsheet as employed in Test 113 was selected for the open circuit variability testing and further locked cycle testing.

13.7 Flotation Open Circuit Variability Test Work

As part of the Platreef 2017 FS test work programme, an extensive bench-scale variability test programme was conducted. As part of this programme, 80 open circuit tests were conducted on samples representing T1, T2U, T2L, CZ and various ore blends as follows:

- Domain point samples representing the individual geometallurgical units from individual drill core samples. For example T1 from drill core hole UMT104D1.
- Point sample blend composites representing a blend of geometallurgical units from individual drill core samples. For example a blend of T1, T2U, T2L and CZ from drill core hole UMT166D1.
- Blend composites representing blends of geometallurgical units from various drill core samples. For example a blend of 14%T1, 51%T2U, 30%T2L and 5% CZ using sample selected from all the drill core holes listed in Table 13.3.

The open circuit variability flowsheet is presented in Figure 13.20.

Figure 13.20 Optimised Platreef 2017 FS Flowsheet for Open Circuit Variability Testing

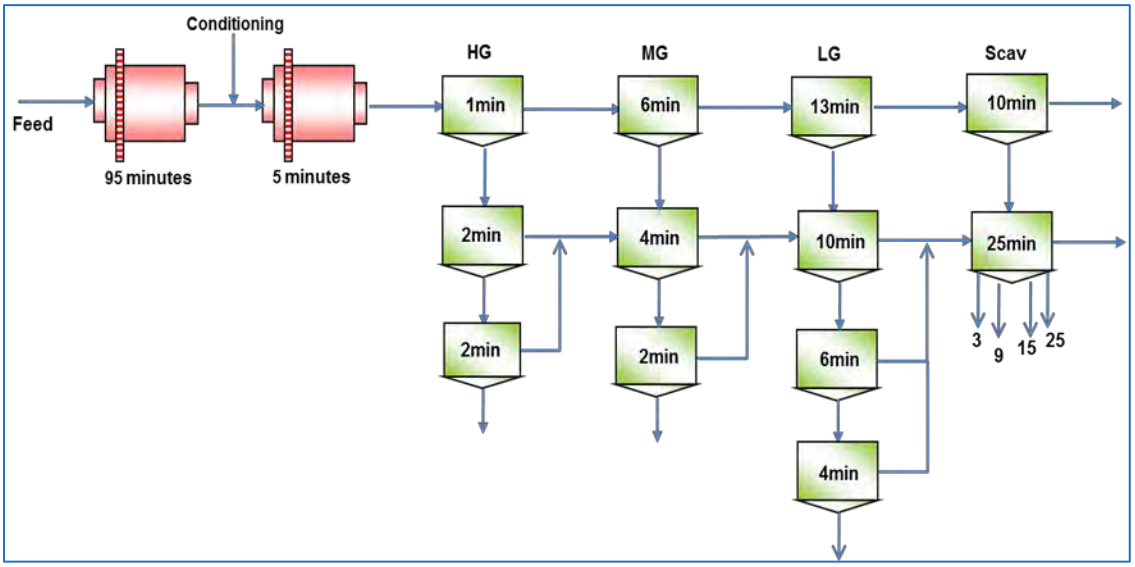


Figure by DRA, 2017.

13.7.1 Summary of Open Circuit Variability Results

13.7.2 3PE+Au Recovery and Grade Relationships

Open circuit flotation test data was used to derive an estimate for the expected locked cycle test recovery and concentrate grade as follows:

- Locked cycle test mass pull was, on average observed to be approximately 20% higher than the mass pull achieved in open circuit testing

- The expected 3PE+Au and sulphur recovery was based on the recovery to final concentrate and the average of the recovery to scavenger cleaner concentrate 2 and 3
- The expected copper and nickel recovery was based on the reported recovery to final concentrate and the recovery to scavenger cleaner concentrate 4
- The expected final concentrate grades where then calculated based on the test mass pull and recovery data

This method was derived based on comparison of open circuit and locked cycle mass pull and recovery data for testing conducted on Platreef samples. This method provides a reasonable approximation of the achievable recovery and concentrate grade based on open circuit test results, however it does not provide a definitive representation of locked cycle recovery. This method as applied to all variability test data, is considered adequate for the purpose of comparing the open circuit test results.

The 3PE+Au head grade-recovery curves derived from the variability testing conducted on domain point samples and blend composites are presented in Figure 13.21 and Figure 13.22.

Figure 13.21 Variability Testing Head Grade Recovery Relationships for Domain Point Samples

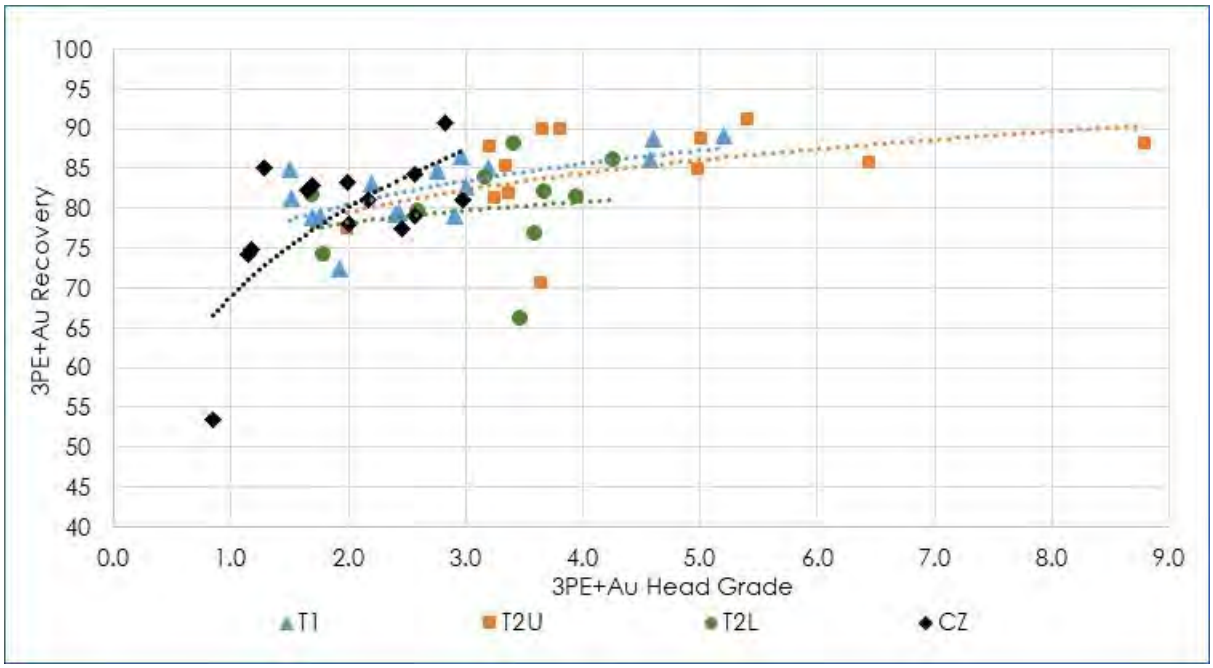


Figure by DRA, 2017.

Figure 13.22 Variability Testing Head Grade - Concentrate Grade Relationships for Blend Composites

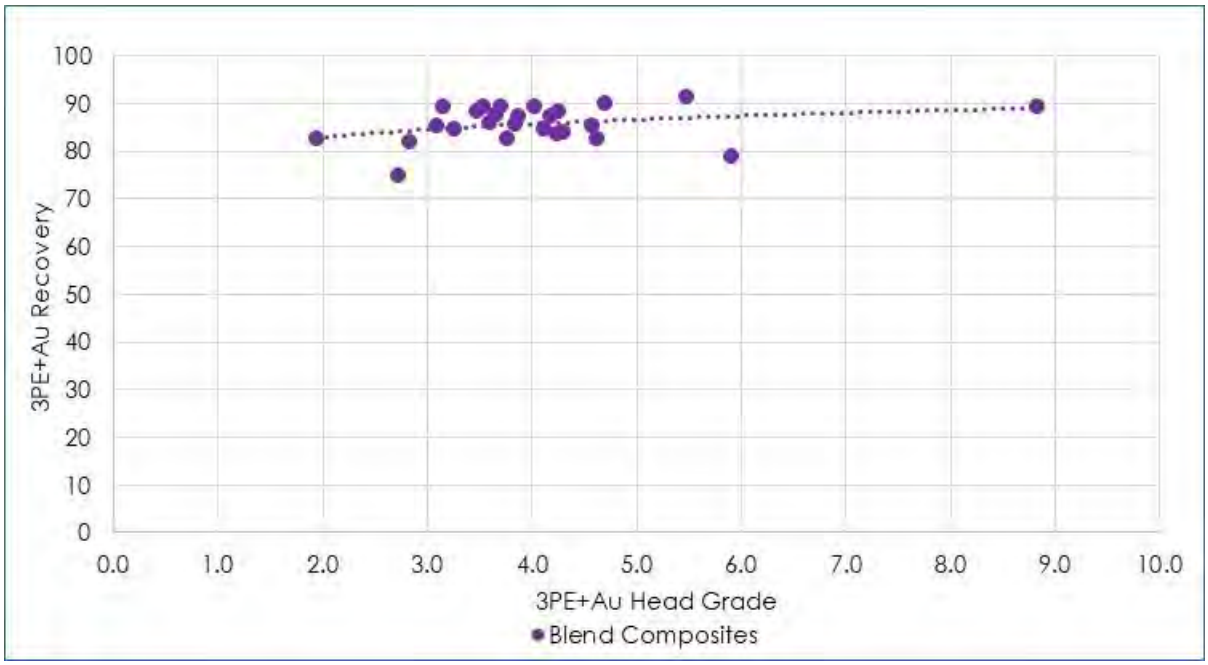


Figure by DRA, 2017.

There was a positive correlation between head grade and recovery with an increase in 3PE+Au recovery as 3PE+Au head grade increased for all ore types tested. The final concentrate 3PE+Au concentrate grade also increased with increasing head grade as presented in Figure 13.23.

Figure 13.23 Variability Testing Concentrate Grade as a Function of Head Grade

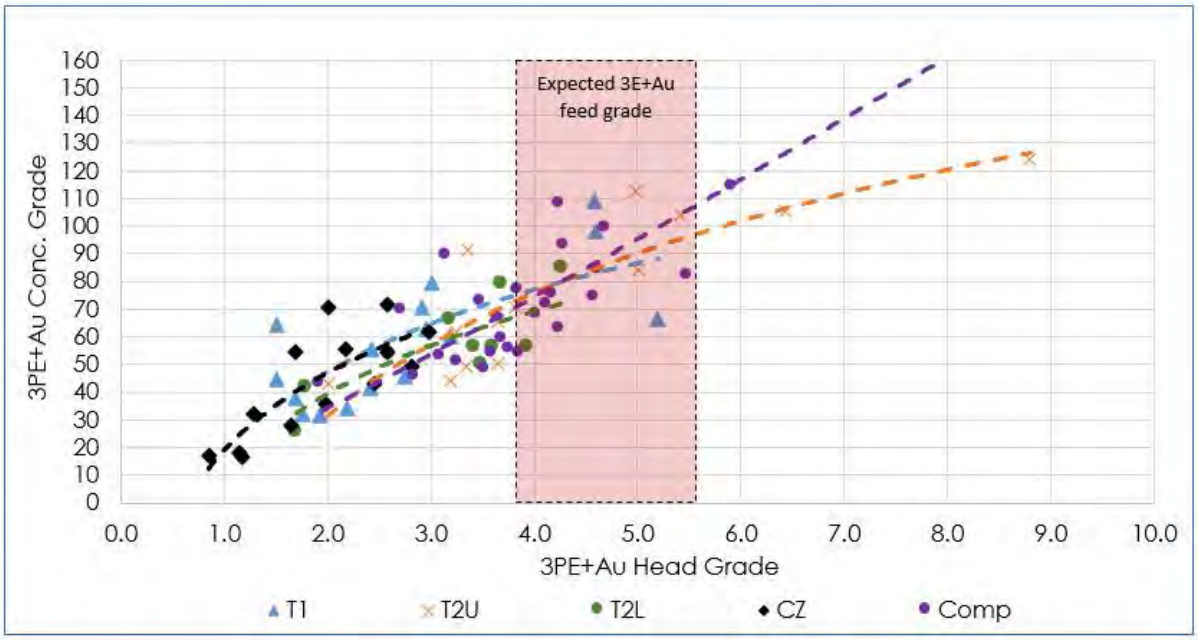


Figure by DRA, 2017.

Further to this, 3PE+Au recovery was plotted relative to head grade and grouped according to the final concentrate grade estimate as presented in Figure 13.24. The test data showed only minor variation between the performance for the individual domain point samples and the blend composites, rather the recovery and achievable concentrate grade was primarily dependant on the 3PE+Au head grade.

The achievable concentrate grade was shown to have a dependence on 3PE+Au head grade for all ore types tested. Based on the variability testing, the minimum 3PE+Au concentrate target grade of 85 g/t can be achieved for head grades in the range 3.2 g/t - 4.6 g/t. 3PE+Au concentrate grades in excess of 100 g/t can be achieved for head grades in the range 4.6 g/t - 8.8 g/t.

Figure 13.24 Variability Testing Recovery as a Function of Head Grade

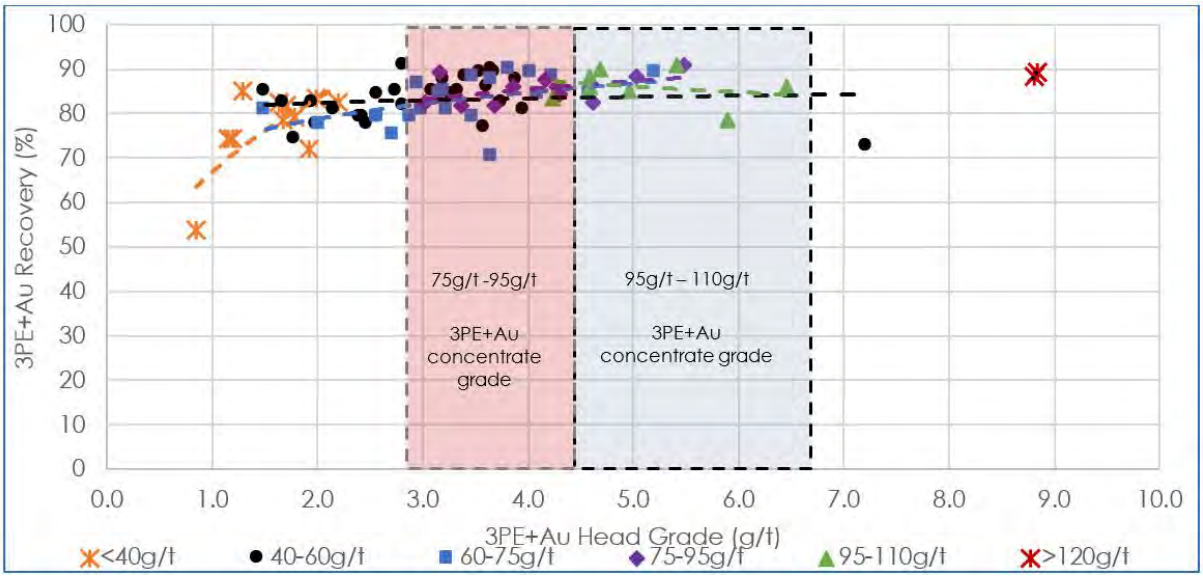


Figure by DRA, 2017.

13.7.2.1 Copper and Nickel Recovery and Grade Relationships

The nickel and copper head grade-recovery curves derived from the variability testing conducted on domain point samples and blend composites are presented in Figure 13.25.

Figure 13.25 Variability Testing Recovery as a Function of Head Grade

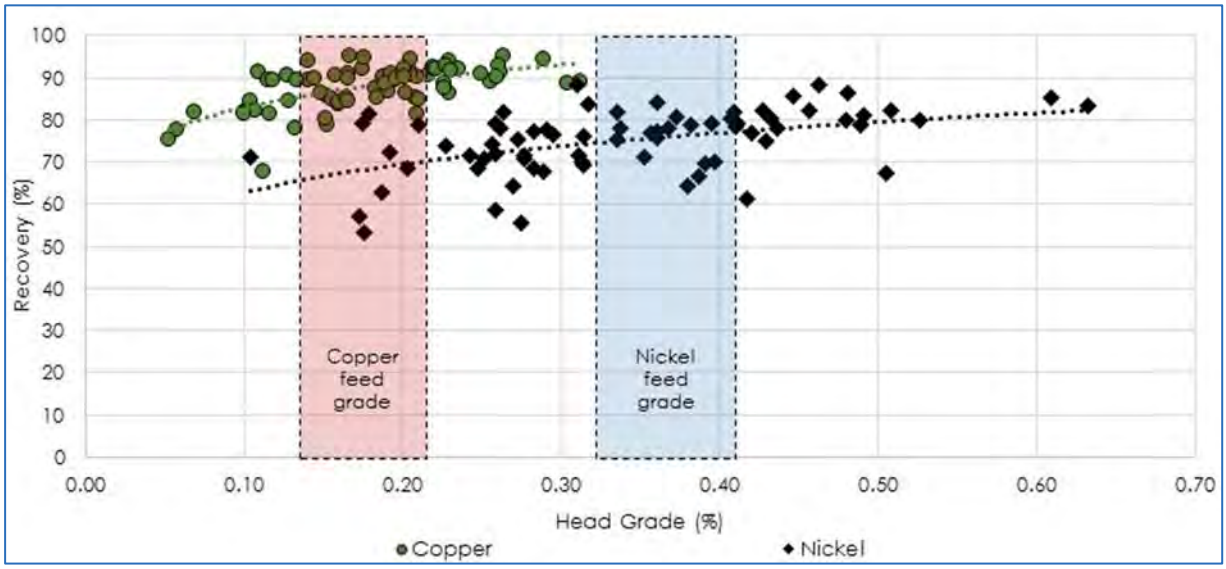


Figure by DRA, 2017.

13.7.2.2 Sulphur Recovery

A plot of the sulphur recovery as a function of 3PE+Au recovery (Figure 13.26) for all test data, indicated increased sulphur recovery as 3PE+Au recovery increased, albeit with significant scatter in the data.

Figure 13.26 Variability Testing Sulphur Recovery as a Function of 3PE+Au Recovery

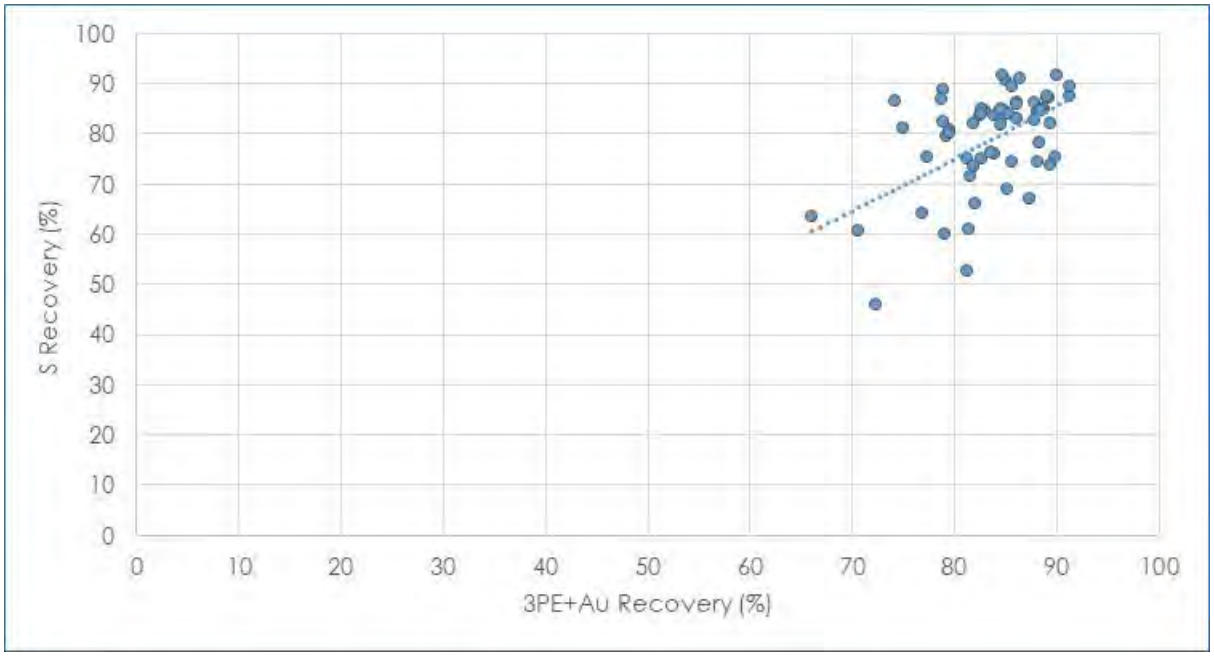


Figure by DRA, 2017.

13.8 Flotation Locked Cycle Test Work

13.8.1 Summary of Locked Cycle Test Flowsheets

A summary of the locked cycle test flowsheets used during the initial development and optimisation test work phase are presented in Figure 13.27 and Figure 13.28. The optimised Platreef 2017 FS locked cycle test flowsheet is presented in Figure 13.29.

Figure 13.27 Development Testing Locked Cycle Test Flowsheet 1 (Flowsheet 1)

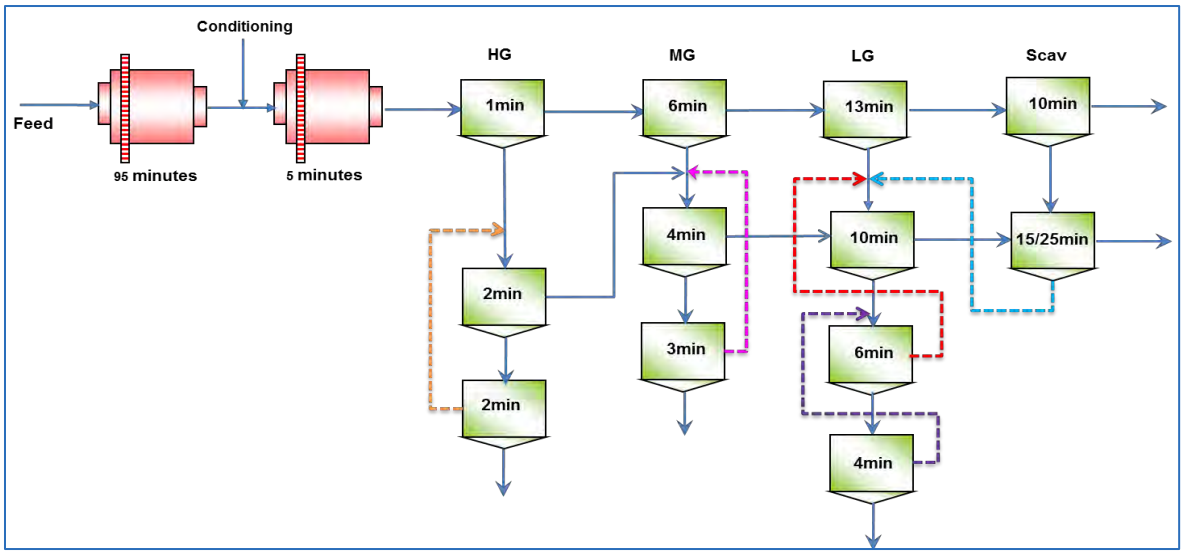


Figure by DRA, 2017.

Figure 13.28 Development Testing Locked Cycle Test Flowsheet 2 (Flowsheet 2)

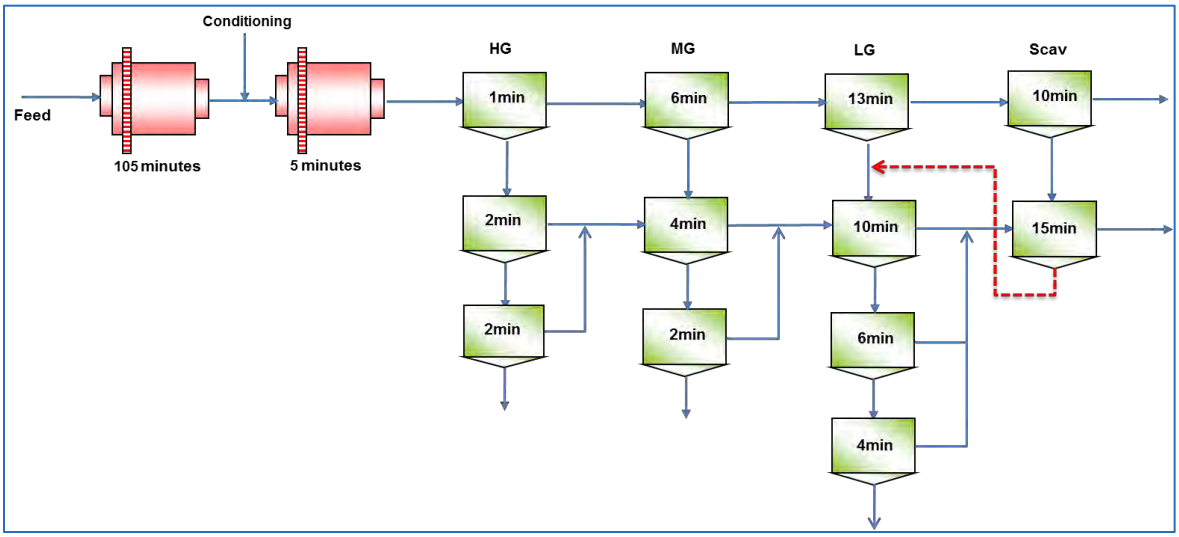


Figure by DRA, 2017.

Figure 13.29 Optimised Platreef 2017 FS Locked Cycle Test Flowsheet (Flowsheet 3)

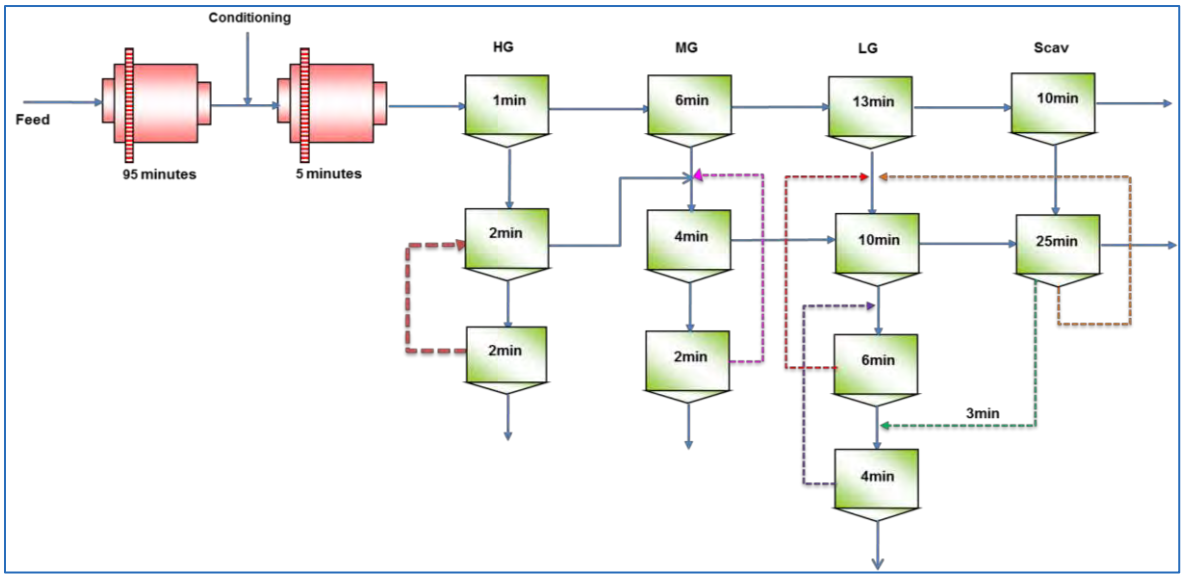
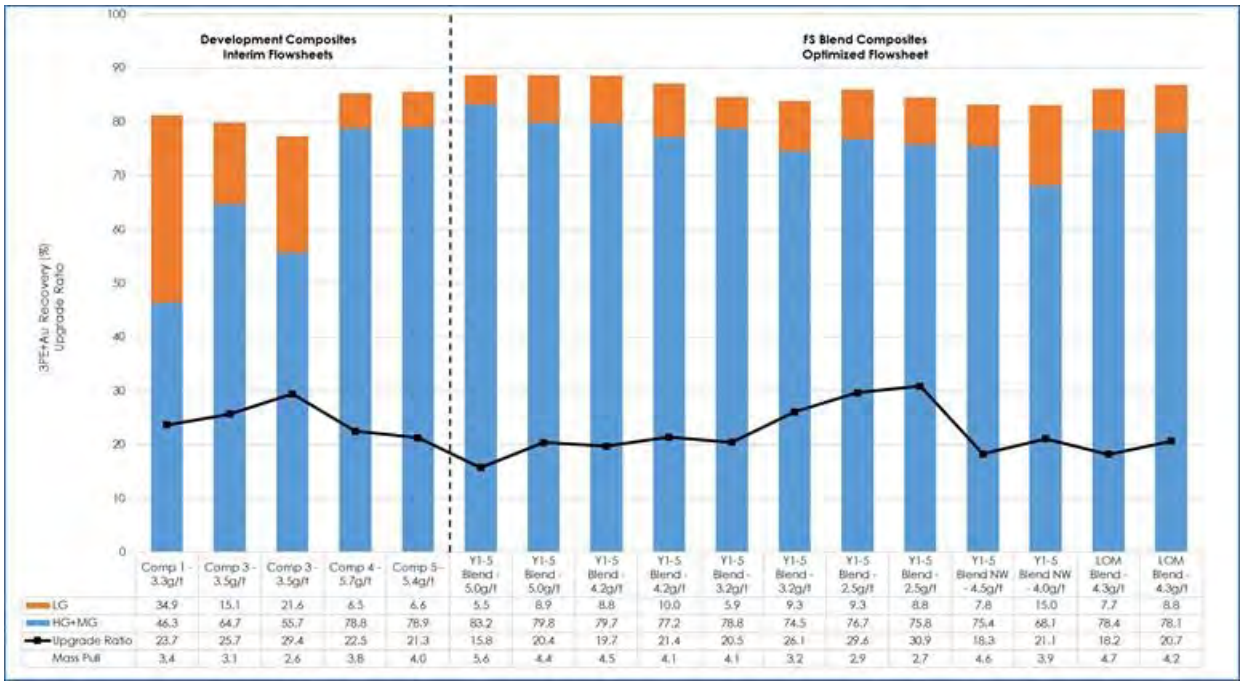


Figure by DRA, 2017.

13.8.2 Summary of Locked Cycle Test Results

The head grade and domain blends of the locked cycle test samples are presented in Table 13.14. A summary of the locked cycle test results is presented in Figure 13.30, while the grade and recovery achieved for the final combined concentrate (high grade, medium grade and low grade) is presented in Table 13.15.

Figure 13.30 Summary of Locked Cycle Test Results



	Comp 1 - 3.3g/t	Comp 3 - 3.5g/t	Comp 3 - 3.5g/t	Comp 4 - 5.7g/t	Comp 5 - 5.4g/t	Y1-5 Blend - 5.0g/t	Y1-5 Blend - 5.0g/t	Y1-5 Blend - 4.2g/t	Y1-5 Blend - 4.2g/t	Y1-5 Blend - 3.2g/t	Y1-5 Blend - 3.2g/t	Y1-5 Blend - 2.5g/t	Y1-5 Blend - 2.5g/t	Y1-5 Blend NW - 4.5g/t	Y1-5 Blend NW - 4.0g/t	LOM Blend - 4.3g/t	LOM Blend - 4.3g/t
LG	34.9	15.1	21.6	6.5	6.6	5.5	8.9	8.8	10.0	5.9	9.3	9.3	8.8	7.8	15.0	7.7	8.8
HG+MG	46.3	64.7	55.7	78.8	78.9	83.2	79.8	79.7	77.2	78.8	74.5	76.7	75.8	75.4	68.1	78.4	78.1
Upgrade Ratio	23.7	25.7	29.4	22.5	21.3	15.8	20.4	19.7	21.4	20.5	26.1	29.6	30.9	18.3	21.1	18.2	20.7
Mass Pull	3.4	3.1	2.6	3.8	4.0	5.6	4.4	4.5	4.1	4.1	3.2	2.9	2.7	4.6	3.9	4.7	4.2

Figure by DRA, 2017.

Table 13.14 Locked Cycle Test Measured Sample Head Assays

Test ID	Approximate Ore Blend	Grade									
		3PE+Au	Pt	Pd	Rh	Au	Cu	Ni	Total S	MgO	Fe
		(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	(%)	(%)	(%)
Development test work domain blends											
Composite 1	12%T1, 32%T2U, 50%T2L, 6%CZ	3.25	1.34	1.62	0.06	0.23	0.20	0.39	1.28	23.02	9.15
Composite 3		3.48	1.53	1.61	0.10	0.24	0.18	0.36	0.81	27.56	10.15
Composite 4		5.71	2.49	2.70	0.19	0.33	0.21	0.46	0.97	27.06	10.40
Composite 5		5.39	2.24	2.70	0.15	0.30	0.21	0.46	0.97	25.07	9.77
FS domain blends											
Year 1-5 Blend 1	14%T1, 51%T2U, 30%T2L, 5%CZ	5.05	2.36	2.17	0.13	0.39	0.22	0.40	1.02	24.24	9.53
Year 1-5 Blend 2		4.20	1.91	1.90	0.11	0.28	0.20	0.38	1.00	25.01	9.62
Year 1-5 Blend 3		3.21	1.39	1.42	0.10	0.30	0.20	0.37	1.19	25.15	9.72
Year 1-5 Blend 4		2.49	1.05	1.16	0.09	0.20	0.19	0.36	1.09	24.79	9.68
Year 1-5 Blend 5	10%T1, 52%T2U, 33%T2L, 5%CZ	4.56	1.93	2.20	0.15	0.28	0.18	0.34	0.81	24.65	9.31
Year 1-5 Belnd 6		3.98	1.69	1.89	0.12	0.28	0.16	0.30	0.80	22.85	9.48
LOM Blend	22%T1, 47%T2U, 26%T2L, 5% CZ	4.27	1.90	1.96	0.15	0.26	0.20	0.43	1.00	24.07	9.68

Table 13.15 Summary of Locked Cycle Test Results

Test ID	Flowsheet	Mass Pull Target	Depressant		Head Grade	Mass	Grade						Recovery					
			Rough	Clean	3PE+Au		3PE+Au	Cu	Ni	Total S	MgO	Fe	3PE+Au	Cu	Ni	Total S	MgO	Fe
			(g/t)	(g/t)	(g/t)	%	(g/t)	(%)	(%)	(%)	(g/t)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Development Locked Cycle Testing																		
Composite 1	1	Low	150	130	3.25	3.42	77.90	5.72	7.77	20.63	9.41	27.82	81.21	90.96	68.79	59.19	1.43	9.90
Composite 3	1	Low	150	90	3.48	3.10	80.43	5.05	8.06	17.07	12.67	22.12	79.82	88.92	70.85	65.22	1.54	7.00
Composite 3	2	Low	150	90	3.48	2.63	90.03	5.55	7.51	19.59	13.07	23.52	77.29	84.84	53.72	67.72	1.31	6.25
Composite 4	1	Low	150	90	5.71	3.79	123.4	4.73	7.95	18.42	11.81	30.56	85.34	90.23	71.00	82.00	1.65	11.15
Composite 5	1	Low	250	90	5.39	4.02	110.3	4.19	7.27	18.93	12.95	25.80	85.52	87.92	71.24	81.06	1.99	10.09
Optimised FS Flowsheet Locked Cycle Testing																		
Year 1-5 Blend 1	3	Low	250	90	5.05	4.36	93.37	4.55	7.68	18.20	9.94	25.81	88.72	93.51	83.29	82.43	1.87	11.47
Year 1-5 Blend 1	3	High	-	90	5.05	5.63	76.45	3.65	5.64	15.45	15.69	20.33	88.66	92.24	81.87	82.63	3.76	11.51
Year 1-5 Blend 2	3	Low	250	90	4.20	4.07	86.50	4.26	7.05	16.61	15.70	19.71	87.16	89.57	74.10	69.15	2.45	8.23
Year 1-5 Blend 2	3	High	125	90	4.20	4.49	82.70	4.07	7.14	15.80	13.91	22.15	88.55	95.36	80.15	74.71	2.61	10.73
Year 1-5 Blend 3	3	Low	250	90	3.21	3.21	77.31	5.85	8.82	20.61	11.88	25.77	83.84	90.23	73.95	66.22	1.53	8.26
Year 1-5 Blend 3	3	High	125	90	3.21	4.14	61.69	4.40	6.96	19.53	11.43	25.88	84.70	89.75	74.47	78.95	1.87	10.57
Year 1-5 Blend 4	3	Low	250	90	2.49	2.74	73.62	6.07	8.64	21.72	10.03	26.38	84.57	88.48	70.66	70.64	1.14	7.49
Year 1-5 Blend 4	3	High	125	90	2.49	2.90	72.96	5.89	8.95	21.89	8.83	27.48	85.98	93.70	74.09	71.86	1.02	8.08
Year 1-5 Blend 5	3	Low	250	90	4.56	4.55	81.96	3.36	4.76	12.74	18.06	17.45	83.25	87.22	66.22	73.33	3.22	8.71
Year 1-5 Blend 6	3	Low	250	90	3.98	3.94	84.97	3.76	5.70	13.62	14.34	20.44	83.13	86.22	72.30	68.87	2.49	8.32
LOM Blend	3	Low	250	90	4.27	4.20	86.12	4.88	8.40	19.44	11.53	24.78	86.88	93.35	80.59	79.54	2.04	10.56
LOM Blend	3	High	125	90	4.27	4.73	76.82	4.06	6.82	14.35	16.06	20.52	86.11	95.27	78.19	70.38	3.12	9.68

13.8.2.1 Locked Cycle Testing on Development Composites

Locked cycle testing on development composites representing the Platreef 2015 PFS mine blend ratio achieved recoveries in the range 77.3%-85.5% at a final concentrate grade of approximately 80 g/t-120 g/t.

The locked cycle test on development Composite 1 was conducted with a high cleaner depressant addition rate of 130 g/t with the objective of reducing sulphur recovery. In this test using flowsheet 1 (refer to Figure 13.27), it was found that this higher cleaner addition rate negatively impacted on recovery. In this test only 46% 3PE+Au recovery was achieved in the combined high grade and medium grade concentrate with further 35% 3PE+Au recovery via the low grade cleaner circuit resulting in a total overall recovery of 81%. The large portion of misplaced PGE's in the low grade cleaner circuit was attributed to the high depressant addition and cleaner depressant addition was reduced to 90 g/t in all further locked cycle tests.

Two locked cycle tests were conducted on development Composite 3. The first test, using flowsheet 1(refer to Figure 13.27) with a reduction in cleaner depressant dosage achieved 65% 3PE+Au recovery in the combined high grade and medium grade concentrate with further 15% 3PE+Au recovery via the low grade cleaner circuit resulting in a total overall recovery of 80%. This confirmed that lower cleaner depressant addition allowed for maximum 3PE+Au recovery via the high grade and medium grade cleaner circuit. A second test on Composite 3, was conducted using flowsheet 2 (refer to Figure 13.28) with the aim of targeting a lower overall mass pulled higher concentrate grade. This test achieved a lower mass pull of 2.6% as compared to 3.1% for the first test. Concentrate grade improved by 10 g/t, however, overall 3PE+Au recovery was 2.5% lower. Thus, no further locked cycle testing of flowsheet 2, was conducted.

A further two locked cycle tests were conducted on high grade Composites 4 (5.7 g/t 3PE+Au) and Composite 5 (5.4 g/t 3PE+Au) using flowsheet 1 (refer to Figure 13.27) with a cleaner circuit depressant addition rate of 90 g/t. These tests achieved a PGE recovery of approximately 85% at a smelter-grade final concentrate grade of 120 g/t and 110 g/t for Composite 4 and 5 respectively.

13.8.2.2 Locked Cycle Testing on Platreef 2017 FS Blend Composites

Locked cycle testing was conducted on blend composites representing blends of geometallurgical units from various drill core samples as per the expected mined ore blends, with the focus placed on testing of an ore blend representing the first 5 years of mining. These tests were conducted using flowsheet 3 (refer to Figure 13.29)

For the Platreef 2017 FS blend sample and Year 1-5 Blends 1-4, two locked cycle tests were conducted. The first targeting a low mass pull and the second targeting a high mass pull. For the Year 1-5 Blend 5 and 6 a single locked cycle test, targeting a low mass pull was conducted. The locked cycle testing was conducted on ore blend composites representing 3PE+Au head grades in the range 2.5g/t – 5.1g/t.

The low mass pull tests were conducted using the optimized depressant regime as derived in the development testing phase. In the tests targeting a high mass pull, the rougher depressant addition was reduced based on the visually performance of the float. Initially for the high mass pull test on the 5.1g/t composite no rougher depressant was added, however after reviewing the test results it was decided to increase the rougher depressant to 125g/t and adjust cleaner scrape rates in the remaining tests targeting a high mass pull.

This locked cycle flotation testing achieved recoveries in the range 83.1%-88.7% at a smelter-grade final concentrate of approximately 60g/t-95g/t. In, this locked cycle testing, concentrate mass pull was found to be dependent on the 3PE+Au concentrate grade relative to the 3PE+Au head grade (upgrade ratio). When a low mass pull was targeted, higher concentrate grades were achieved (high upgrade ratio). Conversely when a high mass pull was targeted lower concentrate grades were achieved (low upgrade ratio).

With the exception of the 5.1g/t Year 1-5 blend composite and the 4.3g/t LoM blend composite, 3PE+Au recovery was found to be lower for tests in which a higher concentrate grade was targeted. This was attributed to the fact that the lower mass pull tests on these two composites achieved near optimal recovery and at an upgrade ratio of approximately 21, which is lower than the target upgrade ratio of approximately 25 for the low mass pull operating conditions. Attempts to improve recovery at higher mass pull for these composites only resulted in grade dilution due to increased MgO recovery via the low grade cleaner circuit.

In the locked cycle tests on the FS ore blend composites, the high grade and medium grade cleaner circuit was found to produce a high-grade concentrate of approximately 100g/t - 130g/t, at a 3PE+Au recovery of 68% - 83%. The low grade cleaner circuit produced a low-grade concentrate of 15g/t – 35g/t and allowed for additional 3PE+Au recovery of 5%-15%. It is thus evident that it is possible to achieve high concentrate grades of 100g/t – 130g/t for the Platreef ore, however this would result in a lower 3PE+Au recovery as it would not be possible to achieve optimal 3PE+Au recovery from the low grade cleaner circuit within the concentrate grade constraints.

13.9 Tailings Thickening Test Work

13.9.1 Tailings Thickening Testing at Vietti Slurrytec

Vietti Slurrytec conducted test work on one tailings composite sample representing the combined rougher and cleaner flotation tailings from testing on FS Composites 1-4. The objective of the testing was to obtain design and operating parameters. The test work included material characterisation, slurry conditioning, static sedimentation and bench-top dynamic thickening test work.

The material characteristics and operating parameters obtained from the Vietti Slurrytec test work are presented in Table 13.16 and Table 13.17.

Table 13.16 Summary of Material Characterisation Results Vietty Slurrytec

Parameter	Value
Tailings pH	8.6
Process Water pH	7.6
Tailings D80	82 microns
Specific Gravity	3.12

Table 13.17 Summary of Optimal Tailings Thickening Test Results Viette Slurrytec

Parameter	Value
Thickener Type	High Rate
Feed Solids Concentration	12.5% (m/m)
Flocculant Type	Magnafloc 5250
Flocculant Dosage	30g/t

The bench-top dynamic thickening tests indicated a decrease in underflow solids concentration from 66% to 57% as the solids flux rate increased from 0.30-0.90 t/m2/h, with good overlow clarity achieved.

It is recommended that pipe loop and pump de-rating test work be conducted prior to the detailed design phase, in order to confirm the tailings pumping system design and operating parameters.

13.9.2 Golder Tailings Thickening and Filtration for Backfill Plant Design

In addition to the tailings settling test work conducted at Vietti Slurrytec, further test work was undertaken by Golder on Platreef Mine (Platreef) tailings during the Feasibility paste fill study in 2016 (Golder project no. 1530678) to allow the paste plant) design to proceed to a feasibility level. For detail laboratory test results please refer to Platreef 4 Mtpa Feasibility Study – Section 6.2 – Backfill (DRA Report Number: DRA-J0283-STU-REP-909, Ivanplats Report Number: 1051-EV-00-209) Appendix 1 Laboratory Assessment on Concentrator Tailings for the Production of Paste Backfill for the Platreef Project (Golder Lab Report) dated October 28, 2016 (Report Number: 1530678 Rev1).

This test work was undertaken to determine the material characteristics as well as performing settling and filtration tests. Dewatering testing was performed at the Golder Process Infrastructure and Design (PID) Laboratory in Sudbury, Ontario, Canada. The results from the following tests are used in this study:

- Material characterisation;
- Settling; and
- Filtration

13.9.2.1 Test Work Samples

Two samples were received in dry form: "1530678 Combined Final Tails" and "1530678 Combined Final Tails - S2". The samples were received at the Golder laboratory in Sudbury, Ontario Canada on January 7, 2016 and April 14, 2016 respectively. Sample "1530678 Combined Final Tails" was used for all materials property testing in the laboratory.

Process water was also received at the Golder laboratory in Sudbury on January 7, 2016 and was used, when required, for all the laboratory material testing.

As required by Golder, the tailings and process water were assessed individually for concentrations of volatile organic compounds, hydrogen cyanide gas, hydrogen sulphide gas, heavy metals, and dissolved cyanide. All concentrations were found to be within acceptable levels.

The test work included material characterisation, settling tests and filtration tests. It should be noted that the Golder test work was done to obtain a maximum underflow solids concentration to produce a vacuum filter feed for the paste plant while the Vetti Slurrytec test work was done to obtain a suitable underflow solids concentration for the pumping of tailings to the TSF.

The material characterisrics of the samples as tested are presented in Table 13.18.

Table 13.18 Summary of Material Characterisation Results Golder

Parameter	Value
Tailings pH	8.9 – 9.0
Process Water pH	6.9 which was adjusted to pH 9.0 prior to testing
Tailings D80	105 microns
Specific Gravity	3.06 – 3.08

The D80 particle size distribution (PSD) of both the tailings samples were 105 microns as measured by Golder through sieve and laser evaluation. The overall PSD curves were also verified by Platreef as being in the expected range. In comparison, the D80 PSD as determined by laser diffraction method in the Vietti Slurrytec testing was 82 microns. The D80 measured PSD of samples prior to shipment for tailings testing at Golder and Vietti was 75 microns as determined by sieve and hydrometer which was colloborated through the same method of testing at XPS on a Golder tailings sample

It should be noted that the sieve and hydrometer method is unable to quantify the PSD of a sample in the 20 micron range.

The tailings specific gravity as provided by DRA ranged from 3.04 to 3.19, this was confirmed in both the Golder and Vietti Slurrytec material characterisation testing. The latter value was used as basis in the TSF design criteria.

Mineralogy and chemistry tests were conducted on the tailings solids for the following purposes:

- To confirm that the sample as tested falls within anticipated mineralogical variations of the ore;
- To allow comparison, should future testing occur.

The chemistry and mineralogy of the tailings samples indicated that the major constituent of the samples are SiO2 and MgO (~75%) and the remaining major constituents (~25%) being Fe2O3, Al2O3, and CaO. Enstatite is the predominant mineral.

The concentration of Chlorite, a mica like mineral, and Talc, can have a detrimental effect on vacuum filtration results.

It should be noted that since the two tailings samples (Final Tails and Combined Final Tails S2) had nearly identical characteristics, were therefore deemed "the same" only one tailings sample was used for all remaining laboratory testing.

13.9.2.2 Test Work Samples

The settling tests for the 1530678 Combined Final Tails were completed in an iterative fashion to determine optimum values involving flocculant type, flocculant dosage, and feed slurry solids concentration. Different flocculant dosage and feed rates were evaluated in 500 ml, 1 l and 4 l vessels.

The dewatering tests conducted by Golder are focused on the use of non-ionic and anionic flocculants. Cationic flocculants are not normally tested due to their potential harmful effects on fish species should they be released into the aquatic environment.

The dewatering / settling results are summarised in Table 13.19 and Figure 13.31.

Table 13.19 Summary of Tailings Thickening and Filtration Test Results for Backfill Plant Design

Parameter	Value
Thickener Type	High Rate
Feed Solids Concentration	15 - 20% (m/m)
Flocculant Type	AN 905 VHM
Flocculant Dosage	20 - 25g/t
24 Hour Underflow Solids	69% (m/m)
Maximum Underflow Solids	78% (m/m)

The settling tests, as presented in Figure 13.31provided an observation regarding the rapidity of settling that will be an important consideration in the flocculation process and the operation of a thickener. The settling curve indicates very rapid settling into the compression zone within 1 minute of flocculation with a dosage rate of 25 g/t and 20 wt% solids feed. An accidental overdose of flocculant above the recommended maximum of 25 g/t range will greatly increase the settling rate and could be problematic in "bogging" the thickener's rake mechanisms. Consideration is warranted in retaining good flocculant addition control.

As a general summary, the tailings will dewater acceptably using available thickening technology. A further improvement of as much as 5 wt% in underflow density is often observed in production scale thickeners when compared to the bench scale 4 l test 24 hour results.

Figure 13.31 Settling Curve, 15-20% Solids Feed, 20-25 g/t

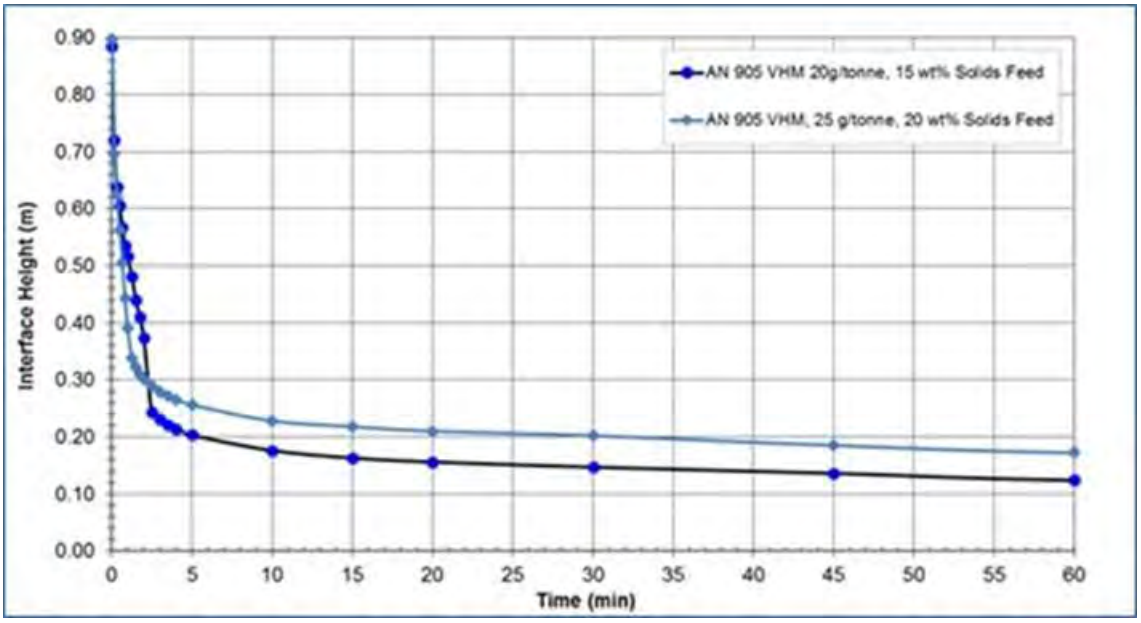


Figure by Golder, 2017

13.9.3 Filtration Testing

Vacuum filtration tests, to simulate the results associated with disc filters, were conducted at a vacuum level of 557 mm (22") Hg. Typical cycle times (a full rotation) for vacuum disc filters are in the range of 90 to 180 seconds.

The vacuum filtration testing program was performed with a slurry feed solid concentration of 69 wt% which is the underflow value obtained in the 4 l tests. An additional series of filtration tests were also performed at 64 and 74 wt% solids.

The filter leaf was equipped with a small section of industrial grade polypropylene felt filter cloth. The leaf was immersed into the slurry and simulated production scale vacuum filters where the sectors dipped into the slurry in an agitated filter tank as the disc rotated. Proper technique and cycle times simulating continuous filters provide an estimate of cake loading, moisture and discharge characteristics.

Since the test was performed in the laboratory, under ideal conditions, actual loading is multiplied by 0.7 to reflect variable or upset conditions which may occur in plant operations.

The following parameters were used for testing:

- Elevation:1109 masl (1)
- Vacuum Level: 557 mm (22") Hg
- Temperature: 18.1˚C
- Filter Cloth: Industrial grade polypropylene felt 133-03 (25-40 cfm rating)
- Apparatus: 100 mm (4 inch) diameter, dip style filter head
- The Process water pH was adjusted to 9.0 prior to filtration testing.

The results are presented in Table 13.20, Table 13.21 and Table 13.22 as well as on Figure 13.32.

Generally, the higher the solids content feed to the vacuum disc filters, the higher the loading capacity. Loading capacity from 64 wt% to 74 wt% solids feed provides a range of values to consider for disc filter sizing purposes. Based on the settling and filtration results, 609 kg/m^2/hr cake loading rate at 69 wt% solids with 90s cycle time is chosen for disc filter sizing.

Table 13.20 Filtration Results - 74 wt% Solids

Vacuum Level	Cycle Time	Cake Thickness	Cake Loading	Cake Moisture	Final Density
(mm Hg)	(sec)	(mm)	(kg/m^2/hr)	(wt%)	(wt% Solids)
557	30	10.5	1535	18.3	81.7
	45	11.7	1221	18.1	81.9
	90	14.9	769	17.4	82.6
	135	17.8	596	17.1	82.9
	180	18.9	504	17.1	82.9

Table 13.21 Filtration Results - 69 wt% Solids

Vacuum Level	Cycle Time	Cake Thickness	Cake Loading	Cake Moisture	Final Density
(mm Hg)	(sec)	(mm)	(kg/m²/hr)	(wt%)	(wt% Solids)
557	30	7.5	1079	18.0	82.0
	45	9.4	892	17.1	82.9
	90	12.6	609	16.7	83.3
	135	14.1	460	16.9	83.1
	180	15.8	383	16.7	83.3

Table 13.22 Filtration Results - 64 wt% Solids

Vacuum Level	Cycle Time	Cake Thickness	Cake Loading	Cake Moisture	Final Density
(mm Hg)	(sec)	(mm)	(kg/m²/hr)	(wt%)	(wt% Solids)
557	30	5.7	818	17.2	82.8
	45	7.5	694	17.8	82.2
	90	8.8	428	17.2	82.8
	135	11.5	346	16.7	83.3
	180	12.4	283	16.6	83.4

Figure 13.32 Cake Loading vs. Cycle Time - 1530678 Combined Final Tails

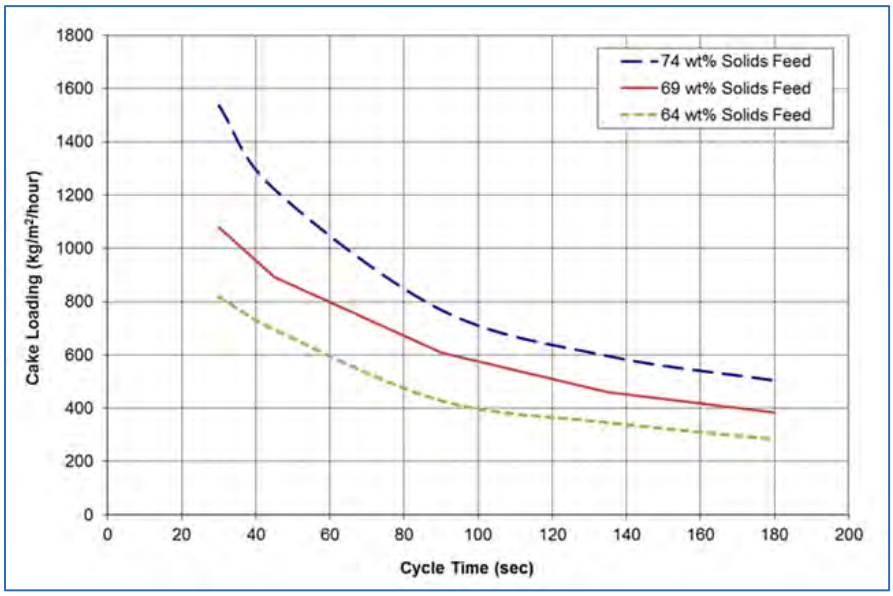


Figure by Golder, 2017

13.9.4 Golder Tailings Thickening and Filtration Test Work Conclusions

From the laboratory assessment conducted on the feasibility paste fill samples it was concluded that the paste fill tailings can be initially dewatered through the use of available thickener and filtration technology.

13.10 Plant Recovery Estimate

A nominal head grade recovery relationship was derived based on locked cycle test results conducted using the MF1 copper collector reagents suite and the optimised Platreef 2917 FS flowsheet (Flowsheet 3) as presented in Figure 13.29.

13.10.1.1 Summary of Test Data Used

The Platreef recovery regression models, as presented, have been derived using the locked cycle test results conducted using domain composites, comprising of the expected mine blend in the first five years of production and the overall life of mine blend. The locked cycle test data for the initial development samples were based on the Platreef 2015 PFS mine blend using interim development flowsheets and have thus not been used in the recovery modelling. The development sample locked cycle test data has however been compared to the recovery model predictions and was found to produce a good fit to the recovery models as presented below.

13.10.1.2 Derivation of 3PE+Au Recovery Models

Locked cycle test work showed a strong relationship between concentrate mass pull, achievable upgrade ratios and consequently final concentrate grades. Mass pull was dependent on the PGE upgrade ratio targeted in the locked cycle testing as presented in Figure 13.33. This curve illustrates that as higher concentrate grades are targeted (High upgrade ratio), the overall concentrate mass pull will decrease.

Figure 13.33 Locked Cycle Test Mass Pull as a Function of PGE Upgrade Ratio

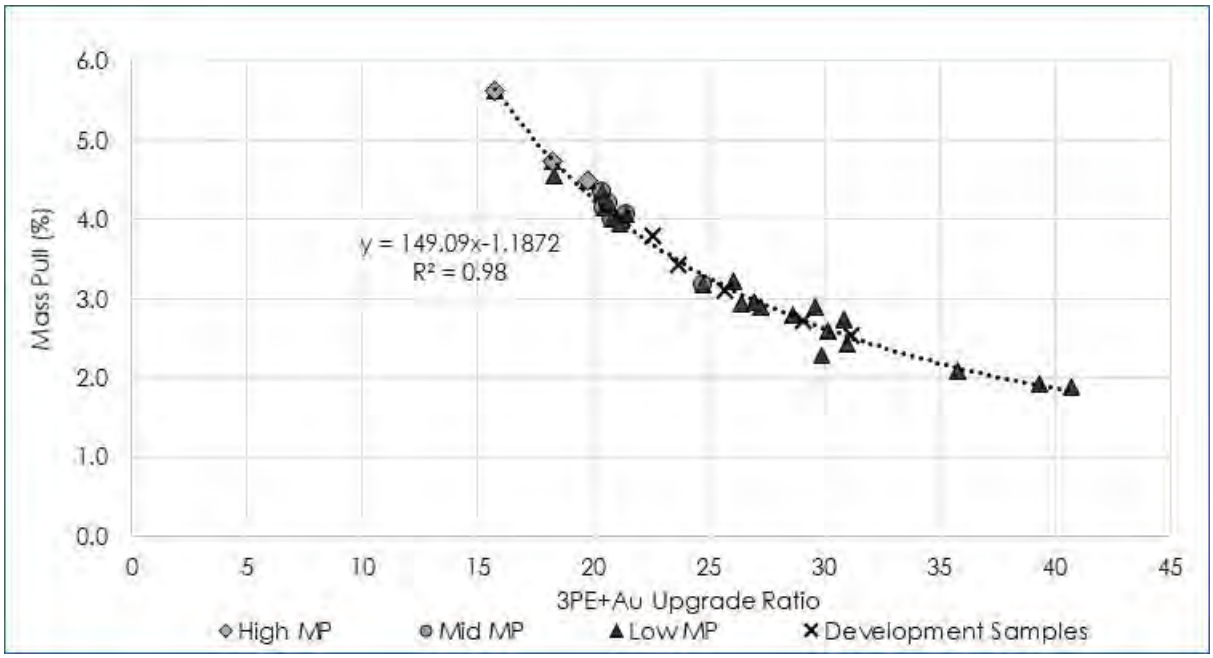


Figure by DRA, 2017.

In turn, test work showed that metal recoveries were dependent on mass pull as presented in Figure 13.34. It is noted that some scatter exists between the results, which is believed to be attributed to sample variances, test operator inconsistencies, assay uncertainty and variances in the final concentrate grade achieved.

Figure 13.34 3PE+Au Recovery as A Function of Mass Pull

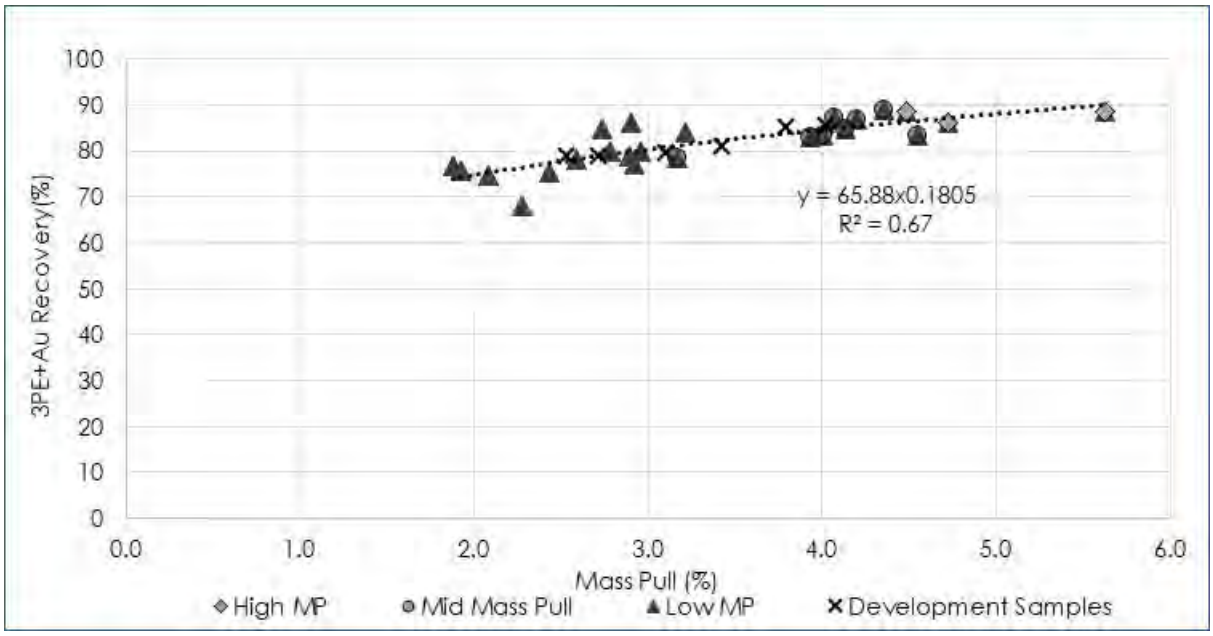


Figure by DRA, 2017.

The combination of Figure 13.33 and Figure 13.34 thus illustrates that as higher 3PE+Au concentrate grades are targeted (High upgrade ratio) the overall concentrate mass pull will decrease, and consequently the 3PE+Au recovery would be lower than when targeting a lower concentrate grade (low upgrade ratio).

These relationships between mass pull and 3PE+Au recovery, as a function of upgrade ratio, allow for modelling of the expected recoveries based on feed grade and target concentrate grade and form the basis of the Platreef recovery derivation.

The recovery model has been derived as follows:

1. Mass pull is calculated based on the targeted 3PE+Au upgrade ratio (concentrate grade/feed grade).
2. Individual Pt, Pd, Rh and Au recoveries are calculated based on the mass pull recovery curves derived per element.

13.10.1.3 3PE+Au Recovery Model Verification

The PGE predicted recovery was compared to actual test results in order to verify the accuracy of the models. The modelled PGE, Pt, Pd, Rh and Au recovery as compared to actual test recoveries are presented in Figure 13.35 and Figure 13.36.

Figure 13.35 Combine PGE (3PE+Au) Recovery Model Verification

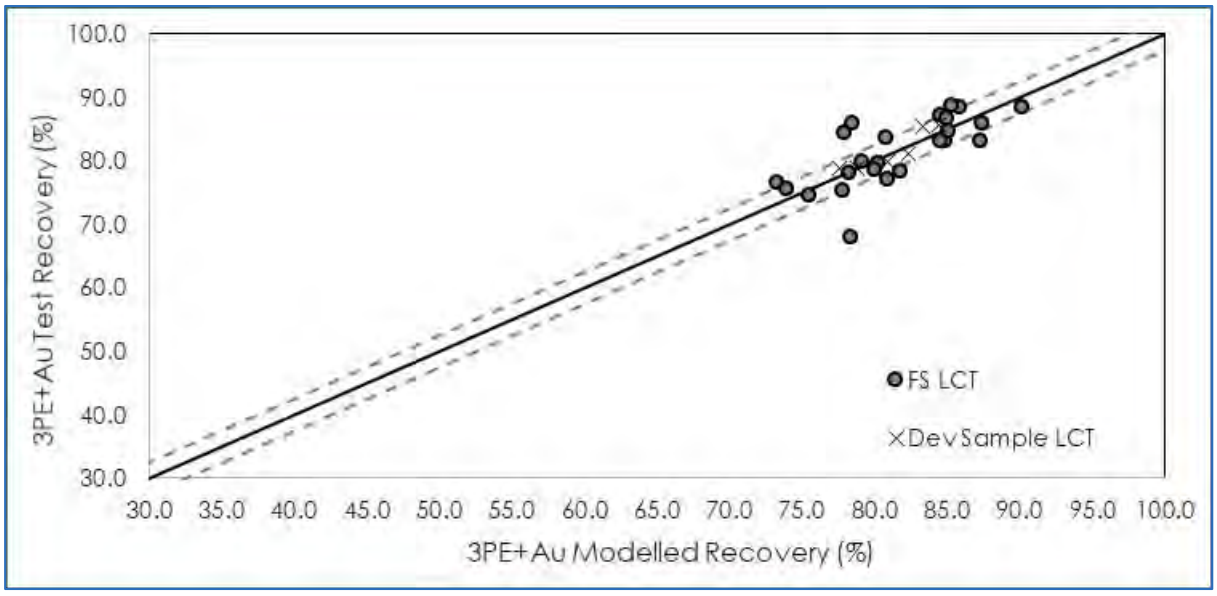


Figure by DRA, 2017.

Figure 13.36 Individual, Pt, Pd, Rh and Au Recovery Model Verification

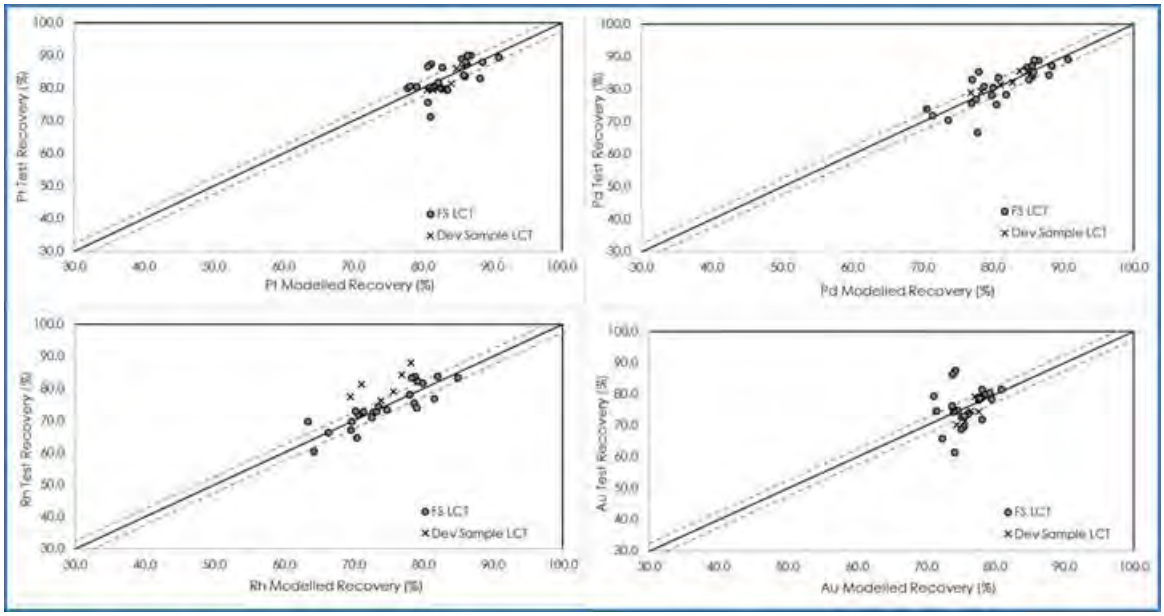


Figure by DRA, 2017.

The modelled recovery data was found to fit the actual test data well, the upper and lower lines in these curves represent the bounds associated with modelled vs actual recovery variance. For Pt, Pd, Rh and Au, the variance was found to be within ±2.5% in the vast majority of instances.

13.10.1.4 Derivation of Copper, Nickel and Sulphur Recovery Models

During commercial operation, the final concentrate mass pull and 3PE+Au recovery will be determined by the upgrade ratio required to achieve a minimum final concentrate grade of 85 g/t (3PE+Au). For the range of head grades tested, and the expected mass pull range as determined based on the Platreef 2017 FS mining schedule, the nickel, copper and sulphur recoveries are best described by head grade versus recovery relationships.

The copper, nickel and sulphur test recoveries achieved to the combined final concentrate (HG+MG+LG) in locked cycle testing were plotted relative to the respective test head grade in order to derive head grade-recovery curves.

13.10.1.5 Base Metal and Sulphur Recovery Model Verification

The method as outlined was used to derive recovery correlations for copper, nickel and sulphur. The recovery predictions were compared to laboratory test results in order to verify the model accuracies as presented in Figure 13.37.

Figure 13.37 Copper, Nickel and Sulphur Recovery Model Verification

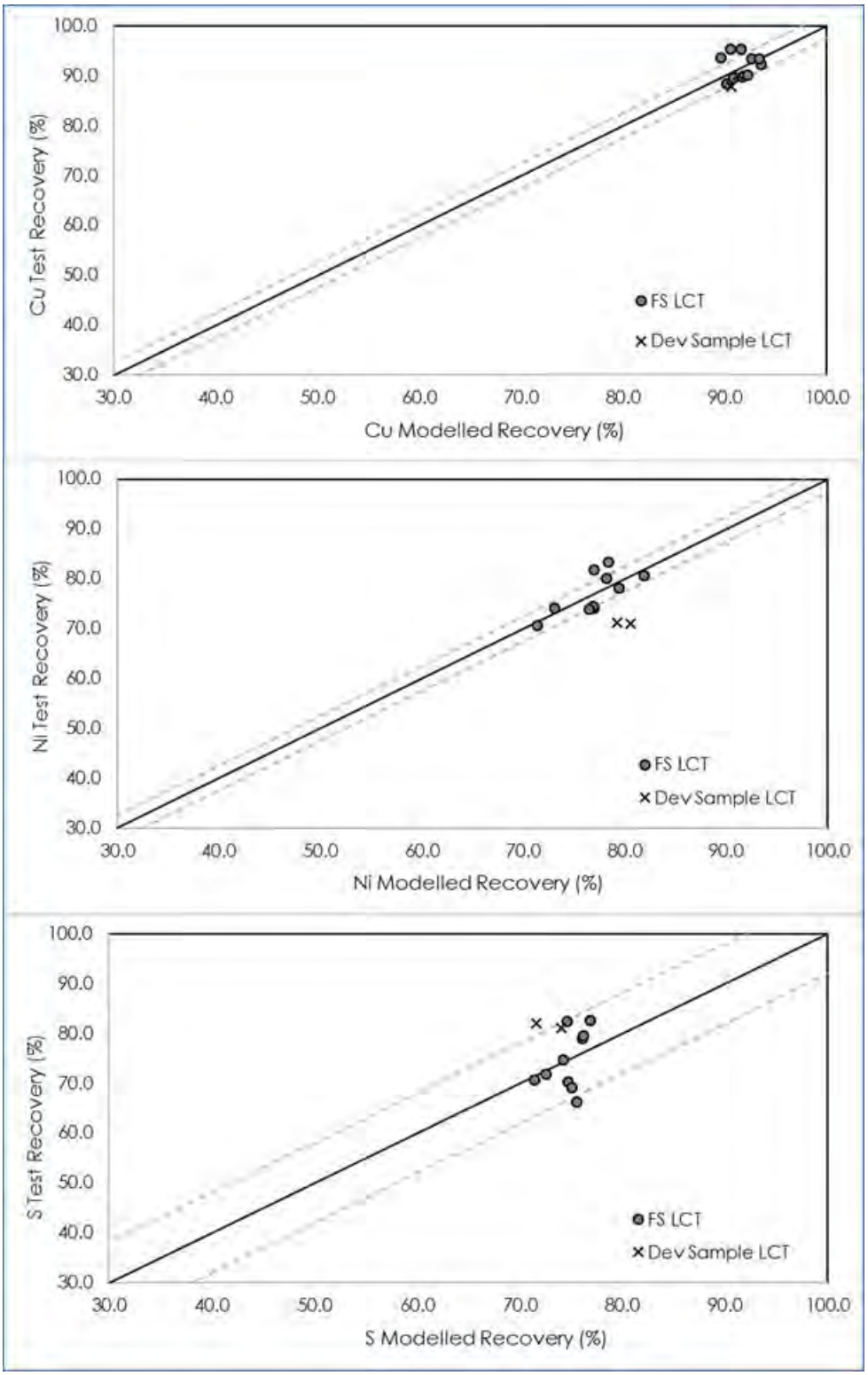


Figure by DRA, 2017.

The modelled recovery data was found to fit the actual Platreef 2017 FS locked cycle test data well for copper and nickel, where the variance was found to be within ±2.5% in the majority of instances. The variance in the modelled vs actual sulphur recovery was found to be within ±10.0%.

The higher variance for sulphur is possibly due to the analytical methods employed and is currently being reviewed in consultation with Mintek, however this would not affect the overall project economics as sulphur is not a payable metal. The effect of the uncertainty is discussed and illustrated in Sections to come.

13.10.1.6 Recovery Algorithms

The final recovery model (as a function of feed grade and targeted concentrate grade) for the individual metals are summarised in Table 13.23.

Table 13.23 Platreef 2017 FS Recovery Algorithms

Element/Metal	Recovery Algorithm	Head Grade Range	Max Recovery
Platinum	Recovery = $e^{4.1501+0.15166\times\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}$	1.0 – 2.7g/t	90.9
Palladium	Recovery = $e^{4.8401-\frac{0.79022}{\left(149.09\times\left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.5}}}$	1.2 – 2.7g/t	90.7
Rhodium	Recovery = $52.042 + 19.041 \times Ln\left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)$	0.06 – 0.19g/t	84.9
Gold	Recovery = $66.424 \times \left(149.09 \times \left(\frac{PGE\ Concentrate\ Grade}{PGE\ Head\ Grade}\right)^{-1.1872}\right)^{0.1137}$	0.18 – 0.35g/t	80.8
Copper	Recovery = $129.78 \times Cu\ Head\ Grade^{0.2174}$	0.15 – 0.23%	93.6
Nickel	Recovery = $125.33 \times Ni\ Head\ Grade^{0.5145}$	0.30 – 0.44%	82.2
Sulphur	Recovery = $75.569 \times S\ Head\ Grade^{0.328}$	0.78-1.05%	76.8

By using these recovery algorithms for each individual element, it is possible to calculate the expected overall 3PE+Au recovery based on the 3PE+Au target concentrate grade.

The PGE (3PE+Au) recovery models have been plotted relative to the locked cycle test results and expected closed circuit recoveries derived from the open circuit variability testing in Figure 13.38 to Figure 13.42. As previously noted, the recovery estimates as derived from open circuit variability results are not definitive, however these results do provide an indication of the expected variability. It should be noted that the Platreef 2017 FS recovery estimates compare well with models developed during the Platreef 2015 PFS phase, highlighting the consistency between the various campaigns. Variances when compared to test results are exaggerated due to the fact that variances exist in the final concentrate grade achieved during test work and modelled.

Figure 13.38 3PE+Au Recovery as a Function of Feed Grade and Targeted Concentrate Grade

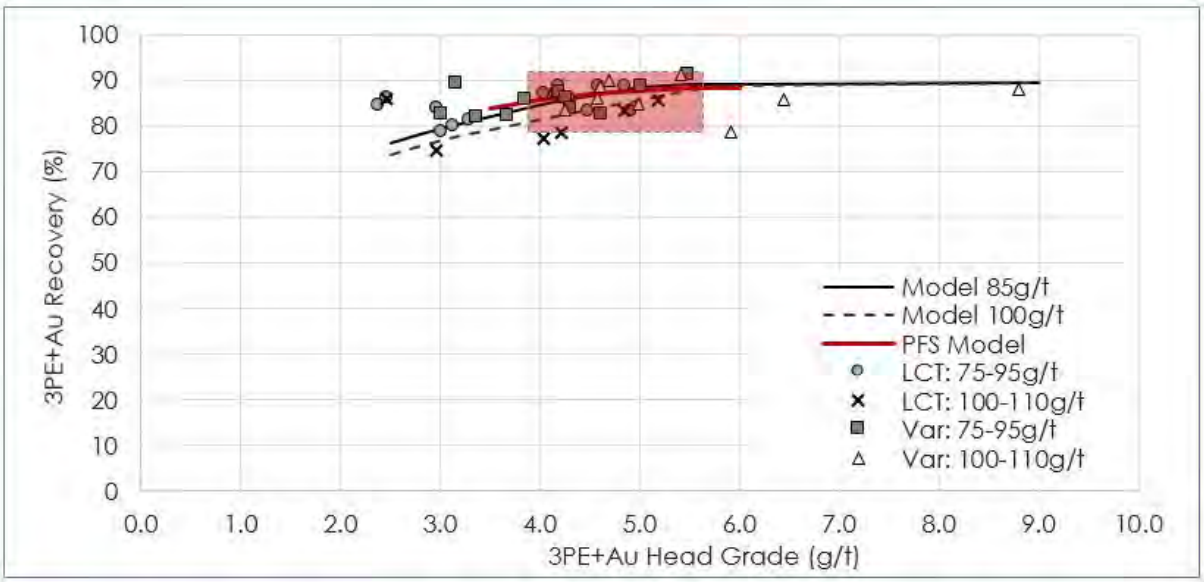


Figure by DRA, 2017.

Figure 13.39 Platinum Recovery as a Function of Feed Grade and Targeted Concentrate Grade

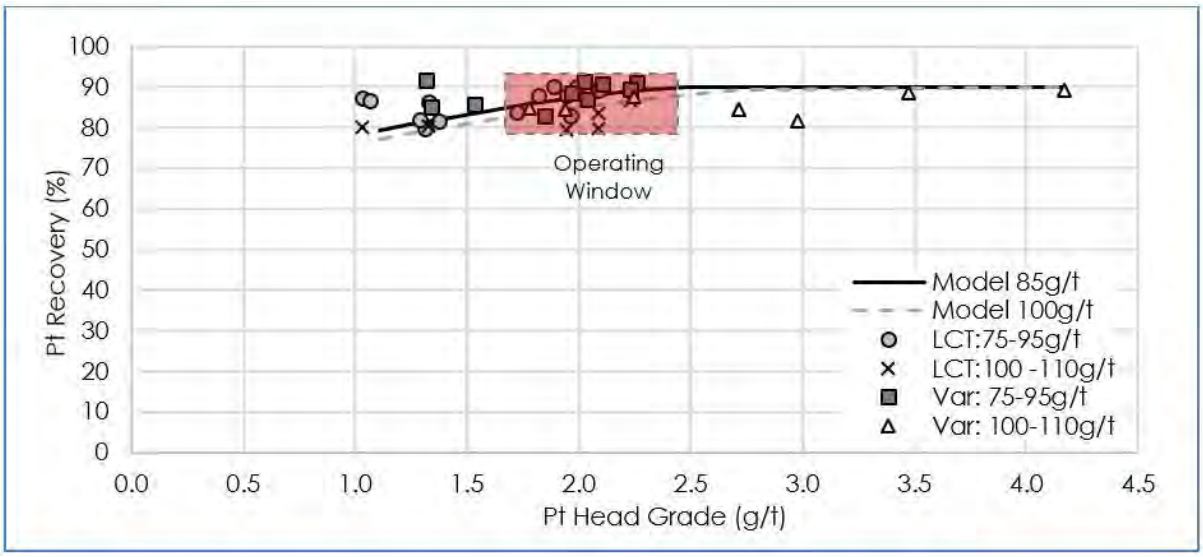


Figure by DRA, 2017.

Figure 13.40 Palladium Recovery as a Function of Feed Grade and Targeted Concentrate Grade

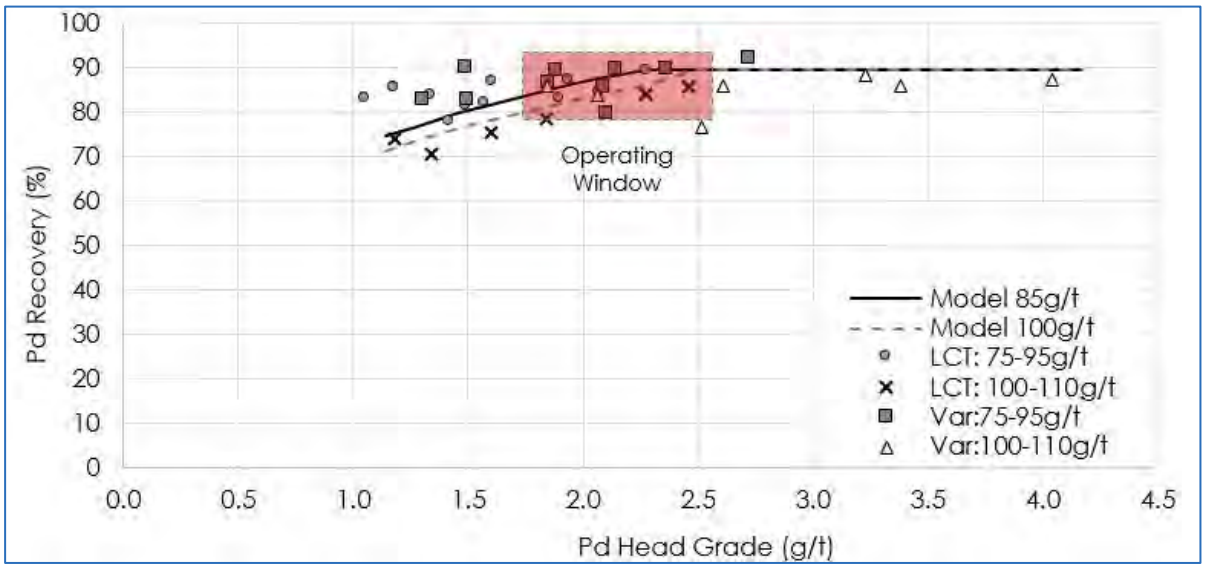


Figure by DRA, 2017.

Figure 13.41 Rhodium Recovery as a Function of Feed Grade and Targeted Concentrate Grade

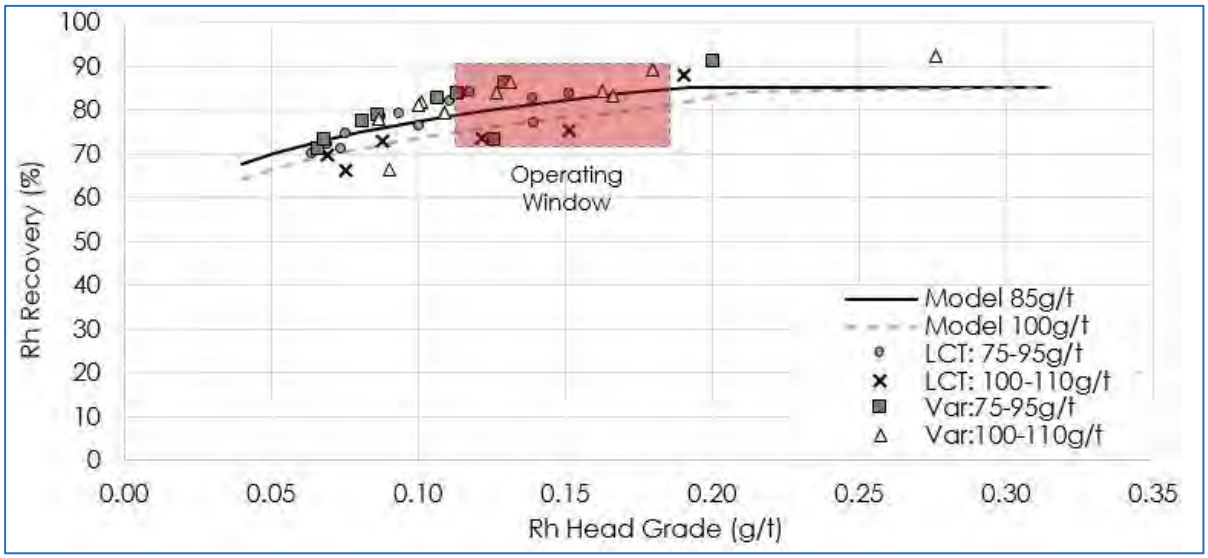


Figure by DRA, 2017.

Figure 13.42 Gold Recovery as a Function of Feed Grade and Targeted Concentrate Grade

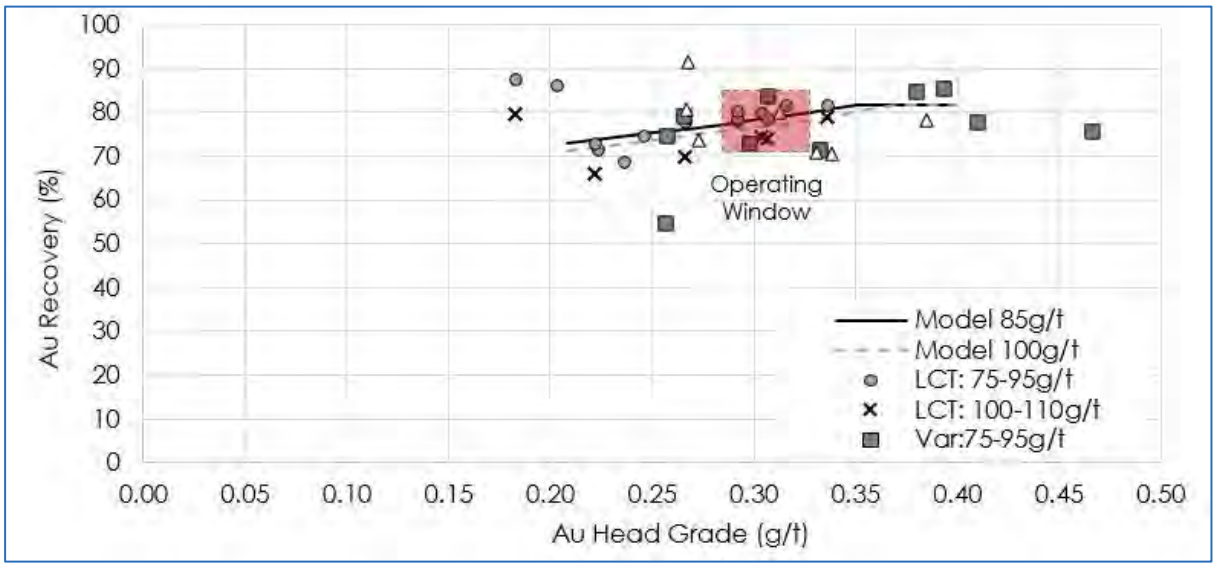


Figure by DRA, 2017.

The copper and nickel recovery models have been plotted relative to the locked cycle test results and expected closed circuit recoveries derived from the open circuit variability testing in Figure 13.43 and Figure 13.44.

As previously noted the recovery estimates as derived from open circuit variability results are not definitive, however these results do provide an indication of the expected variability. It should be noted that the Platreef 2017 FS recovery estimates have indicated improved copper and nickel recovery as compared to Platreef 2015 PFS testing at similar head grades. This is possibly as a result of the optimised Platreef 2017 FS flotation conditions, targeting high mass pull using the copper collector reagent suite.

Figure 13.43 Copper Recovery as a Function of Feed Grade

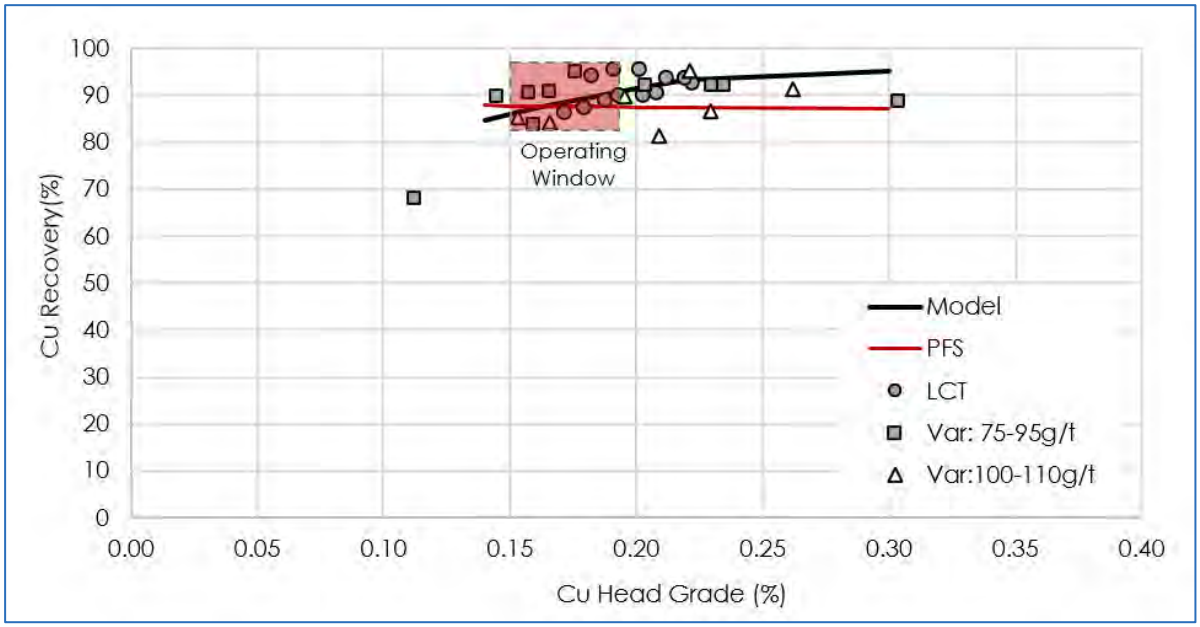


Figure by DRA, 2017.

Figure 13.44 Nickel Recovery as a Function of Feed Grade

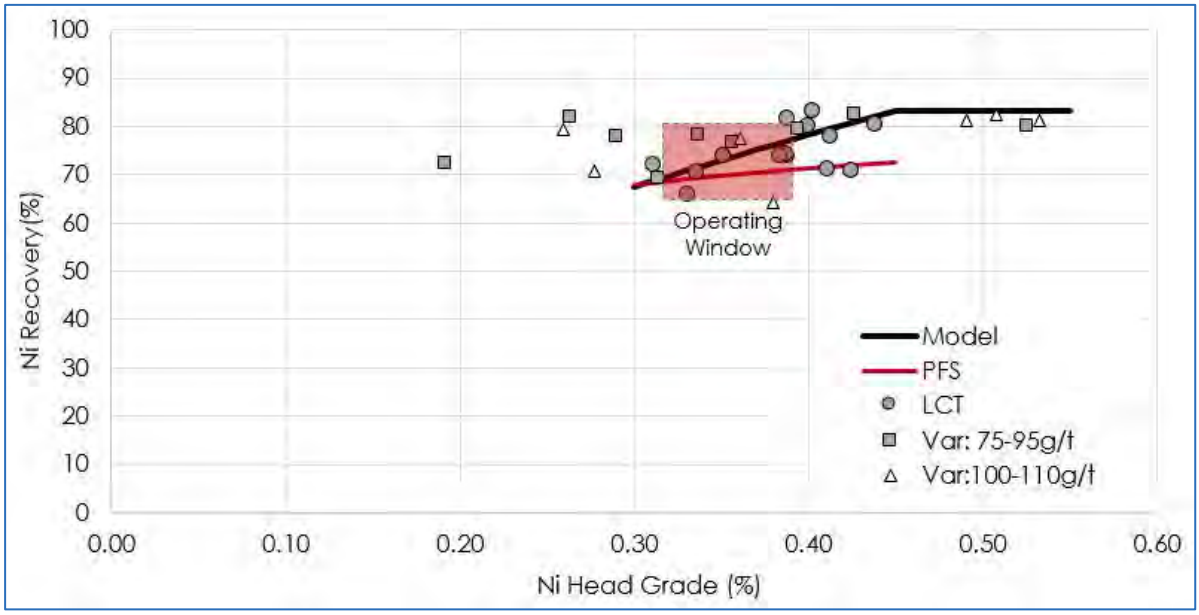


Figure by DRA, 2017.

13.10.1.7 Life of Mine Recovery Estimates

The recovery estimate for PGE's and expected concentrate mass pull over life of mine are presented in Figure 13.45 and Figure 13.46. The concentrate grade profile and calculated sulphur grade in concentrate are presented in Figure 13.47 and Figure 13.48. These recoveries and grade profiles have been based on the expected concentrator feed grades from the Platreef 2017 FS mine schedule. The recoveries have been derived for a minimum target grade of 85 g/t. It should be noted that the modelled recoveries are steady-state recoveries and do not consider any transient operations. The modelled recovery profile includes a 2% recovery discount in the first six months of operation to cater for start-up and commissioning.

Figure 13.45 Modelled Steady State Lom 3PE+Au Recovery Profile

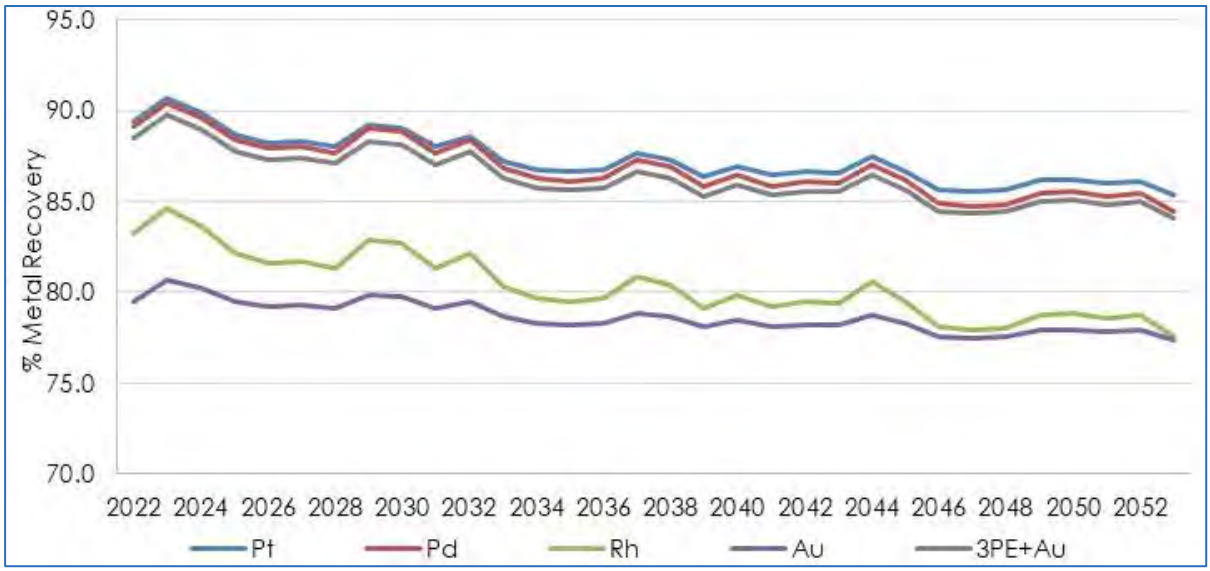


Figure by DRA, 2017.

Figure 13.46 Modelled LOM Head Grade and Mass Pull Profile

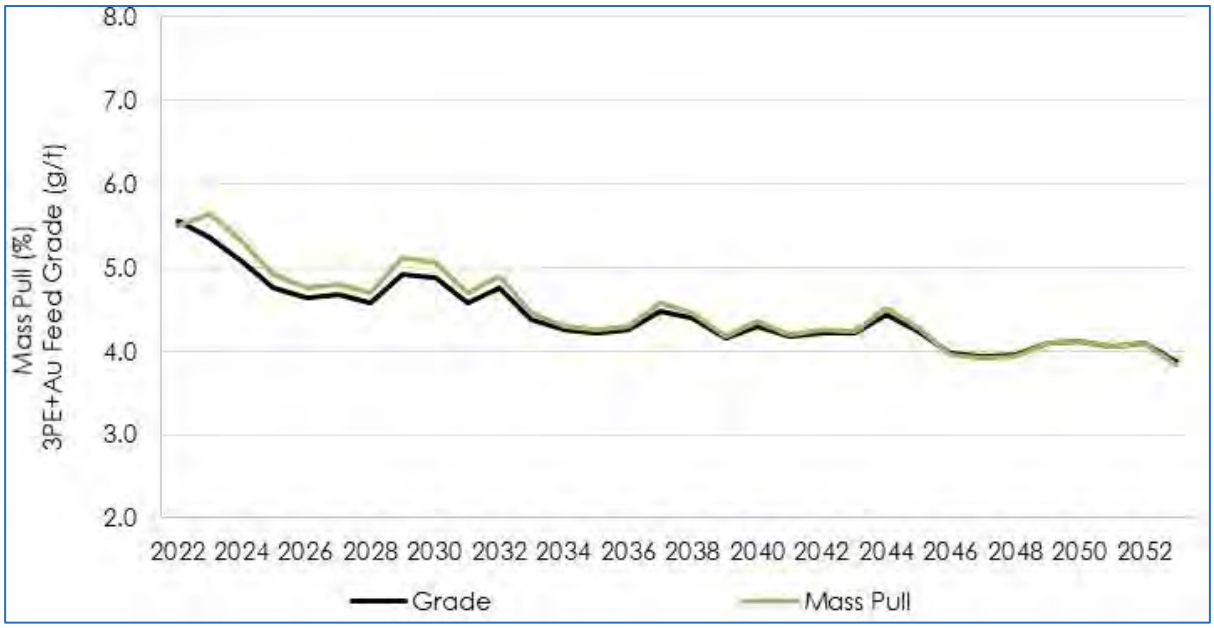


Figure by DRA, 2017.

Figure 13.47 Modelled LOM Concentrate Grade Profile

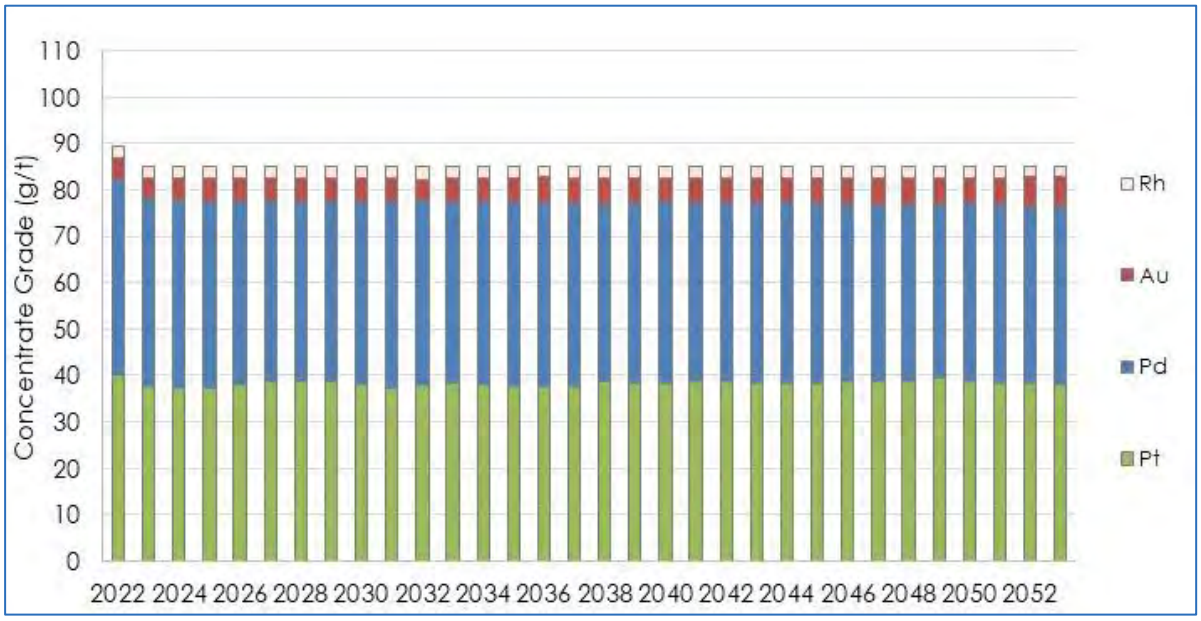


Figure by DRA, 2017.

Figure 13.48 Modelled LOM Sulphur in Concentrate

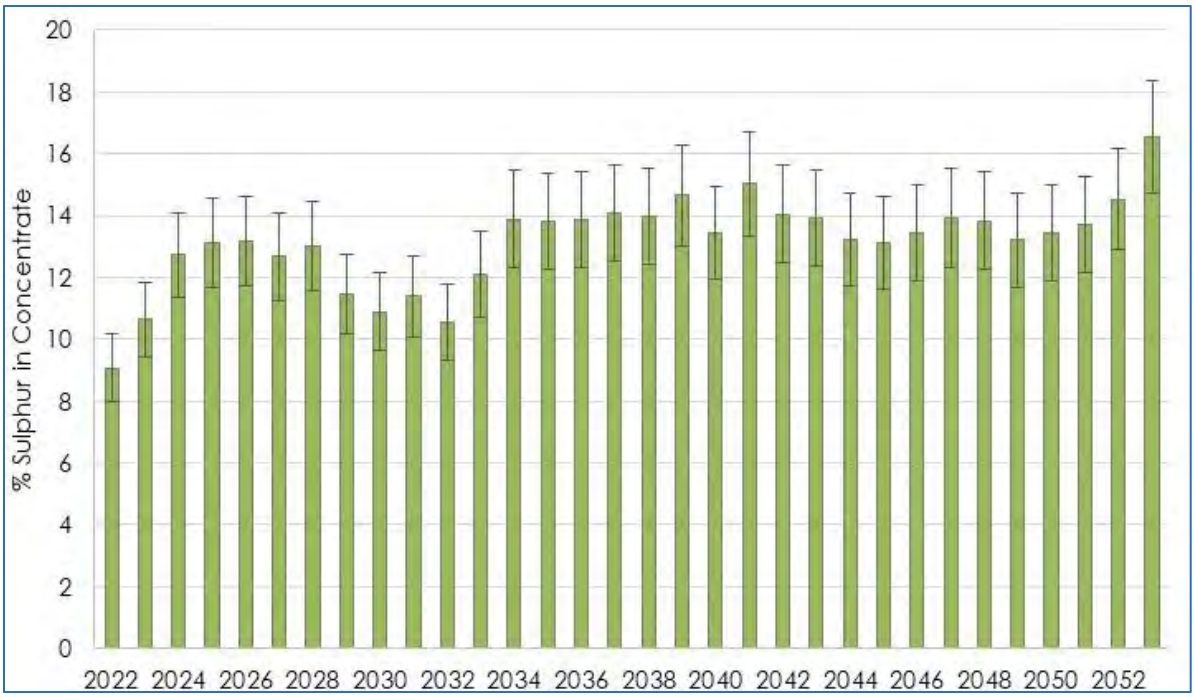


Figure by DRA, 2017.

The error bars as presented on the expected sulphur grades in concentrate (Refer Figure 13.48) represent the 10% variance as evidenced in the modelling of the sulphur recovery test data.

13.10.2 Comments on the Recovery Estimate

Locked cycle testing on mineralised ore blends with the inclusion of footwall and hanging wall has been conducted. Further to this, detailed open circuit variability testing was conducted in order to quantify the expected recovery ranges and to highlight the degree of variability that can be expected.

It is also noted that in addition to the recovery uncertainty demonstrated during variability testing, flotation recovery for full-scale operations could be lower than that achieved in a laboratory due to operational inefficiencies such as those listed below:

- Variation in ore types/blends: The mine plan includes geometallurgical units T1, T2U and T2L as well as fractions of hanging wall and footwall. The mine plan caters for mining in ten distinct areas.
- Power: The laboratory flotation cell power (and air) inputs are extremely high (typically in the order of 10 kWh/m^3). This may tend to give higher recoveries due to the improved fines (<20 µm) recovery.
- Milling type: The milling in the laboratory is generally undertaken using rod mill, as opposed to the actual plant which is often undertaken with ball milling. The difference in particle size distribution between these two types may have an effect on performance.

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

In order to address as many of these problems as possible the plant design will allow a high level of instrumentation and control within the flotation and milling circuit with the allowance for installation of Float Star and a Blue Cube online analyser to allow for improved flotation control. Process operators need to be trained and supervised so as to reduce the occurrence of losses due to poor operational practices.

It should also be noted that Mintek has reported that the Pt, Pd, Rh and Au detection limit is 0.02 ppm by ICP-OES analysis method used during the Platreef test campaign. All Rh recovery calculations as presented in this document have been based on this minimum detection limit in instances where values below this limit have been reported.

13.11 Future Test Work

The test work conducted to date is considered adequate to meet the requirements of the Platreef 2017 FS. However, in order to further evaluate optimisation opportunities, confirm design parameters, evaluate scale-up factors and establish criteria for performance guarantees, a mini pilot plant test work programme will be undertaken as part of the project implementation phase. It is not believed that this will have a material impact on the overall capital and operating cost estimates derived in the current study phase, and is furthermore expected to validate the recovery and concentrate specifications achieved during bench scale testing on similar material.

The mini plant test work programme will be conducted at using an 8 t composite sample comprised of the remaining drill core sample from the variability drilling programme. The mini plant sample selection process will aim to target the expected life of mine ore blend and feed grade, however the exact sample composition can only be finalised during the sample selection process, based on the available core sample.

The mini pilot plant testing will focus on establishing and/or confirming the following:

- The effect of rougher flotation feed PSD for the pilot scale primary rod mill and secondary ball mill circuit as compared to the bench scale flotation performance using a laboratory rod mill.
- Determination of the optimal high chrome grinding media specification.
- Evaluate pilot scale flotation mass pull as compared to bench scale testing.
- Measuring the steady state flotation concentrate solids concentrations and confirm the expected cleaner circuit operating density.
- Evaluate the impact of scavenger cleaner tails recycle.
- Quantify the benefit of column flotation and establish the optimal location within the flowsheet.

- Evaluate the impact of reagent build up in recycle streams and optimise reagent addition requirements.
- Evaluate the effect of operating with process water sourced from Mokopane.
- Mach reactor scouting tests to evaluate potential opportunities for improved fines recovery.
- Generate tailings sample for confirmatory thickening test work on a classified feed sample, to confirm the design parameters and establish criteria for process guarantees.
- Generate sufficient cleaner concentrate for thickening and filtration test work, to confirm the design parameters and establish criteria for process guarantees.
- Generate tailings sample for backfill and tailings pumping pipe loop test work.
- Generate concentrate sample for Kell test work.

Shaft sinking is currently underway, and it is anticipated that a 3,000 t bulk sample generated from the reef intersection of Shaft 1 will be available in quarter one 2018. Likewise, the reef intersection of Shaft 2, at a later date will also generate a bulk sample of approximately 3,000 t. These bulk samples will only be available after completion of the mini pilot plant campaign, and no additional metallurgical testing of these samples is planned at this stage.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Platreef Mineral Resource model update includes three Mineral Resource estimates completed in 2015 and 2016:

- TCU Model (UMT-TCU) – The TCU Mineral Resources includes material within and adjacent to grade shells (3PE+Au) in the TCU. This Mineral Resource has been updated using the revised geological interpretation. Additional drilling has been completed in Zone 1, Zone 3 and Zone 5 (see Figure 7.1). The Mineral Resource amenable to selective underground mining methods is supported by the UMT-TCU model. Indicated and Inferred Mineral Resources were estimated for the UMT–TCU model.
- Bikkuri Model (UMT-BIK) – This consists of material within and adjacent to grade shells in the Bikkuri Reef. This Mineral Resource has been estimated using a revised geological interpretation and incorporation of additional drilling in Zone 1. The Bikkuri reef has also been identified in Zone 2. The Mineral Resources amenable to selective underground mining methods in the Bikkuri Reef are supported by the UMT-BIK model. Indicated and Inferred Mineral Resources were estimated for the UMT-BIK model.
- Footwall Model (UMT-FW) – The UMT-FW model includes material that is footwall to the UMT-TCU model. This Mineral Resource has been estimated using the revised geological interpretations and additional drilling in Zone 1. The UMT-FW Mineral Resources are potentially amenable to selective underground mining methods and possibly mass mining methods in local areas. Inferred Mineral Resources were estimated for the UMT–FW model.

The recognition of lithological controls (TCU stratigraphy) on grade has enabled declaration of Inferred Mineral Resources at wider drill spacings than would normally be possible. Infill drilling in Zone 1, Zone 2 and Zone 3 permitted the declaration of Indicated Mineral Resources in that portion of the Platreef Project area. Inferred Mineral Resources are declared in Zone 2, Zone 3 and the Madiba area (see Figure 14.1).

Additional drilling down-dip in Zone 3 and in the Madiba area permitted the expansion of the Inferred Mineral Resource in the UMT–TCU portion of the deposit. Additional down-dip/lateral potential could support estimation of additional Mineral Resources with further drilling. Revised geological interpretations decreased the extent of the TCU stratigraphy and decreased Inferred Mineral Resources in Zone 2.

The UMT–TCU deposit is the main focus of the Platreef Project underground mine development. The limits of the UMT-TCU model are shown in Figure 14.1. The UMT-TCU Mineral Resources are located in Zone 1, Zone 2, Zone 3 and the Madiba Area (See Section 14.2.11).

The UMT–BIK Mineral Resources are located in Zones 1 and 2. The UMT-BIK resource model is located stratigraphically above the UMT–TCU resource model (Figure 14.2).

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

Figure 14.1 Mineral Resource Area for the UMT-TCU

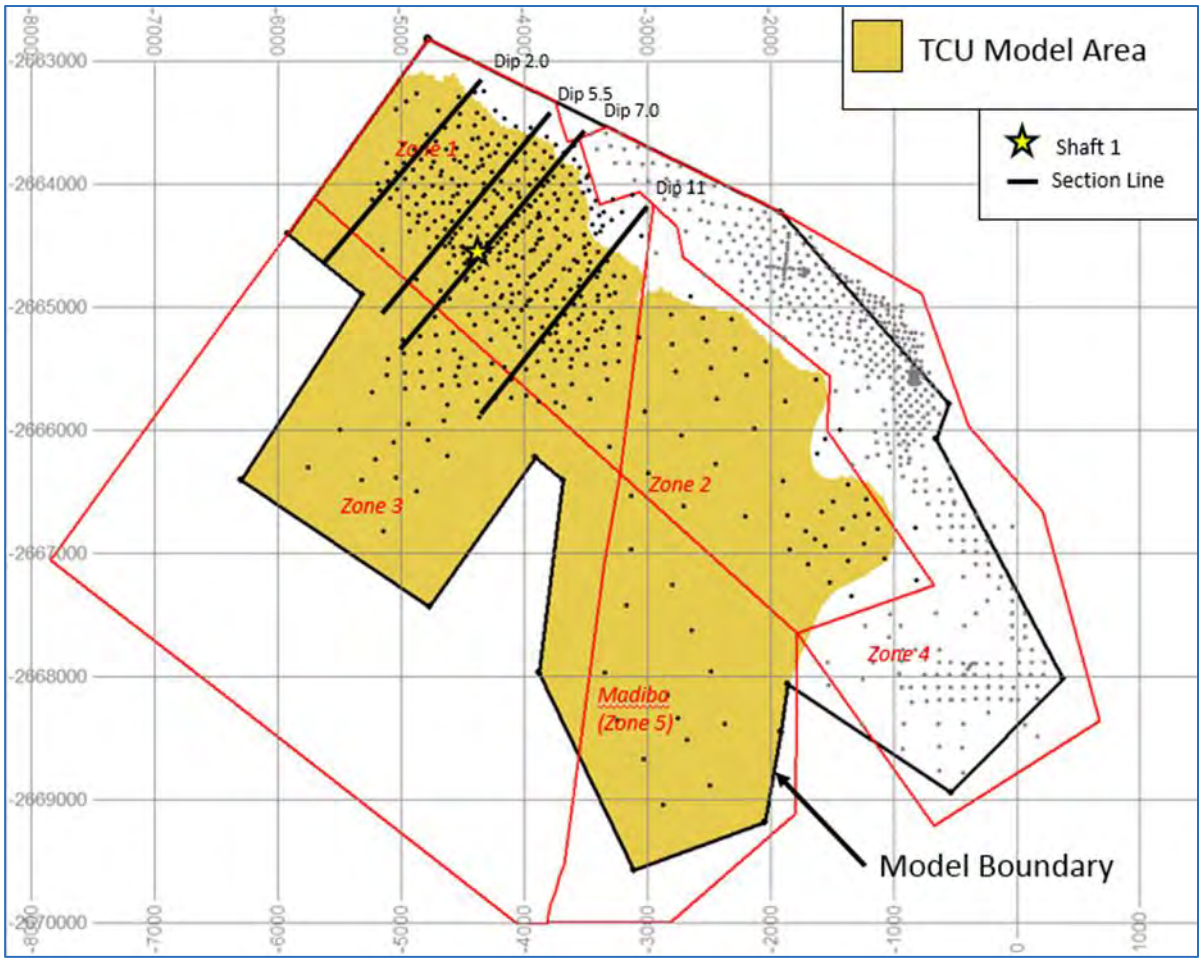
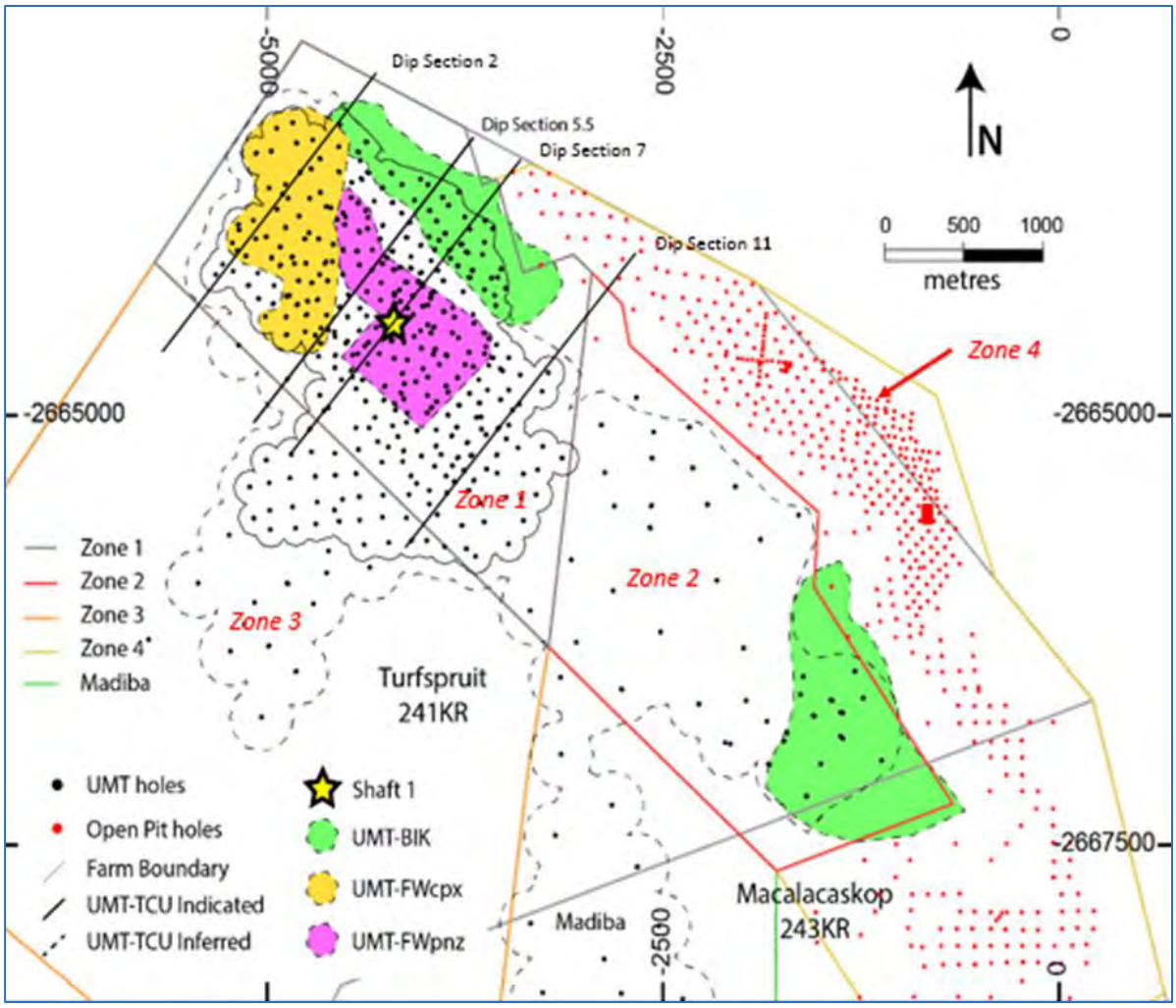


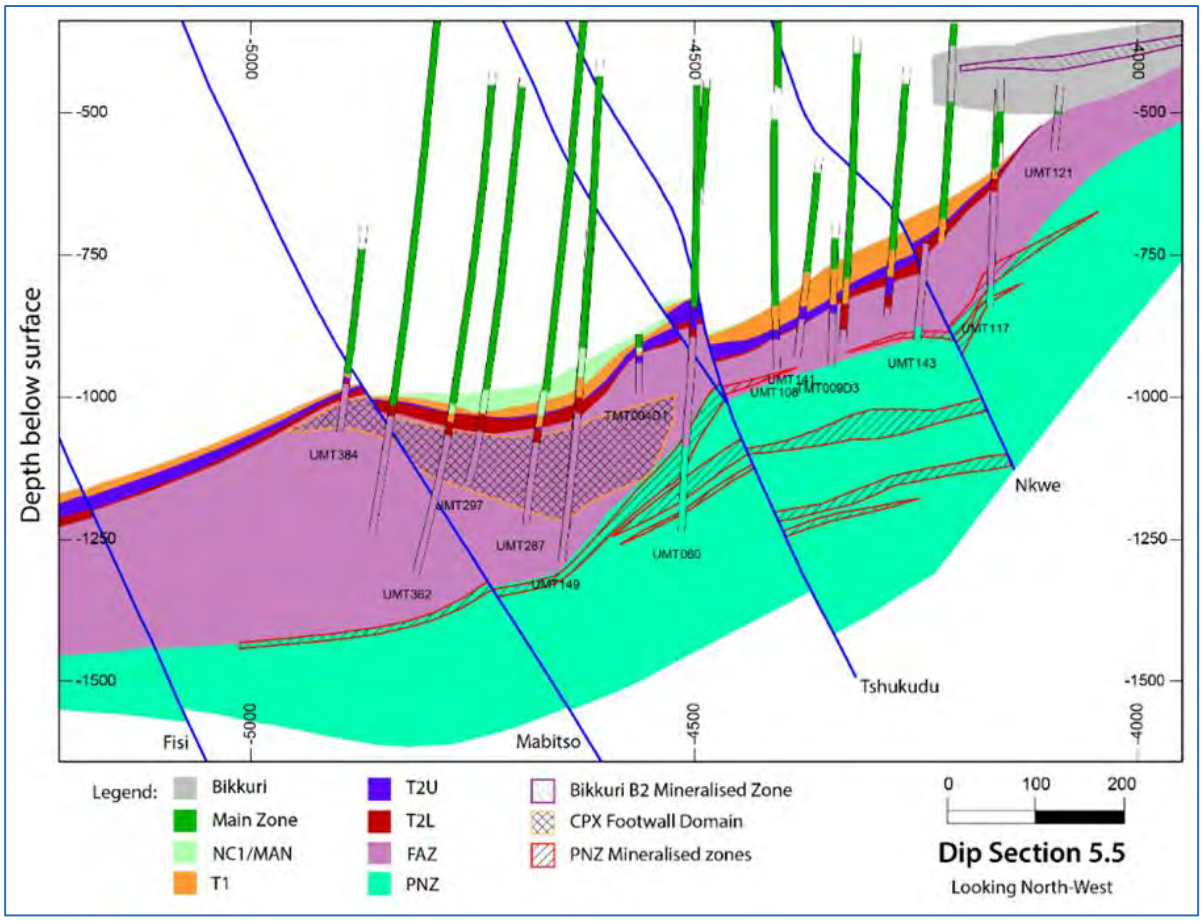
Figure prepared by Amec Foster Wheeler, 2016.

Figure 14.2 Mineral Resource Areas for the UMT-Bikkuri (BIK) and UMT-FW



Courtesy Ivanhoe, 2016.

Figure 14.3 Dip Section 5.5 Showing Relationship of Bikkuri, TCU and FW Models



Courtesy 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)

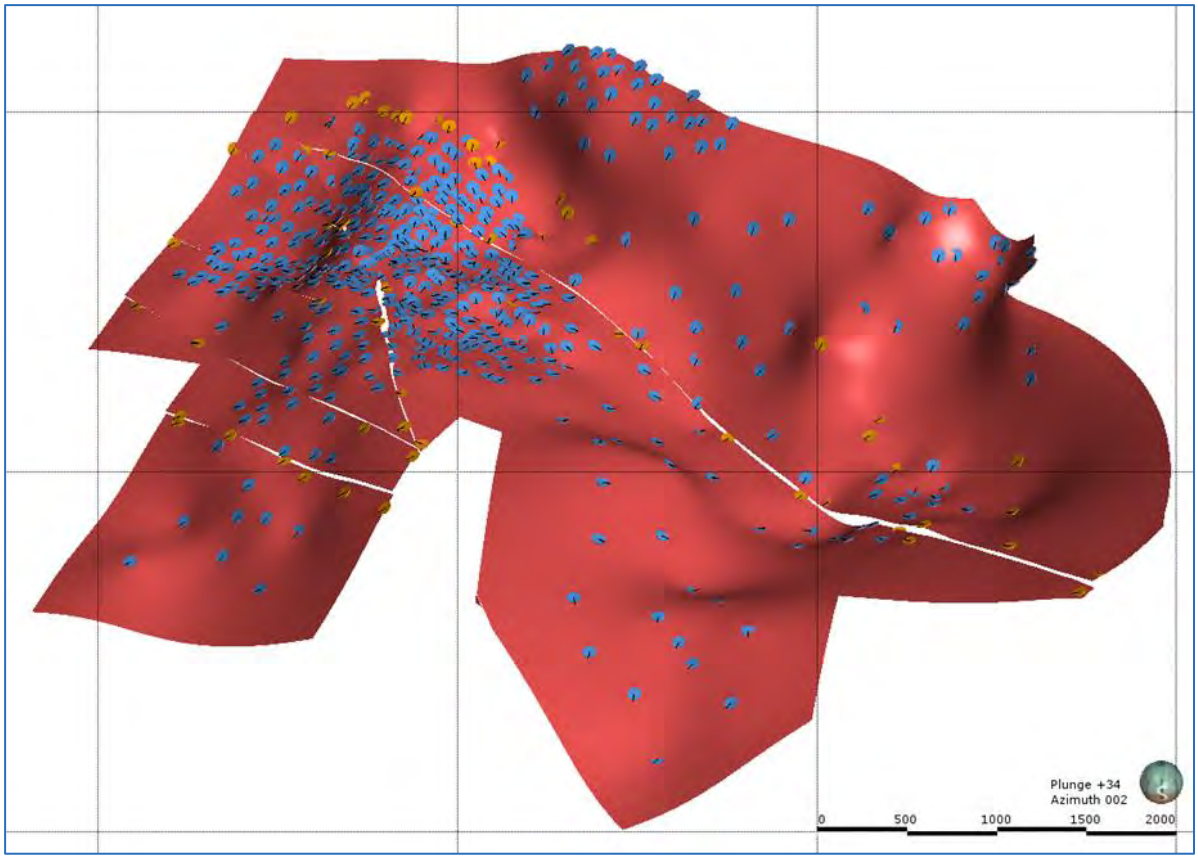
Relogging of drill core, infill drilling in Zones 1 and 2, expansion drilling in the Madiba area and a revised structural model since 2013 has resulted in modifications incorporated in the 2016 geology model.

The revised structural model identified or inferred numerous faults. Only faults with a high degree of confidence were used for the geology model. These include the Nkwe, Tshukudu, Tau, Mabitso, Fisi and Tlou faults (see Figure 7.14).

Mineralisation in the southern portions of Zone 2 that is stratigraphically above the TCU has been interpreted as similar to the Bikkuri Reef in Zone 1. Both areas are included in the UMT-BIK model (Figure 14.2).

The relogging of drill core in the footwall of the TCU identified the Footwall Assimilation Zone (FAZ) that includes the CPX (clinopyroxenite) domain and the underlying Pyroxenite-Norite Zone (PNZ). The CPX and PNZ domains were estimated separately and comprise the UMT-FW model (Figure 14.3).

Figure 14.4 T1 Reference Surface



Courtesy Ivanhoe, 2016. Blue discs are drillhole intersections. Orange discs are control points. Black lines on discs show dip direction.

Table 14.1 Strat and MCODE Description

Strat Unit	STRAT	MCODE
Main Zone in Bikkuri	MZBK	10
NC1 in Bikkuri area	NC1BK	11
Mottled Anorthosite in Bikkuri	MANBK	12
B1	B1	13
B2	B2	14
NC2 in Bikkuri	NC2BK	15
Lower Zone in Bikkuri	LZ1BK	16
Main Zone	MZ	20
Norite Cycles 1	NC1	21
Mottled Anorthosite	MAN	22
T1	T1	23
T2U	T2U	24
T2L	T2L	25
Norite Cycles 2	NC2	26
UG2 Hanging Wall	UG2HW	27
UG2	UG2	28
UG2 Footwall	UG2FW	29
Footwall Assimilated Zone	FAZ	30
Pyroxenite Norite Zone	PNZ	31
Lower Zone	LZ	32
FW	FW	33

14.1.2 Mineralised Zones

Nested grade shells were used to constrain the grade estimation in the UMT-TCU and UMT-BIK resource models. Nested grade shells were made for the TCU T1 mineralised zone (T1MZ) and T2 mineralised zone (T2MZ). Nested grade shells in the BIK model were identified for the B1 (B1MZ) and B2 (B2MZ) stratigraphic units. The nested grade shells were constructed using 1 g/t, 2 g/t and 3 g/t 3PE+Au. Mineral Zones (MZ) were identified in the FW-PNZ domain and were used to constrain higher grade mineralisation. The grade shells and mineral zones were validated on cross-sections to ensure consistency.

14.2 UMT-TCU Resource Model

The UMT-TCU model is the main focus of the Platreef Project and is considered amenable to selective underground mining methods. The UMT–TCU resource model update was limited to that portion of the UMT area that includes the TCU stratigraphic sequence. The limits of the UMT–TCU area Mineral Resource estimate are shown in Figure 14.1. The UMT–TCU model includes a densely-drilled area in Zone 1 and less densely drilled areas in Zones 2, and 3 and the Madiba area.

In the discussion which follows, some tables include information for the UMT-BIK and UMT-FW models so as to avoid the need for duplicate presentation of model parameters.

14.2.1 Drillhole Data

Only valid drillholes from the UMT drill programme were used for the grade estimation of the UMT-TCU mineral resource. The cut-off date for the drillhole database used for the Mineral Resource estimate was 24 July 2015.

Drillholes were considered as not valid when:

- Drillhole was abandoned prior to intersecting the mineralised zones.
- Drillhole Intersected a mineralised zone that was interpreted as not representative due to significant faulting.
- Wedge holes.

Wedges off the parent hole were commonly drilled to intersect the T1MZ and T2MZ, but locally drillholes targeted deeper footwall mineralisation. Wedges were primarily drilled for metallurgical purposes resulting in a cluster of wedges around the parent hole. An analysis was completed to determine the possibility of merging wedge holes (for wedges with available assay data) with their parent drillholes and thus to provide a single intercept per cluster of wedge holes. The analysis determined that differences in mineralised thicknesses between the wedges and parent hole caused unequal alignment of the mineralised zones and thus a smearing and smoothing of grades. The decision was taken to exclude wedge holes and only use the first intercept of the mineralised zones in any case where multiple intersections from a single parent hole occurred.

14.2.2 TCU Geology Model

The TCU Mineral Resource model occurs within the stratigraphic sequence referred to as the Turfspruit Cyclic Unit discussed in Section 7.2.7.

The geology model for the UMT-TCU was created in Leapfrog using two dimensional gridded seam models constructed in Datamine for control. The top surface of the T1 feldspathic pyroxenite was used as a reference surface (see Figure 14.4). The reference surface was used to construct the stratigraphic surfaces above and below the T1 reference surface. Control points were added near faults. Each stratigraphic unit was assigned a MCODE used in coding the drillhole composites and block model (see Table 14.1). The Nkwe, Tshukudu, Tau, Mabitso, Fisi and Tlou faults were used to divide the UMT-TCU model into seven structural blocks.

14.2.3 Model Envelope

The UMT-TCU model was constrained laterally within a polyline. The north and eastern limits were defined as the limit of recognisable TCU cyclical stratigraphy. Elsewhere the boundary was extended approximately 450 m beyond the drillhole data (see Figure 14.1). The UMT-TCU model was also constrained vertically by an envelope defined by surfaces controlled by the geological stratigraphy. The upper surface was defined as 20 m above the top of the T1MZ. The lower surface of the model envelope was defined as 75m below the base of the T2L. The UMT-TCU model does not extend above the 650 m elevation.

14.2.4 High-Grade Shells – UMT- TCU

Nested grade shells were constructed for the T1 mineralised zone (T1MZ) and the T2 mineralised zone (T2MZ) to constrain the grade estimation. The nested grade shells were identified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE (Pt+Pd+Rh) +Au cut-offs. The stratigraphic location of the mineralised intercepts was considered in the grade shell construction. The grade-shell intercepts were coded into the drillhole database. The grade-shell drillhole intercepts were validated on dip and strike sections to ensure consistency. The grade-shell drillhole intercepts were used to construct wireframes of the nested grade shells using Leapfrog and Datamine functions. Grade-shell codes (GCODES) were used to code blocks within and outside the grade shells. The GCODES are summarised in Table 14.2.

14.2.4.1 T1MZ

The T1MZ is interpreted to transgress the T1-NC1 stratigraphic boundary in the southern portions of Zone1 and into Zone 2 (see Figure 14.5). This transgression is localised and occurs in response to thickening of the T1 and NC1 units and the development of weak cyclicity within the T1. This relationship suggests the T1 is an undifferentiated portion of the cyclical units developed below the base of the Main Zone. Where the T1 and NC1 units are thinned, the T1MZ cannot be readily identified, and these areas have been excluded from the T1MZ model (see Figure 14.6).

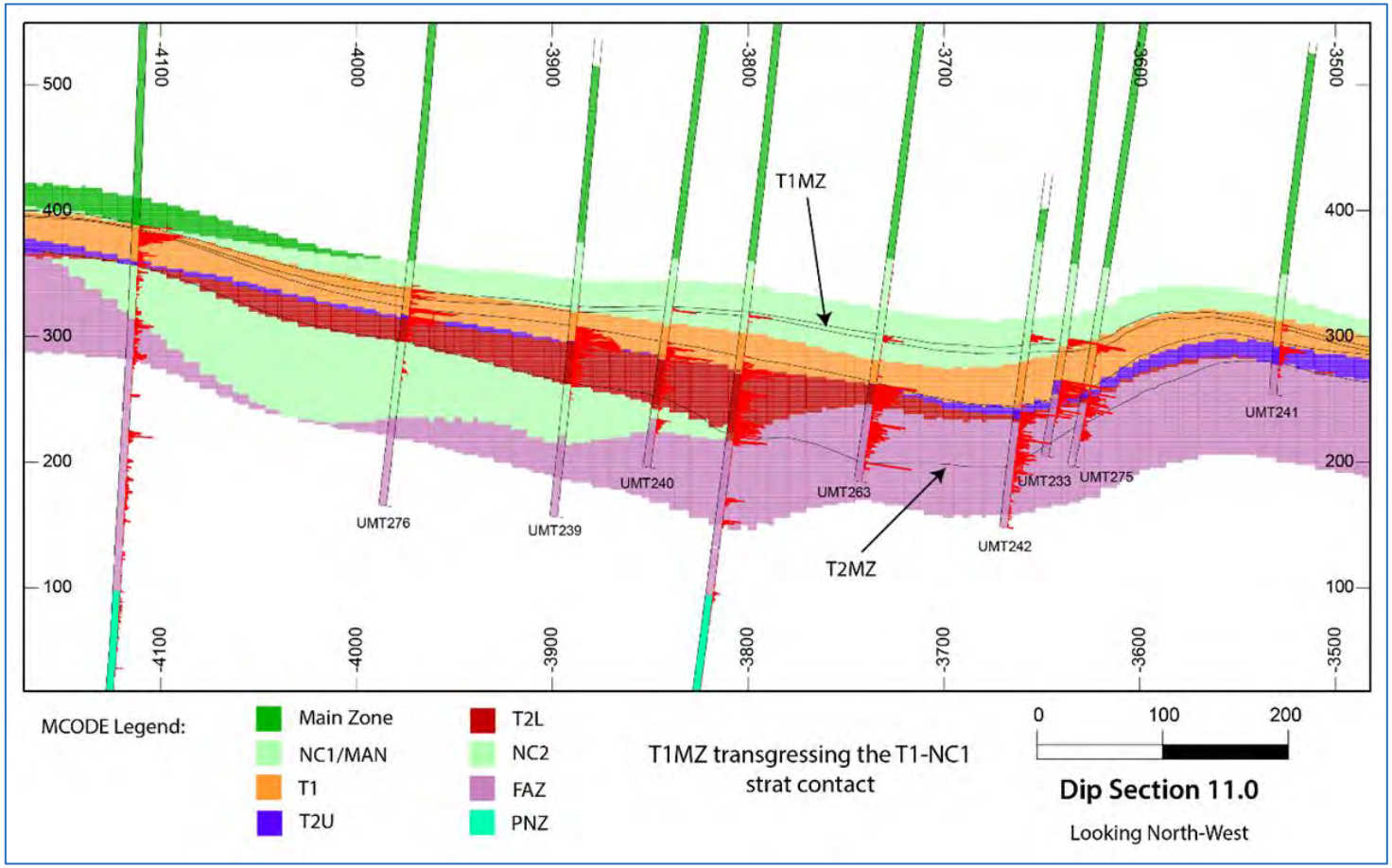
14.2.4.2 T2MZ

The T2MZ is defined by 3PE+Au assays and commonly occurs in the T2 stratigraphic unit of the TCU; however, the nested grade shells are not restricted to specific stratigraphic horizons and may transgress locally into the T1 or FW (see Figure 14.5).

14.2.5 Mineralisation Adjacent to the TCU Mineralised Zones

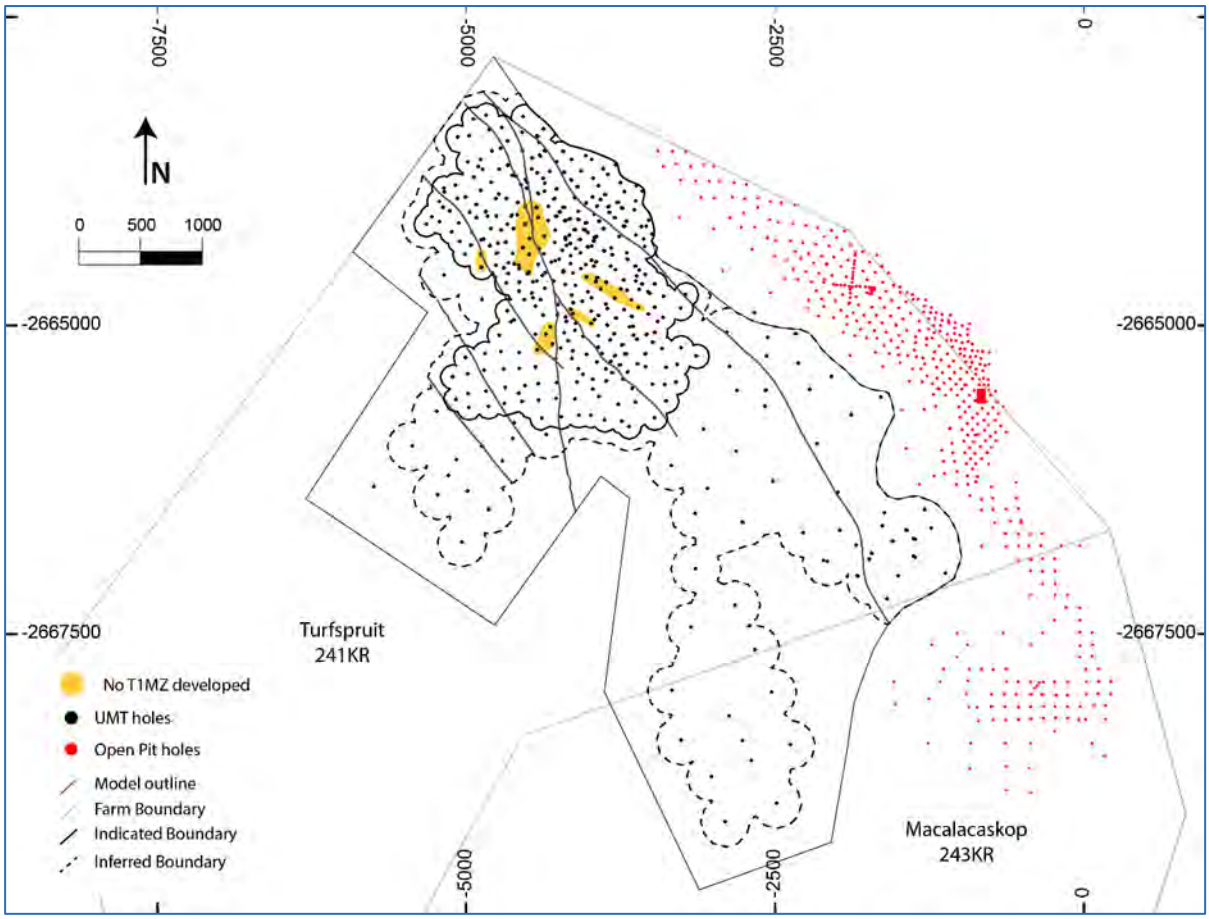
There is scattered mineralisation adjacent to the TCU mineralised zones that is locally continuous. Floating stope software is used in mining-related studies, and mineralisation adjacent to the TCU mineralised zones can be included in the resultant stopes; hence there is a need to estimate grades in blocks in an envelope around the T1MZ and T2MZ zones. Table 14.2 summarises the GCODES adjacent to the T1MZ and T2MZ mining zones within the TCU model envelope.

Figure 14.5 T1MZ Transgressing the Boundary Between the T1 and NC1 (Dip Section 11.0)



Courtesy Ivanhoe, 2016. Location of Dip 11 is shown on Figure 14.2,

Figure 14.6 Areas Where the T1MZ is Not Developed



Courtesy Ivanhoe, 2016.

Table 14.2 Summary of GCODES For TCU and Bikurri (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

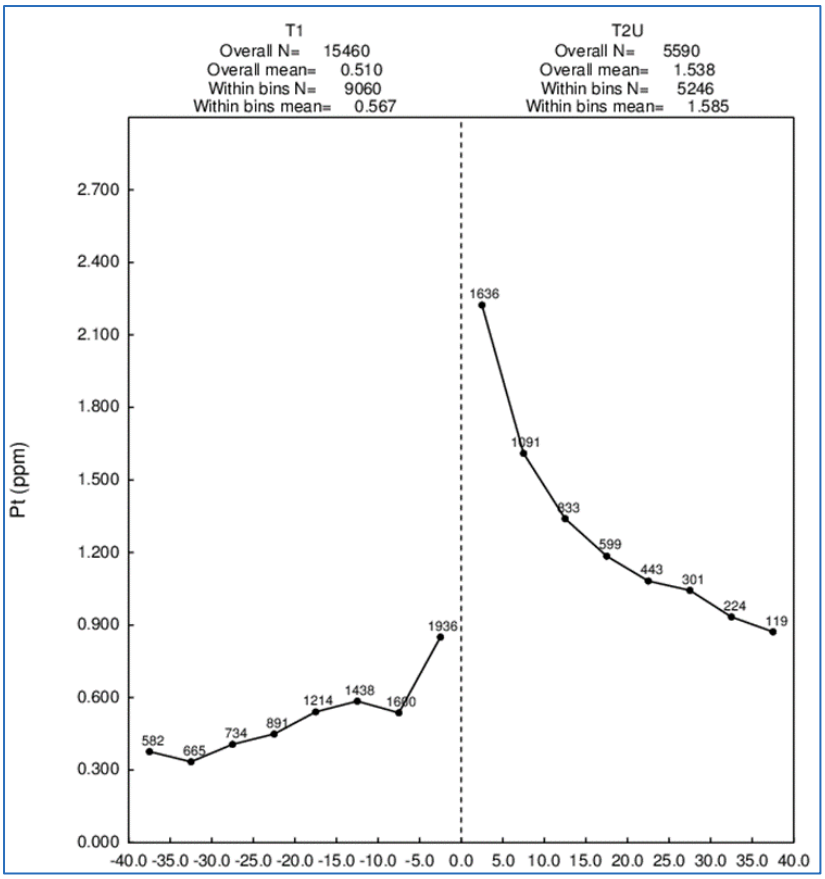


Figure prepared by Amec Foster Wheeler, 2015.

14.2.6.1 Rhodium Regressions

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

Figure 14.8 and Figure 14.9 show rhodium as a function of platinum regression for the T1 and T2 respectively. Table 14.3 summarises the proportions of assays with rhodium analysis within the grade shells and by stratigraphic unit. Table 14.4 summarises the lithology groups. The proportion of rhodium assays exceeds 50% within the 2 g/t 3PE+Au shell and exceeds 60% in the in T2U and T2L. Table 14.5 summarises the number of missing Rhodium analyses by grade shell.

Figure 14.8 Rhodium Regression for The T1

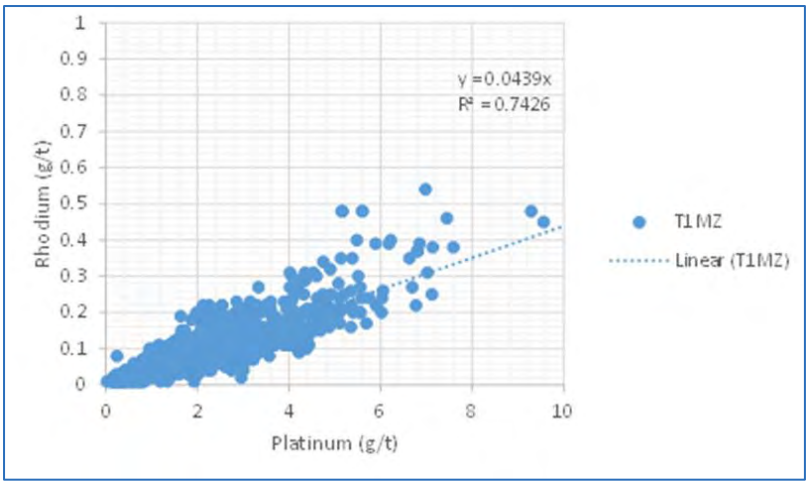


Figure prepared by Amec Foster Wheeler, 2015.

Figure 14.9 Rhodium Regression for The T2

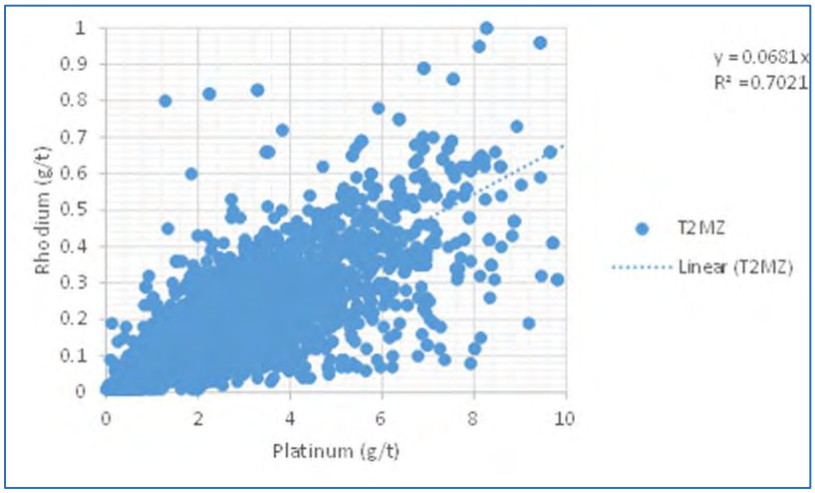


Figure prepared by Amec Foster Wheeler, 2015.

Table 14.3 Proportions of Rhodium Assays by Strat Code and 3PE+Au Grade Shell

Inside Mineralised Zones									
Zone	Regression Equation	No. Data for Equations	Average Rh Value	Average Pt Value	Avg Rh/ Avg Pt Ratio	No. Data without Rh Values	Aver Rh Value After Eqn	Average Pt Value	Avg Rh/ Avg Pt Ratio
T1MZ	Rh=0.0439 Pt	2080	0.079	1.766	0.0445	1049	0.023	0.519	0.0439
T2MZ	Rh=0.0681 Pt	13526	0.133	1.874	0.0708	5387	0.043	0.634	0.0681
B1MZ	Rh=0.0437 Pt	47	0.052	1.163	0.0450	26	0.030	0.694	0.0437
B2MZ	Rh=0.0519 Pt	601	0.067	1.248	0.0533	281	0.028	0.534	0.0519
Outside Mineralised Zones									
Zone	Regression Equation	No. Data for Equations	Average Rh Value	Average Pt Value	Avg Rh/ Avg Pt Ratio	No. Data without Rh Values	Aver Rh Value After Eqn	Average Pt Value	Avg Rh/ Avg Pt Ratio
Group 1	Rh=0.0721 Pt	59	0.032	0.418	0.0766	4171	0.002	0.034	0.0720
Group 2	Rh=0.0585 Pt	1586	0.032	0.599	0.0534	18172	0.006	0.104	0.0585
Group 3	Rh=0.0630 Pt	582	0.047	0.619	0.0763	3440	0.015	0.232	0.0630
Group 4	Rh=0.0614	4045	0.061	1.081	0.0566	42932	0.061	0.242	0.2539
Group 5	Rh=0.1102 Pt	125	0.099	0.898	0.1102	179	0.029	0.260	0.1102
Group 6	Rh=0.0691	545	0.069	1.811	0.0382	22194	0.069	0.105	0.6571

Table 14.4 Group Definitions Table Caption

Group	Strat Unit
1	MZ, MZBK, MAN, MANBK
2	BAR, B1, T1, NCBK, NC1
3	B2 BBK, T2U, T2L, NC2
4	FAZ, FAZBK, UGHW
5	UG2, UG2FW
6	HFR, PNZ, TVL

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

MinZone	Grade Shell (3PE+Au)	Number Sample Inverals	Samples Intervals with Rh Analysis	Samples Intervals Missing Rh Analysis	% Intervals Missing Rh Analysis
B1MZ	1 g/t	16	0	16	100
	2 g/t	8	0	8	100
	3 g/t	130	33	97	75
B2MZ	1 g/t	746	293	453	61
	2 g/t	231	162	69	30
	3 g/t	535	414	121	23
T1MZ	1 g/t	1,026	453	573	56
	2 g/t	303	239	64	21
	3 g/t	1,939	1,406	533	27
T2MZ	1 g/t	6,383	2,781	4,057	59
	2 g/t	5,006	3,515	1,491	30
	3 g/t	8,417	7,812	605	7

14.2.7 Variography

Pair wise relative variograms were completed by grade shell. Figure 14.10 shows a downhole pairwise relative variogram for Platinum. Figure 14.12 shows a directional pairwise relative variogram for Platinum (aziumuth 137, dip 0).

Figure 14.10 Downhole Pair Wise Relative Variogram for Platinum

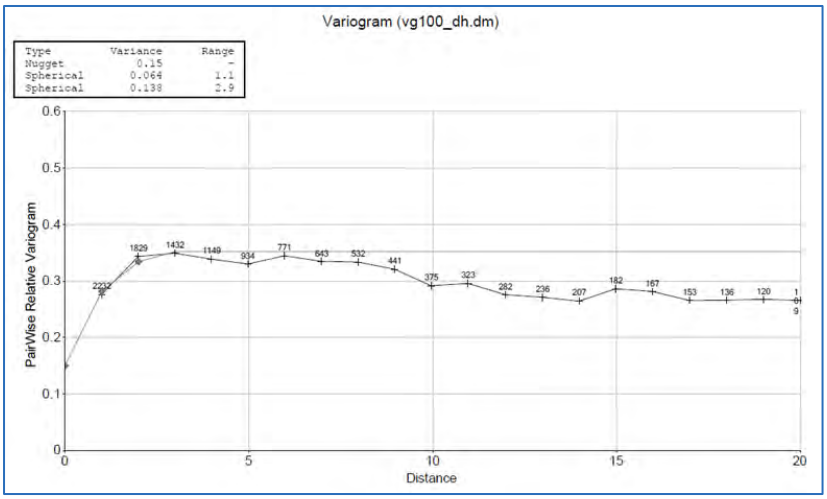


Figure prepared by Amec Foster Wheeler, 2015. Lag distances are in metres.

Figure 14.11 Directional Pair Wise Relative Variogram Model for Platinum (Az=135)

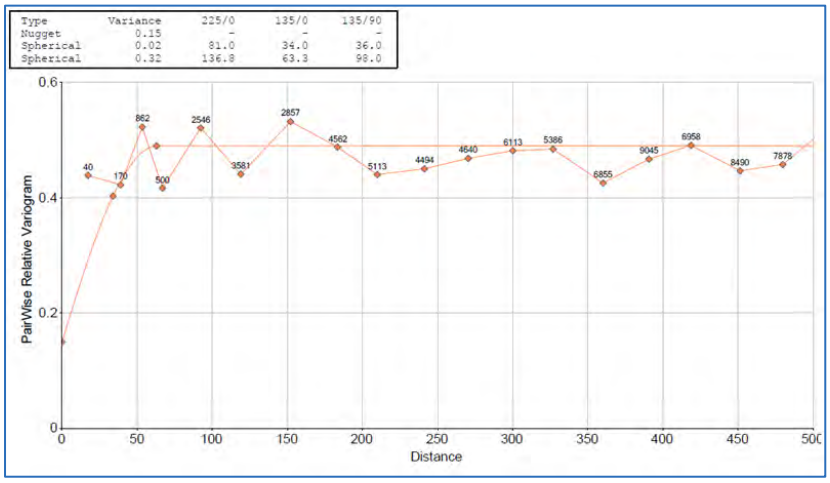


Figure prepared by Amec Foster Wheeler, 2015. Lag distances are in metres.

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 T1MZ

The grade estimation in the T1MZ included block and drillhole composite matching by a combination of MCODE and GCODE to ensure that stratigraphic components of the T1MZ were estimated separately.

14.2.9.2 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.

14.2.9.3 Blocks Outside Grade Shells

Grade estimation for blocks not located within the nested grade shells were estimated by matching blocks and composites by MCODE.

14.2.9.4 Grade Capping and Outlier Restriction

An outlier restriction distance threshold of 15 m was applied to high-grade composites within each stratigraphic unit and mineralised zone. The grade thresholds for outliers were selected from the histograms and probability plots of 1 m drillhole composites; thresholds are summarised in Table 14.8 and Table 14.9. Composites with grades above the grade threshold and with distances from composite to block centre beyond the distance thresholds were not used in grade estimation.

Table 14.8 Outlier Restriction Thresholds for Stratigraphic Units (MCODE)Table Caption

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZ	0.40	0.50	0.10	0.05	0.22	0.30	2.0
NC1/MAN	0.60	1.00	0.25	0.05	0.40	0.25	1.0
T1	2.00	1.20	0.60	0.06	0.50	0.35	2.5
T2	1.50	2.20	0.35	0.10	0.50	0.35	2.0
NC2	1.50	1.10	0.30	0.10	0.30	0.20	-
UG2HW	2.00	1.20	0.40	0.15	0.35	-	-
UG2	-	-	-	-	-	-	-
UG2FW	-	-	-	-	-	-	-
FAZ	1.60	2.50	0.60	0.15	1.00	0.60	5.0
PNZ	0.25	0.50	0.06	0.15	1.00	0.65	10.0

Table 14.9 Outlier Restriction Thresholds for Mineralised Zones (GCODE)

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
T1MZ	7.5	10.0	-	-	-	-	5.0
T2MZ 1g/t	4.0	4.0	0.7	0.25	1.0	0.5	4.0
T2MZ 2g/t	5.0	10.0	-	-	-	-	-
T2MZ 3g/t	9.0	10.0	2.0	-	-	-	4.0
B1MZ							
B2MZ							

14.2.9.5 Unestimated Blocks

Blocks that were not estimated were assigned a default grade determined as the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.10. Unestimated blocks were generally located at the extremities of the block model. Unestimated blocks within the FW stratigraphy were found to be located in areas of wide-spaced drilling and were not assigned an average grade.

Table 14.10 TCU Mean Grades by Stratigraphic Unit

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZ	0.001	0.001	0.001	0.001	0.010	0.010	0.010
NC1	0.056	0.051	0.034	0.003	0.049	0.015	0.096
MAN	0.041	0.034	0.014	0.003	0.012	0.006	0.047
T1	0.212	0.160	0.073	0.012	0.103	0.041	0.244
T2	0.344	0.432	0.066	0.023	0.164	0.078	0.423
NC2	0.261	0.255	0.046	0.017	0.097	0.042	0.227
UG2HW	0.796	0.798	0.13	0.078	0.160	0.790	0.397
UG2	1.229	0.661	0.078	0.161	0.236	0.135	0.680
UG2FW	0.390	0.460	0.071	0.040	0.195	0.118	0.704
LZ1	0.341	0.419	0.067	0.061	0.146	0.080	0.533
LZ2	0.195	0.250	0.040	0.069	0.106	0.061	0.474
T1MZ 1g	0.660	0.571	0.202	0.039	0.255	0.156	0.837
T1MZ 2g	1.088	0.947	0.309	0.057	0.255	0.156	0.837
T1MZ 3g	1.709	1.387	0.420	0.092	0.358	0.166	0.887
T2MZ 1g	0.670	0.765	0.118	0.059	0.225	0.115	0.667
T2MZ 2g	1.059	1.187	0.177	0.083	0.278	0.141	0.786
T2MZ 3g	2.150	2.258	0.322	0.156	0.383	0.190	0.988

14.2.10 Bulk Density

Bulk density was assigned to stratigraphic units using the mean density for each unit (Table 14.11). Whilst some stratigraphic units are comprised of a number of different lithologies (the NC1 and NC2 cyclical units for example), in general, the variability in density values is considered low.

Table 14.11 Density Values by Stratigraphic Code

Zone	Number of Samples	Mean	CV
MZ	17,368	2.90	0.03
NC1/MAN	1,184	2.95	0.05
T1	2,387	3.18	0.03
T2U	787	3.19	0.04
T2L	793	3.04	0.05
NC2	292	3.05	0.06
UG2HW	43	3.10	0.08
UG2	3	3.50	0.01
UG2FW	33	3.17	0.04
FAZ	6,453	3.11	0.05
PNZ	4,001	3.09	0.06
LZ	534	3.14	0.07

14.2.11 Mineral Resource Classification

Mineral Resources have been classified using the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014):

"A Mineral Resource is a concentration or occurrence of solid material, of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify, geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail, to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve."

Classification is determined both laterally and vertically. A drillhole spacing study was completed in 2013 (Parker and Kuhl, 2013) to review the classification parameters used at Platreef. The study was based on the 2013 structural and geology models. The study concluded that the existing 100m drill grid required for Indicated in the 2013 resource could be expanded to 150m, 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 200m grid resulting in a 150m spacing between holes.

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

A triangulated surface was constructed to define the boundary from a drillhole spacing suitable for Indicated Mineral Resources to the wider drillhole spacing below the TCU suitable for Inferred Mineral Resources. This surface was used to define a boundary between Indicated and Inferred Mineral Resources. A similar methodology was applied in the Madiba area where the drillhole spacing below the T2MZ is too wide to support Inferred Mineral Resources and the block model is unclassified.

14.2.12 UMT–TCU Model Classification

The Mineral Resource Classification for the TCU model is shown in Figure 14.12. No Measured Mineral Resources are declared. Indicated Mineral Resources are declared where closer spaced drilling has been completed (Predominantly Zone 1). Inferred Mineral Resources are declared where the drillhole spacing is 400 m to 800 m (predominately Zone 2, Zone 3 and Madiba area). Inferred Mineral Resources are also declared in Zone 1 below the TCU where drillhole spacing increases. The Inferred Mineral Resources are permitted at a wider drillhole spacing than would normally apply because of the well defined geology of the TCU. Figure 14.13 displays the regions of Indicated and Inferred Mineral Resources on Dip Section 7.0.

14.2.13 UMT–TCU Model Validation

Model validation included blocks classified as Indicated Mineral Resources and included visual inspection of block grades relative to composite grades on cross-sections and level plans. Statistical comparisons consisting of box plots and grade profiles tabulated in different directions (swaths) for each metal by stratigraphic unit and 3PE+Au grade shell were constructed to compare the ID3 grade estimates, OK estimate, NN estimates, and 1 m composites.

14.2.13.1 Visual Inspection

Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long-sections and level plans. In general, the block grades honoured the composite grades. Representative cross-sections for 3PE+Au are presented in Figure 14.4 (Dip Section 7.0) and Figure 14.16 (Dip Section 2.0). Representative cross-sections for Ni are presented in Figure 14.15 (Dip Section 7.0) and Figure 14.17 (Dip Section 2.0).

Figure 14.12 Mineral Resource Classification for The TCU Model

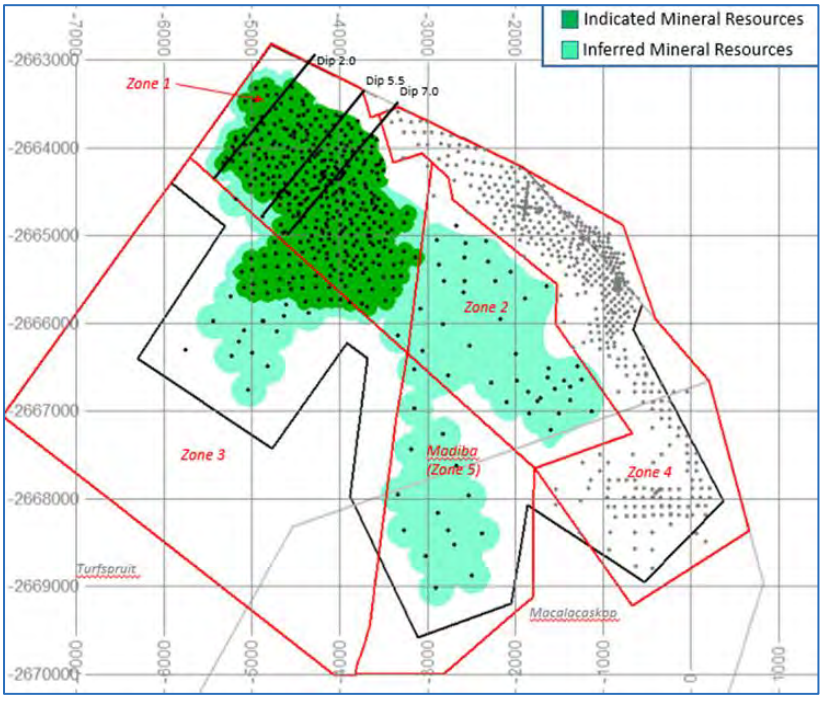
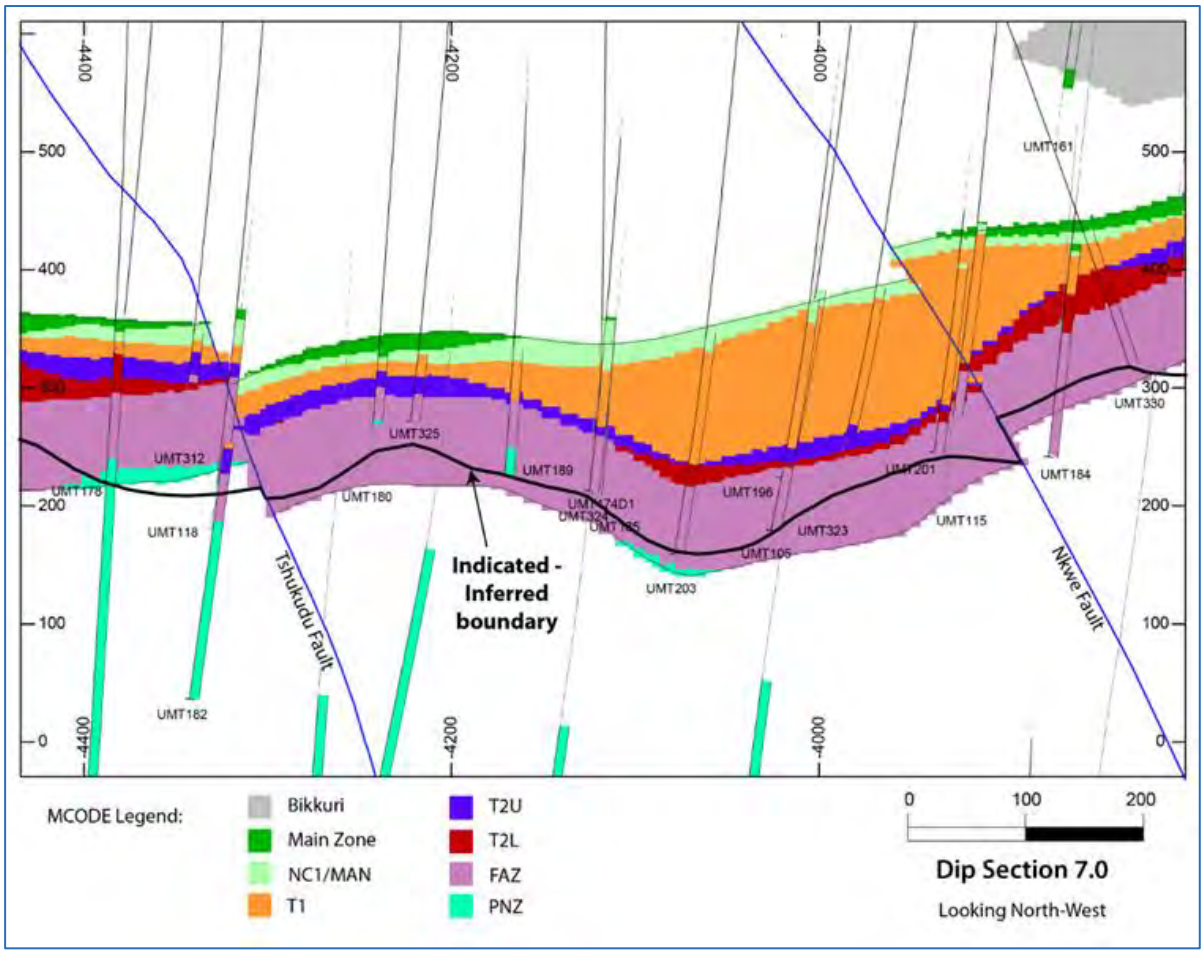


Figure prepared by Amec Foster Wheeler, 2016.

Figure 14.13 Dip Section 7.0; Constraints on Mineral Resource Classification



Courtesy Ivanhoe, 2016. Location of Dip 7 is shown on Figure 14.2.

Figure 14.14 Dip Section 7.0 Showing 3PE+Au (50 m Drillhole Projection) Looking North-West

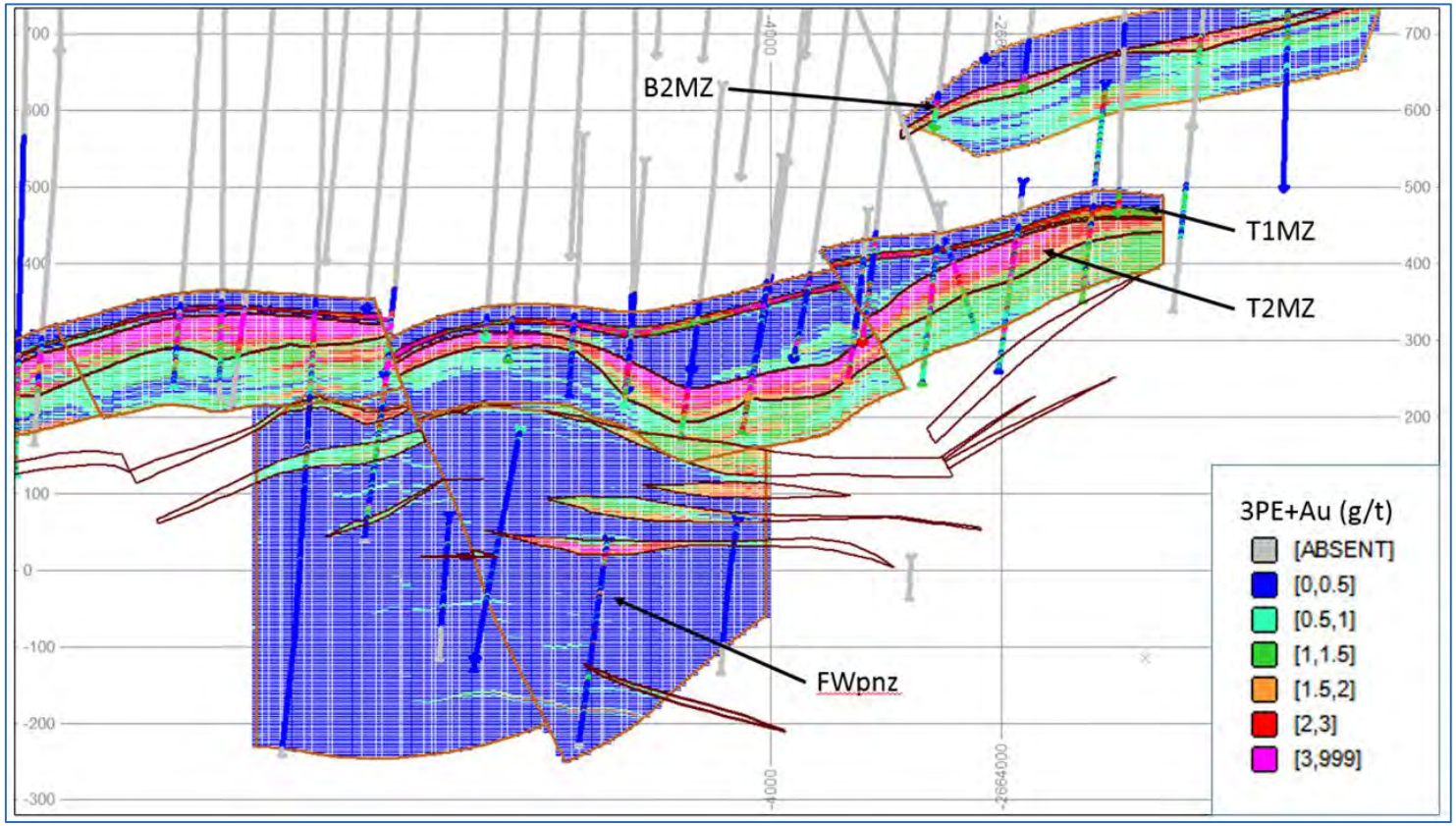


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 7 is shown on Figure 14.2.

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

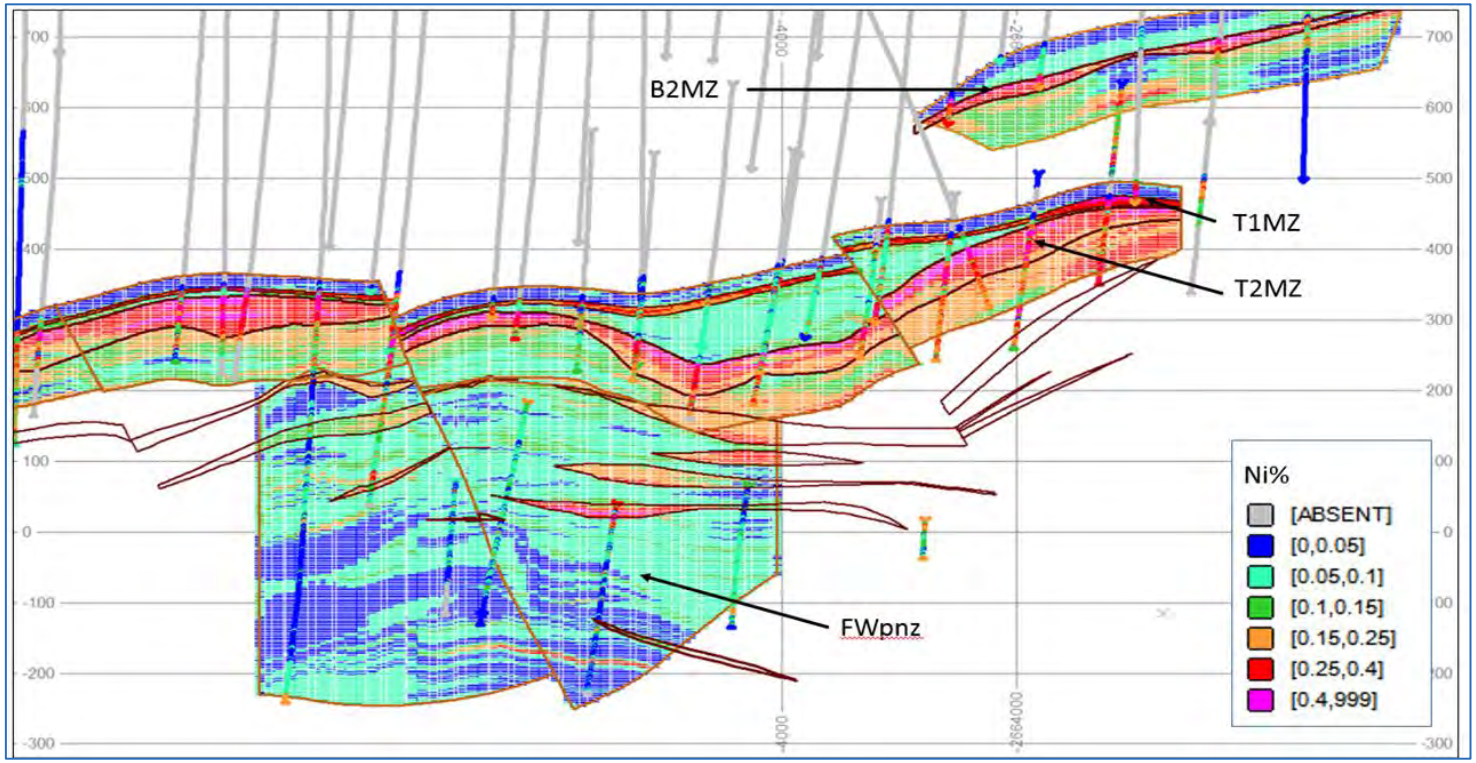


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 7 is shown on Figure 14.2.

Figure 14.16 Dip Section 2.0 Showing 3PE+Au (50 m Drillhole Projection) Looking North-West

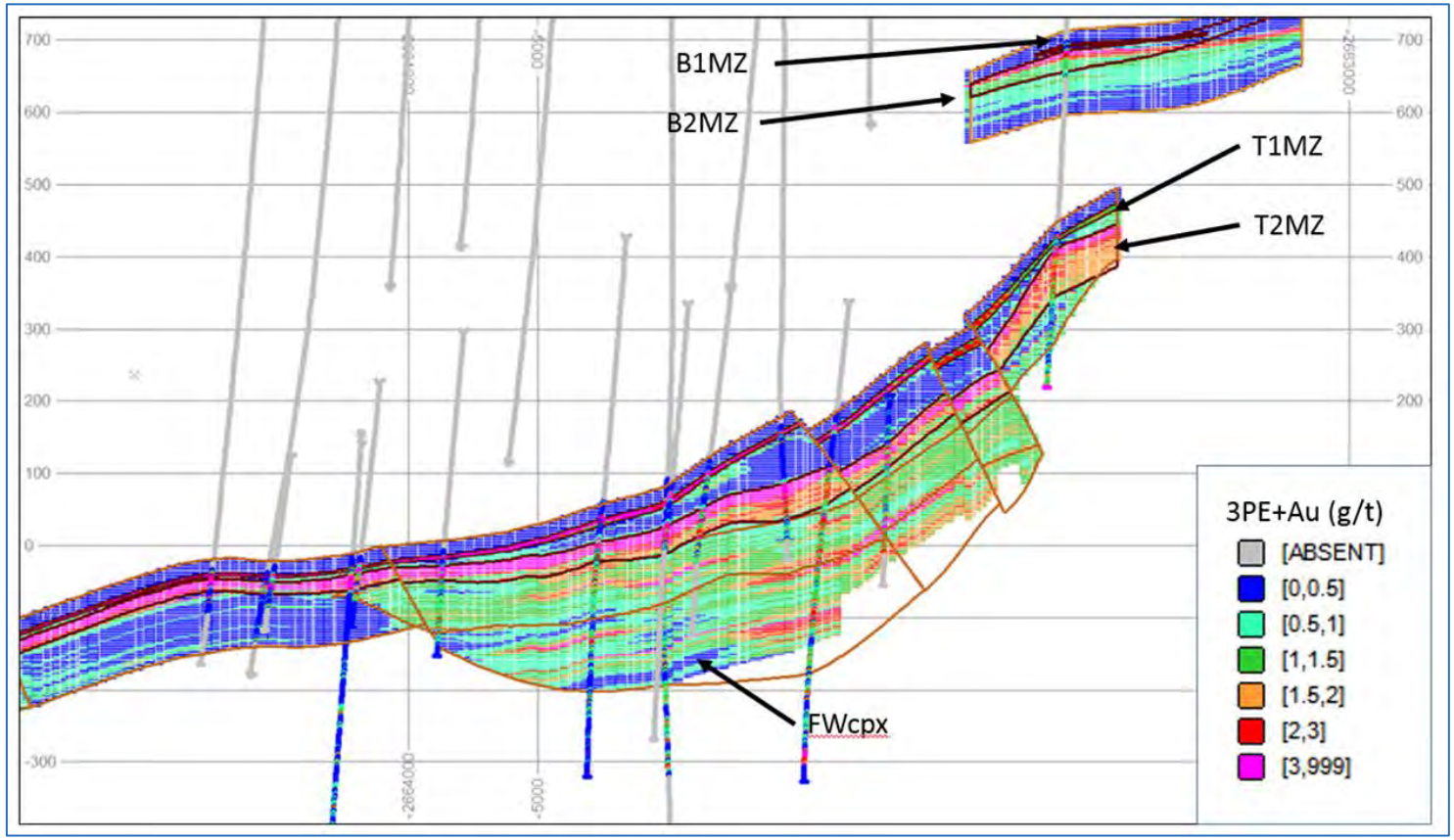


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 2 is shown on Figure 14.2.

Figure 14.17 Dip Section 2.0 Showing Ni% (50 m Drillhole Projection) Looking North-West

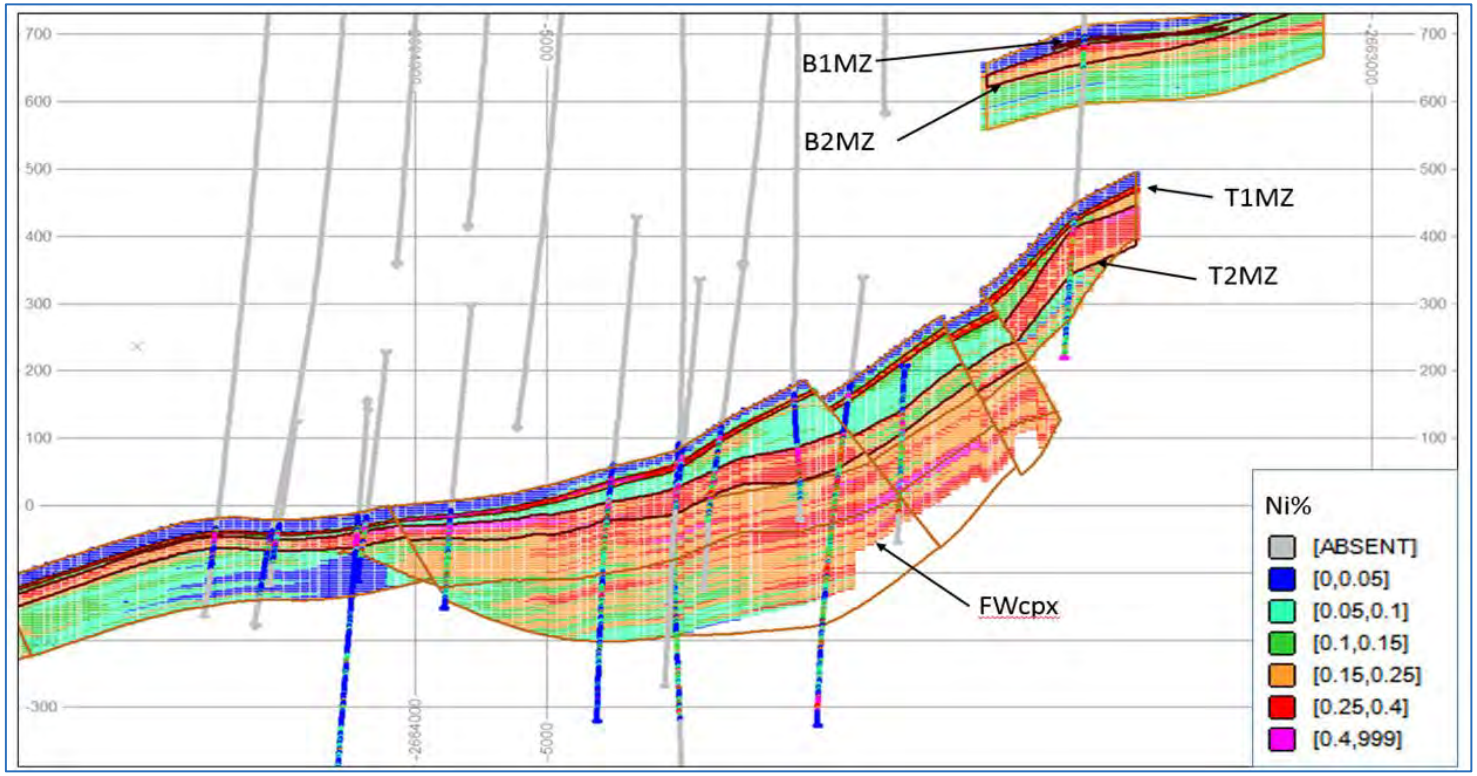


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 2 is shown on Figure 14.2.

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 and is commonly used by Amec Foster Wheeler as, when correctly implemented, it is statistically unbiased to aid in the validation of grade estimates. Amec Foster Wheeler checked the Pt, Pd, Au, Ni, Cu and S resource models for global and local bias.

The checks for global bias were performed by comparing the ID3 average grade (with no cut-off) with NN estimates by mineralised unit. Blocks reviewed were restricted to those classified as Indicated Resources. Domains with a global bias outside the recommended Amec Foster Wheeler guidelines of ±5% (relative) are highlighted (see Table 14.12).

The grade shells for the T1MZ and T2MZ are within the guideline for all elements.

14.2.13.3 Box Plots – Sub-Cell Model

Figure 14.18 displays the box plots for Pt within the T2MZ 2 g/t 3PE+Au.

The sub-celled and regularised models were also compared (Figure 14.19).

The box plots for the UMT-TCU model show good agreement between 1 m composites, ID3, NN and OK grade.

14.2.13.4 Swath Plots

Swath plots (width of 200 m) of the ID3 model, NN model and 1 m composites were completed for Pt, Pd, Au, Rh, Cu and Ni.

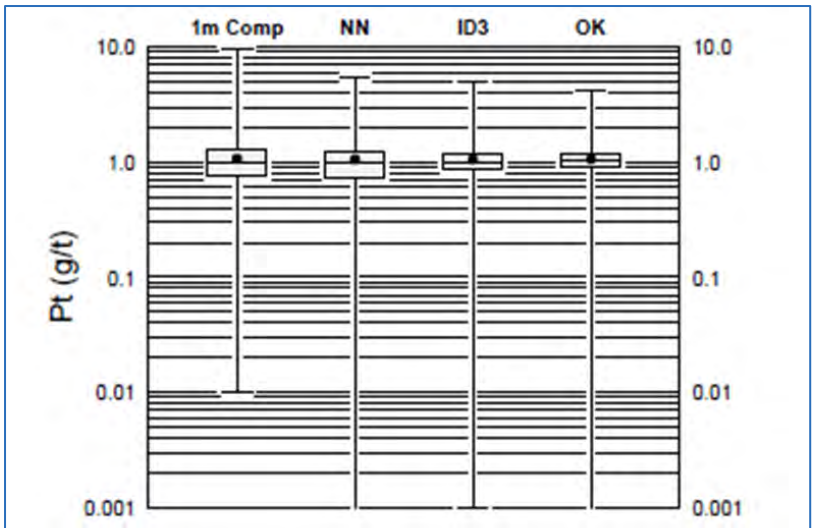
Overall, swath plots display reasonable comparisons between the ID3 estimates and their respective NN estimates; however, locally there are some differences, particularly in areas with limited drilling.

The platinum swath plot for the T1MZ 2 g/t 3PE+Au shell is presented in Figure 14.20.

Table 14.12 TCU Model Global Bias Check for Pt

GCODE	Element	NN	ID3	Relative Difference
101	Pt	0.618	0.604	-2.2%
	Pd	0.489	0.477	-2.4%
	Au	0.210	0.209	-0.7%
	Rh	0.028	0.027	-1.8%
	Ni	0.205	0.201	-1.5%
	Cu	0.107	0.105	-2.3%
102	Pt	1.139	1.155	1.4%
	Pd	0.897	0.901	0.4%
	Au	0.311	0.318	2.2%
	Rh	0.048	0.049	1.0%
	Ni	0.267	0.265	-0.7%
	Cu	0.142	0.141	-0.7%
103	Pt	2.389	2.428	1.6%
	Pd	1.981	2.025	2.2%
	Au	0.558	0.561	0.4%
	Rh	0.109	0.112	2.5%
	Ni	0.366	0.368	0.6%
	Cu	0.189	0.190	0.6%
201	Pt	0.663	0.657	-0.9%
	Pd	0.772	0.765	-0.9%
	Au	0.116	0.114	-1.6%
	Rh	0.046	0.046	-0.8%
	Ni	0.221	0.220	-0.5%
	Cu	0.110	0.110	-0.4%
202	Pt	1.045	1.045	0.0%
	Pd	1.168	1.166	-0.2%
	Au	0.177	0.175	-1.4%
	Rh	0.073	0.073	0.2%
	Ni	0.272	0.274	0.6%
	Cu	0.134	0.136	1.0%
203	Pt	2.295	2.295	0.0%
	Pd	2.326	2.336	0.4%
	Au	0.341	0.343	0.4%
	Rh	0.158	0.159	0.5%
	Ni	0.376	0.375	-0.3%
	Cu	0.187	0.187	-0.3%

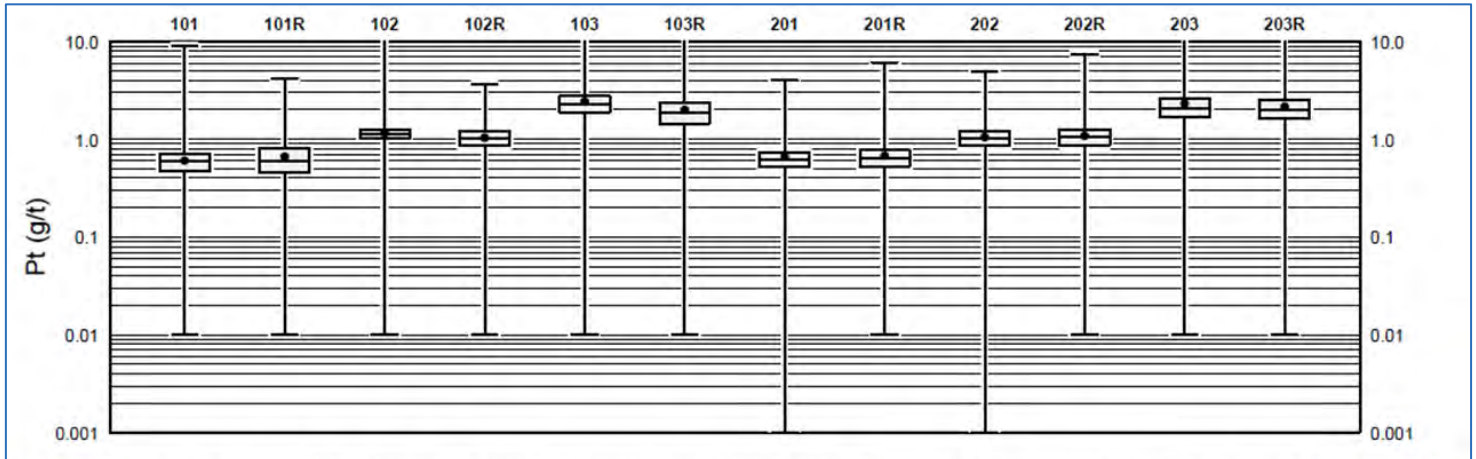
Figure 14.18 Box Plot of Pt for T2MZ 3PE+Au



	1m Comp	NN	ID3	OK	
No. of data	4029	1007591	1007591	1007591	No. of data
Mean	1.068	1.052	1.052	1.056	Mean
Maximum	9.670	5.379	4.870	4.254	Maximum
U.Quartile	1.260	1.246	1.190	1.176	U.Quartile
Median	0.990	0.990	1.006	1.017	Median
L.Quartile	0.747	0.740	0.858	0.884	L.Quartile
Minimum	0.010	0.000	0.001	0.000	Minimum
Variance	0.346	0.280	0.117	0.078	Variance
CV	0.550	0.503	0.325	0.265	CV
Skewness	2.867	1.605	1.266	0.978	Skewness

Figure prepared by Amec Foster Wheeler, 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

Figure prepared by Amec Foster Wheeler, 2015. "R" implies regularised 10 x 10 x 2 m blocks.

Figure 14.20 Platinum Swath Plot for T2MZ – 2 g/t 3PE+Au Shell; Regularised Model

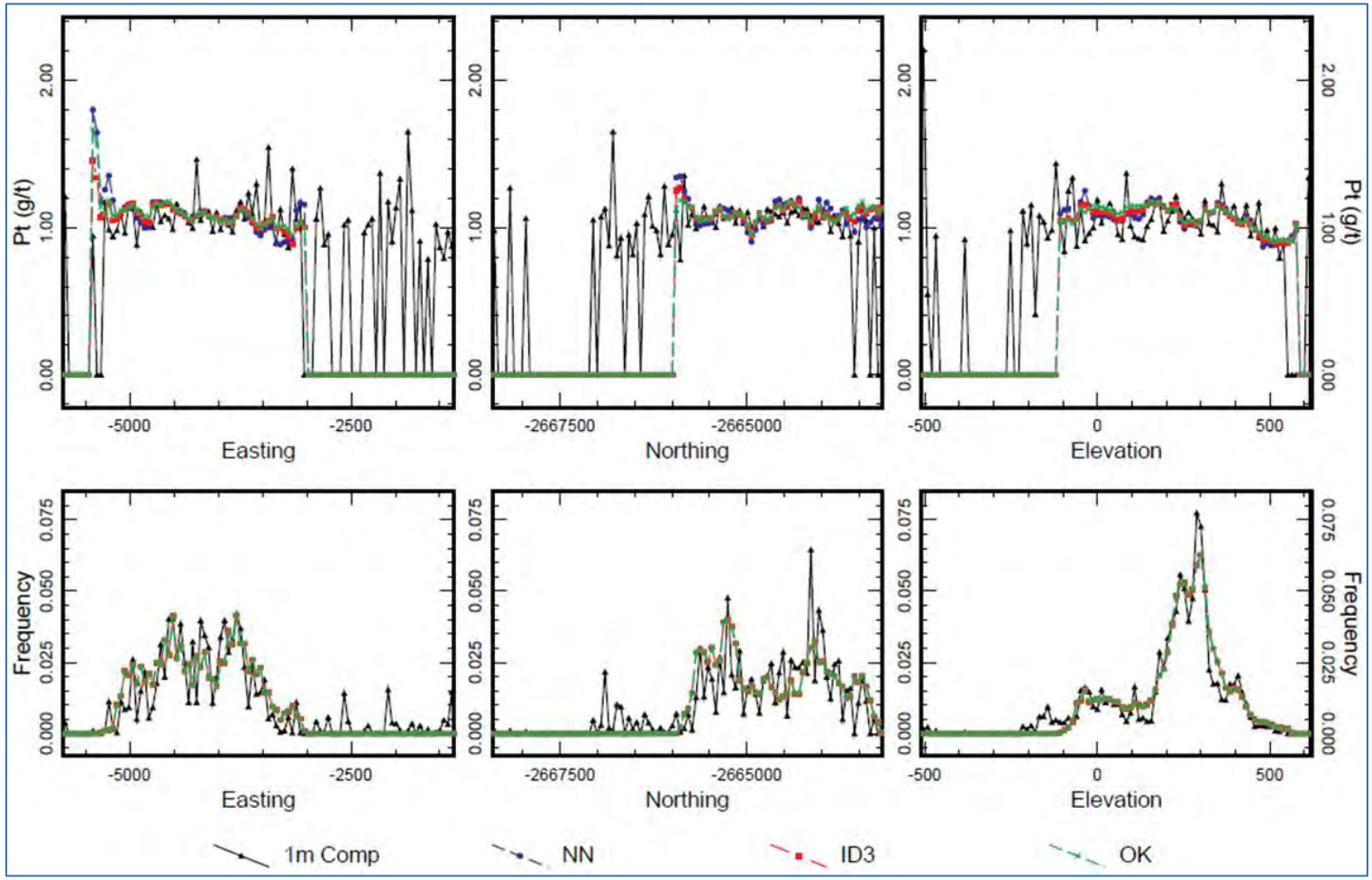


Figure prepared by Amec Foster Wheeler, 2016.

Comments on the UMT–TCU Model

The UMT–TCU model covers the TCU stratigraphic units. The UMT–TCU model also includes estimation of grades in blocks adjacent to the TCU, up to 20 m into the barren Main Zone gabbro norite, and 75 m into the footwall stratigraphy below the TCU.

Additional drilling is required in the areas classified as Inferred to better define the stratigraphic units and the fault domains.

Local bias is expected in the area classified as Inferred Mineral Resources because of the wide-spaced drilling and large search distances required for grade estimation. Additional drilling should permit better grade estimations.

14.3 UMT–Bik Bikkuri Reef Resource Estimate

The Mineral Resource for the Bikkuri Reef is located in Zones 1 and 2 (Figure 14.2) and is situated stratigraphically above the TCU (Figure 14.3). The Bikkuri Reef includes mineralisation that is amenable to underground selective mining methods and consists of material within and adjacent to grade shells for the Bikkuri Reef. The Mineral Resource for the Bikkuri Reef has been constructed using a geological interpretation that is similar to the TCU. The current interpretation is the Bikkuri Reef is a slump block of the main Platreef (see Section 7 and Figure 7.19). 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 14.2) and include 58 drillholes (66,865 m). All UMT drillholes have been re-logged for consideration of the TCU and the Bikkuri Reef. Three ATS drillholes (ATS123, ATS173, ATS176) were included in the Bikkuri model in Zone 2 for constructing the geological model.

14.3.2 Geological Model

The geology model for the UMT–BIK resource model was constructed in Leapfrog. A numeric model code (MCODE) was assigned to each lithology interpreted to be part of the Bikkuri Reef (Table 14.1). Stratigraphic surfacing functions in Leapfrog were used to construct the Bikkuri Reef geological model wireframes.

14.3.3 Model Envelope

The UMT–BIK model envelopes are constructed to include only the Bikkuri stratigraphic sequences.

14.3.4 Mineralised Zones

For the B1 mineralised zone (B1MZ), only a 1 g/t grade shell was modelled. Nested grade shells were constructed for the B2 mineralised zone (B2MZ) to constrain the grade estimation. The nested grade shells were identified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE +Au cut-offs. The grade-shell intercepts were coded into the drillhole database. The grade-shell drillhole intercepts were validated on dip and strike sections to ensure consistency. The grade shell drillhole intercepts were used to construct wireframes of the nested grade shells using stratigraphic surfacing functions in Leapfrog. Grade shell codes (GCODES) were used to code blocks within and outside the grade shells. The GCODES are summarised in Table 14.2.

14.3.5 Mineralisation Adjacent to the Bikkuri Mineralised Zones

There is scattered mineralisation locally adjacent to the B1MZ and B2MZ. Mineralisation adjacent to the Bikkuri mineralised zones may be included in future mine development, and a grade estimate is required for blocks within the Bikkuri model envelope.

14.3.6 Compositing and Exploratory Data Analysis (EDA) For UMT–BIK Model

The subset of drillholes used for the UMT-BIK resource was composited to 1 m length composites within the UMT-BIK model envelope. The compositing was controlled by the nested grade shells and the Bikkuri stratigraphic units.

EDA was completed using box plots, histograms, probability plots and contact profiles. Discontinuities in grade profiles near contacts suggested the grade shells and stratigraphic boundaries should be considered hard boundaries.

14.3.6.1 Rhodium Regressions

Rhodium analyses are only partially complete on the Bikkuri drillholes. Rhodium regressions for the Bikkuri drill data were used to address the missing rhodium data (See Section 14.2.6.1and Table 14.3 and Table 14.4).

14.3.7 Block Model

The UMT-BIK block model includes two areas where the Bikkuri Reef has been interpreted (Figure 14.2). Blocks were oriented parallel to the national coordinate system. The block model used a parent block size of 10 m x 10 m x 2 m. Sub-celling was 10 m x 10 m x 0.5 m. The geological stratigraphic units and grade shells were coded to the blocks. After estimation, the final resource model blocks were regularised to 10 m x 10 m x 2 m block sizes.

14.3.8 Grade Estimation — UMT–BIK

14.3.8.1 B1MZ and B2MZ

Grade estimation in the B1MZ and B2MZ included block and composite matching by GCODE. To eliminate the effects of the structural blocks and variability in elevation, the centre of individual stratigraphic units and mineralised zones were transformed to hang from the 1,000 m elevation.

The grade estimation in the B1MZ and B2MZ included block and drillhole composite matching by a combination of MCODE and GCODE to ensure the stratigraphic components of the B1MZ and B2MZ were estimated separately. Estimation was completed by ID3. A NN and OK estimations were completed for model validation.

After grade estimation, the blocks and composites were back transformed to the original elevation.

14.3.8.2 Blocks Adjacent Grade Shells

Grade estimation in the blocks not included in the B1MZ and B2MZ mineral zones were estimated by matching blocks and composites by MCODE. The individual stratigraphic units were hung from the 1,000 m elevation. Estimation was completed by ID3. Alternate NN and OK grade estimation was completed for model validation. After grade estimation, the blocks and composites were back transformed to the original elevation.

Grade estimations were completed in Datamine using expanding search volumes. Search volumes are summarised in (see Table 14.13).

Table 14.13 Search Strategy for Bikkuri Grade Estimation (All Elements)

Search Volume	Search Distances			Min. Samples	Max. Samples	Max. per Drillhole
	X	Y	Z			
1	300	300	100	4	15	3
2	600	600	200	4	15	3
3	120	1200	400	1	15	3

14.3.8.3 Grade Capping and Outlier Restriction

No grade capping or outlier restriction was implemented

14.3.8.4 Unestimated Blocks

Blocks that were not estimated were assigned a default grade of the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.14.

Table 14.14 Bikkuri Mean Grades by Stratigraphic Unit

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZBK	0.001	0.001	0.001	0.001	0.010	0.010	0.010
NC1BK	0.078	0.075	0.034	0.004	0.069	0.029	0.213
B1	0.125	0.090	0.049	0.007	0.084	0.030	0.195
B2	0.219	0.194	0.083	0.013	0.137	0.065	0.425
NC2BK	0.156	0.220	0.040	0.010	0.108	0.073	0.432
LZ1BK	0.267	0.313	0.060	0.060	0.134	0.086	0.945
B1MZ 1g	0.497	0.364	0.155	0.025	0.172	0.081	0.451
B2MZ 1g	0.632	0.661	0.145	0.034	0.244	0.145	0.867
B2MZ 2g	1.047	1.029	0.229	0.056	0.303	0.168	0.926
B2MZ 3g	1.445	1.323	0.311	0.077	0.389	0.211	1.099

14.3.9 Regularisation

Upon completion of the estimation, the UMT-BIK block model was regularised to 10 m x 10 m x 2 m (no sub-cells) model blocks. The 10 m x 10 m x 2 m regularised model permitted better resolution along the faulted boundaries and softened the hard boundaries used in the grade estimation.

Densities were coded to the blocks by stratigraphic unit using the mean density values for each stratigraphic unit (see Table 14.15).

Table 14.15 Bulk Density Values for The Bikkuri Model

Zone	Mean Density	CV	Max. SG	Min. SG
HW	2.91	0.04	4.47	2.04
NCMANBK	2.86	0.02	2.95	2.63
B1	3.11	0.06	3.90	2.60
B2	3.13	0.04	3.30	2.62
FW	2.91	0.04	4.47	2.04

14.3.10 Mineral Resource Classification

The Bikkuri Mineral Resources has been classified using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014), as discussed in Section 14.2.11.

The boundaries of Indicated and Inferred Mineral Resources for the TCU–BIK resource model are shown in Figure 14.2. The drill spacing in the Indicated Mineral Resource is nominally 100 m. Drill spacing in the Inferred Mineral Resource ranges up to 400 m.

14.3.11 UMT–Bik Model Validation

Model validation included visual inspection of block grades relative to composite grades on cross-sections and level plans. Statistical comparisons consisting of box plots and grade profiles tabulated in different directions (swaths) for each metal by stratigraphic unit and 3PE+Au grade shell were constructed to compare the kriged (where present), ID3 grade estimates, NN estimates and 1 m composites.

14.3.11.1 Global Bias Check

The checks for global bias for the UMT-BIK grade estimate were performed by comparing the ID3 average grade (with no cut-off) from the NN estimates by mineralised unit. Blocks reviewed were restricted to those classified as Indicated Resources. Domains with a global bias outside the recommended Amec Foster Wheeler 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%

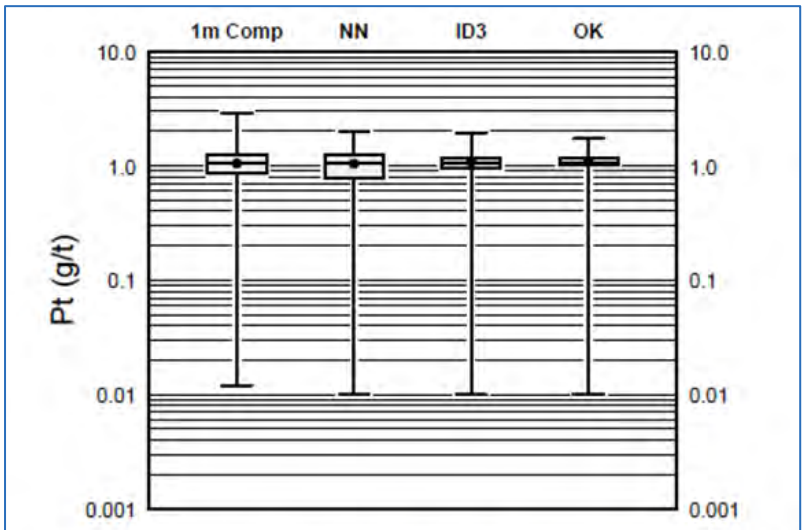
Note: Rows are shaded where the relative bias is outside (±5%)

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.

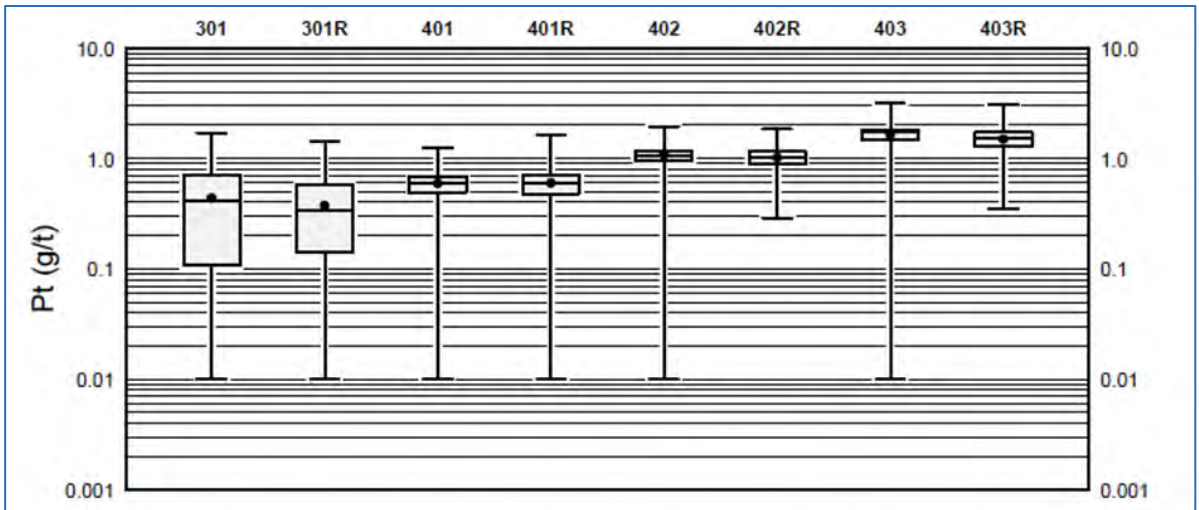
Figure 14.21 Box Plot Pt For B2MZ 2g/t 3PE+Au Figure Caption



	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

Courtesy 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

Courtesy 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.

Amec Foster Wheeler commonly focuses swath plot analysis 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

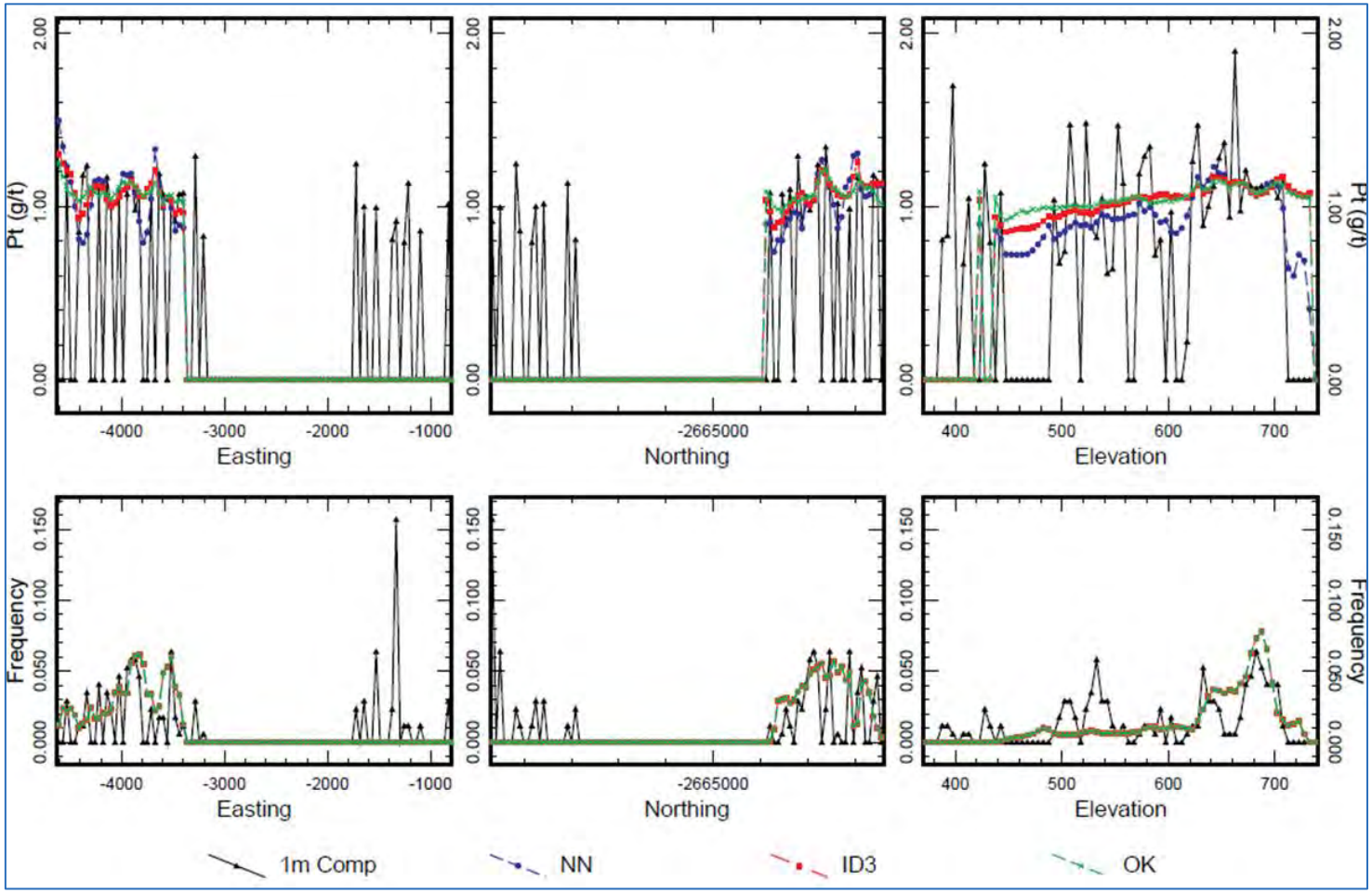


Figure prepared by Amec Foster Wheeler, 2016

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.

14.4 UMT–FW Footwall Resource Estimate

Mineralisation occurs stratigraphically below the TCU mineralised zone and has been interpreted to be within the Footwall Assimilation Zone (FAZ) and the Pyroxenite-Norite Zone (PNZ) (See Section 7.7). The mineralisation found within the FAZ is generally less continuous and disrupted by: 1) rafts of metasedimentary rocks, and 2) rock types that have been heavily assimilated. There are distinct assimilation products associated with dolomite assimilated rocks (calc-silicates and para lithologies) and hornfels assimilated rocks (a variety of norite products).

Numerous areas of footwall mineralisation have been identified. Two footwall domains were identified to have sufficient drillhole density and grade continuity to warrant the construction of Mineral Resource models. These include the Clinopyroxenite domain (FWcpx) and the Pyroxentie-Norite Zone (FWpnz). Figure 14.2 shows the location of the FWcpx and FWpnz domains. Additional footwall mineralisation is recognised, but the insufficient drillhole and sample data does not support resource modelling. These areas represent future exploration potential.

14.4.1 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.

14.4.2 Geology Model (UMT–FW)

The geological interpretations for the UMT–FW Mineral Resource model are based on revised geological interpretation. This interpretation is based primarily on the drill core re-logging campaign. The FW Stratigraphic coding is summarised in Table 14.16. Two mineralised domains are included in the UMT-FW mineral resource model. The upper domain is the FWcpx domain within the FAZ. Below the FAZ is mineralisation associated with the FAZ – PNZ contact and mineralisation within the PNZ associated with hornfels units (Figure 14.2).

14.4.3 CPX Domain

The FWcpx domain is confined to the NW area of Zone 1 (Figure 14.2), located within the FAZ (Figure 14.3). The domain is constrained by a very distinct, homogeneous clinopyroxene-rich pyroxenite where metasedimentary xenoliths have been completely assimilated. There are three main lithologies that make up the FWcpx domain. The main lithology is a clinopyroxenite (CPX) that locally includes added feldspar (FCPX) or added olivine (OLCPX).

14.4.4 PNZ Domain

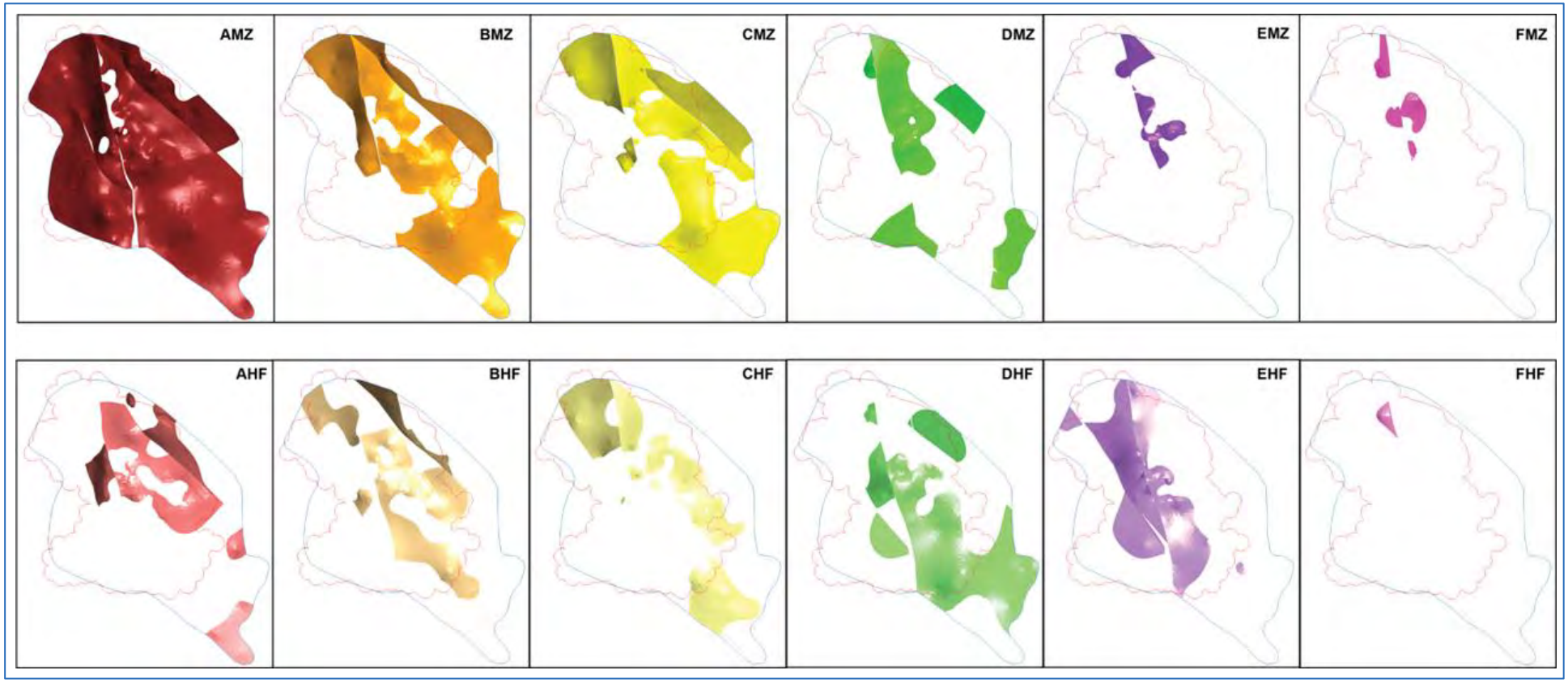
The PNZ is a unit of pyroxenite – norite composition that includes local unassimilated hornfels xenoliths. The mineralisation occurs predominantly as disseminated sulphides. Local massive sulphides are recognised at the contacts with the hornfels xenoliths. The 3PE+Au grades are commonly in the 1 g/t range, but locally can be 2 – 5 g/t 3PE+Au.

The contact between the FAZ and the PNZ is typically sharp (see Figure 14.3). A marked increase in mineralisation occurs locally at or near the contact FAZ-PNZ contact and is designated Mineralised Zone A (AMZ). The AMZ is distributed across the entire UMT-FW model area (Figure 14.24). Below the FAZ-PNZ contact, two distinct styles of mineralisation are observed within the PNZ pyroxenite. Discontinuous mineralisation, commonly as massive sulphides, is observed locally at the contacts of the hornfels xenoliths. More consistent mineralisation forms mineralised zones between hornfels units and it is these zones that were included in the UMT-FW model.

Six hornfels xenoliths (AHF, BHF, CHF, DHF, EHF and FHF in descending order) have been identified in the FWpnz domain. Five minzones (BMZ, CMX, DMZ EMZ and FMZ in descending order) are identified between the hornfels xenoliths. The lateral extent of the correlated mineralisation is less with each deeper minzone (see Figure 14.24). The units and mineralised zones were modelled across a wide portion of the project area (the blue perimeter in (Figure 14.24), the grade estimate was restricted to a narrower zone where tighter drillhole spacing allowed for continuity to be assumed.

The hornfels xenoliths and minzones were correlated in cross-section by Ivanhoe geology staff and coded to the drillhole database. Wireframes for the hornfels xenoliths and the FWpnz mine zones were constructed in Leapfrog using the vein modelling functions. Model and composite coding is summarised in Table 14.17

Figure 14.24 Extent of Mineralised Zones AMZ To FMZ and Hornfels Units AHF to FHF



Courtesy Ivanhoe, 2016. All of FWpnz is Inferred Mineral Resources. Blue boundary is FWpnz model boundary. Red boundary is TCU Indicated Mineral Resource boundary.

Table 14.17 MCODE and GCODE for FW Model

Modelled Unit	MCODE	Modelled Unit	GCODE
CPX	301	CPX	0
FAZ	30	FAZ	0
PNZ	31	PNZ	0
AHF	311	AMZ	311
BHF	312	BMZ	312
CHF	313	CMZ	313
DHF	314	DMZ	314
EHF	315	EMZ	315
FHF	316	FMZ	316

14.4.5 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

14.4.6 Outlier Restriction — UMT–FW

An outlier restriction distance threshold of 15 m was applied to high grade samples within each stratigraphic unit and mineralised zone. The grade thresholds for outliers were selected from inspection of the histograms and probability plots of 1m drillhole composites and are summarised in Table 14.19. Composites with grades above the grade threshold and with distances from composite to block centre beyond the distance thresholds were not used in grade estimation.

Table 14.19 Outlier Restriction Thresholds for Stratigraphic Units (MCODE)

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
CPX	4.50	5.00	1.20	1.50	0.75		6.00
FAZ	7.50	7.50	1.00	1.00	0.80	0.40	8.00
PNZ	2.00	2.00	0.60	1.00	0.90	-	8.00
AMZ	4.00	4.00	-	1.00	0.60	-	
BMZ	4.00	-	-	1.20	0.60	-	6.50
CMZ	4.50	-	0.50	1.20	-	-	7.00
DMZ	5.00	-	-	1.20	-	0.30	7.00
EMZ	5.00	4.00	0.60	0.80	0.60	-	4.50
FMZ	-	-	-	-	-	-	-
AHF	0.30	0.4	-	0.30	0.17	-	-
BHF	0.32	0.4	0.10	0.24	0.30	-	3.20
CHF	0.35	0.35	0.07	0.21	0.18	-	3.80
DHF	0.60	0.6	0.07	0.22	0.26	-	3.50
EHF	0.35	0.45	0.12	0.30	0.29	-	3.50
FHF*	0.35	0.45	0.12	0.30	0.29	-	3.50

14.4.7 Grade Estimation - UMT– FW

14.4.7.1 Grade Estimation - CPX Domain

The CPX model was constrained within a CPX model envelope that defined the limits of the FWcpx geological domain.

The block model used parent blocks of 10 m x 10 m x 2 m with no subcelling (Table 14.6). Fault blocks were not used to sub-domain the grade estimation for the FWcpx domain. The FWcpx model included portions of the TCU model but was not permitted to include blocks within the T2MZ.

Composites and model blocks were transformed so that the centre of the CPX domain was hung from the 1,000 m elevation for grade estimation. The composites and blocks were back-transformed to the original elevation after the grade estimation was completed.

Grade estimation was completed for Pt, Pd, Rh, Au, Cu, Ni and S using ID3. NN and OK estimations were completed for validation purposes. Variograms for the T2MZ g/t model were utilised for the OK estimation.

Grade estimations were completed in Datamine using only the first search volume (see Table 14.20). Blocks not estimated in the first search volume were excluded from the CPX model. The blocks not estimated are commonly located at the base of the CPX domain where drillholes were either not deep enough or the CPX intercept is not sampled.

14.4.7.2 Grade Estimation - PNZ Domain

The FWpnz model was constrained within the FAZ and PNZ envelopes. Fault blocks were not used to sub-domain the grade estimation for the FWpnz domain because only an isolated portion of the PNZ domain occurs east of the Tshukudu Fault.

The block model used parent blocks of 10 m x 10 m x 2 m with subcelling to 5 m x 5 m x 0.5 m (see Table 14.6) for better geological resolution. The PNZ model was permitted to overwrite the TCU model below the lower boundary of the T2MZ.

Composites and model blocks were transformed to hang from the centre of each lithology or MZ from the 1,000 m elevation for grade estimation. The composites and blocks were back-transformed to the original elevation after the grade estimation was completed.

Grade estimation was completed for Pt, Pd, Rh, Au, Cu, Ni and S using ID3. NN and OK estimations were completed for validation purposes. Variograms for the T2MZ 1 g/t model were utilised for the OK estimation. Grade estimations were completed in Datamine in three estimation passes using expanding search volumes (see Table 14.20). Blocks not estimated were excluded from the FWpnz model.

An outlier restriction was applied to the grade estimation using a distance threshold of 15 m and grade thresholds summarised in Table 14.19.

Table 14.20 Estimation Parameters for FW Model

Search Pass	Axis	Azimuth	Dip	Search Range	Min. Samples	Max. Sample	Max. per Drillhole
1	X	90	0	250	4	15	3
	Y	0	0	250	4	15	3
	Z	0	90	10	4	15	3
2	X	90	0	500	4	15	3
	Y	0	0	500	4	15	3
	Z	0	90	20	4	15	3
3	X	90	0	2,000	1	15	3
	Y	0	0	2,000	1	15	3
	Z	0	90	2,000	1	15	3

14.4.7.3 Unestimated Blocks

Blocks that were not estimated were assigned a default grade; the mean grade of the stratigraphic unit. The mean grades used are summarised in Table 14.10. Unestimated blocks were generally located along fault-block boundaries. Unestimated blocks within the FW stratigraphy were found to be located in areas of wide-spaced drilling.

14.4.8 TCU–FW Model Validation Global Bias Check

The checks for global bias for the UMT-FW grade estimate were performed by comparing the ID3 average grade (with no cut-off) from the NN estimates by GCODE. Domains with a global bias outside the recommended Amec Foster Wheeler guidelines of ±5% (relative) are highlighted in Table 14.21 and Table 14.22.

Table 14.21 summarises the global bias check for the FWcpx domain. The grade estimations are within the recommended ±5% tolerance.

Table 14.22 summarises the global bias check for the FWpnz domain. Generally, the grade estimations are within the recommended ±5% tolerance. Exceptions are the Au and Ni estimations for the CMZ zone (313).

Table 14.21 UMT–FWCPX Global Bias Check

Element	NN	ID3	Relative Difference
Pt	0.47	0.46	-1.51%
Pd	0.60	0.59	-0.98%
Au	0.08	0.08	-0.77%
Rh	0.06	0.06	0.48%
Ni	0.20	0.20	-0.70%
Cu	0.09	0.08	-1.82%

Table 14.22 UMT-FWCPX Global Bias Check

MZ	Element	NN	ID3	Relative Difference
AMZ (311)	Pt	0.50	0.50	-0.25%
	Pd	0.55	0.55	0.49%
	Au	0.08	0.08	1.34%
	Rh	0.07	0.07	-0.49%
	Ni	0.19	0.19	1.68%
	Cu	0.11	0.11	0.71%
BMZ (312)	Pt	0.44	0.44	-0.12%
	Pd	0.58	0.56	-2.50%
	Au	0.09	0.09	-1.29%
	Rh	0.07	0.07	0.99%
	Ni	0.21	0.21	0.41%
	Cu	0.12	0.13	0.63%
CMZ (313)	Pt	0.45	0.47	4.81%
	Pd	0.65	0.66	2.47%
	Au	0.09	0.09	5.39%
	Rh	0.07	0.07	1.25%
	Ni	0.25	0.27	5.42%
	Cu	0.14	0.14	2.62%
DMZ (314)	Pt	0.60	0.59	-0.56%
	Pd	0.71	0.71	-0.02%
	Au	0.11	0.11	0.62%
	Rh	0.07	0.07	0.69%
	Ni	0.20	0.21	3.85%
	Cu	0.12	0.13	2.32%
EMZ (315)	Pt	0.18	0.19	2.41%
	Pd	0.11	0.11	2.01%
	Au	0.61	0.62	2.39%
	Rh	0.70	0.71	1.45%
	Ni	0.09	0.09	0.05%
	Cu	0.07	0.07	1.09%
FMZ (316)	Pt	0.18	0.19	3.34%
	Pd	0.12	0.13	2.27%
	Au	0.46	0.48	4.00%
	Rh	0.60	0.61	1.53%
	Ni	0.09	0.09	-0.01%
	Cu	0.07	0.07	-0.14%

14.4.8.1 Visual Inspection

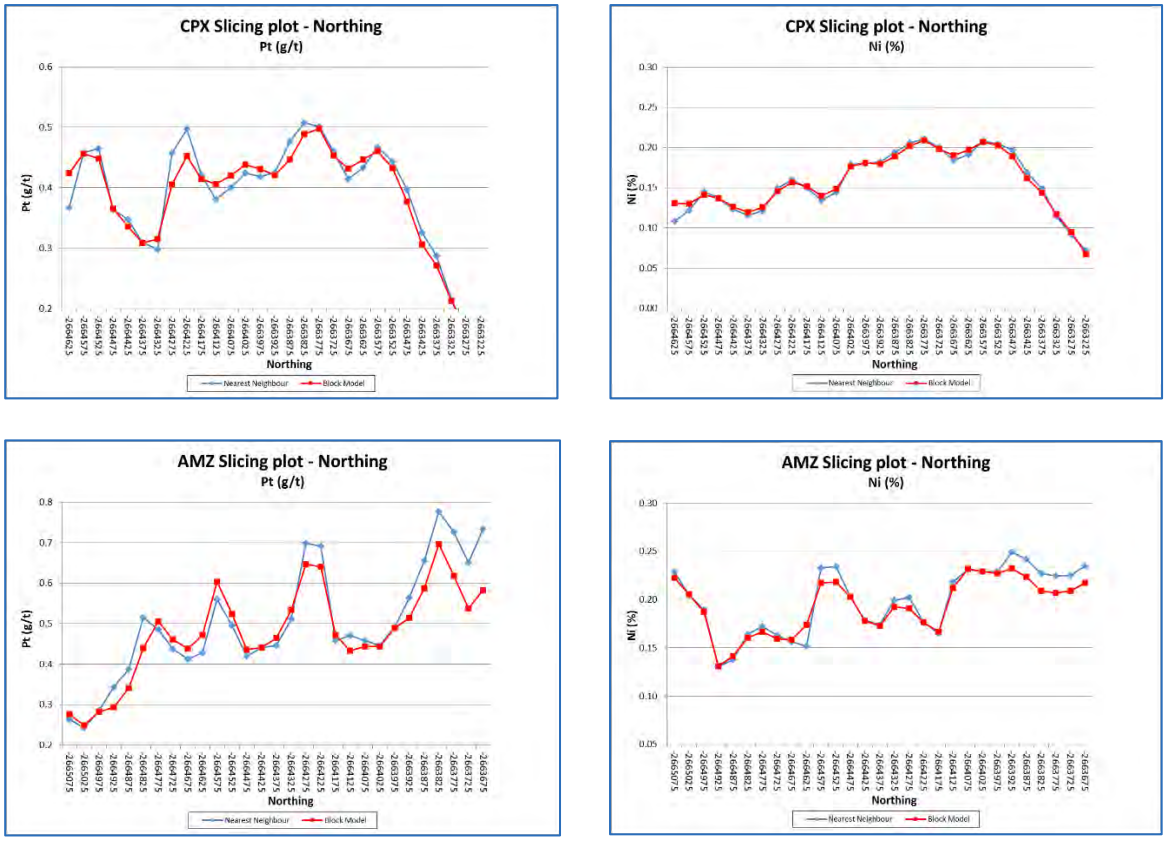
Visual validation of cross-sections for the FWcpx and FWpnz domains compared grade estimations and the 1 m composites. Figure 14.14 and Figure 14.15 shows the grade estimation for the FWcpx domain. Figure 14.16 and Figure 14.17 shows the grade estimation for the FWpnz domain.

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

14.4.8.2 Swath Plots

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

Figure 14.25 Platinum Swath Plot for B2MZ – 2 g/t 2PE+Au Shell



Courtesy Ivanhoe, 2016.

14.4.9 Classification

The Mineral Resource Classification for the FWcpx and FWpnz domains are Inferred due to the limited drilling.

14.4.10 Regularisation

Upon completion of the estimation, the FWpnz and FWcpx model were regularised to 10 m x 10 m x 2 m blocks and combined into a single UMT-FW Mineral Resource model. The 10 m x 10 m x 2 m softened the hard boundaries used in the grade estimation.

14.4.11 Comments on the UMT–FW Model

The UMT–FW model includes geological domains stratigraphically beneath the TCU that are observed to have a degree of geological continuity and homogeneity.

The UMT-FW model is limited to the FWcpx and FWpnz model envelopes that define unique geological domains within footwall stratigraphy. Additional drilling is required to better define the lateral extents of the FWcpx and FWpnz domains.

The continuity of FW mineralisation has been modelled based on limited drill data, as not all of the UMT drillholes extended into the FW. For this reason, estimation of Mineral Resources has been restricted to the north-western area of the Platreef Project where drill spacing is in the order of 100 m to 200 m. Similar mineralisation has been seen in drillholes across the entire Platreef Project, but the current drill spacing is insufficient to define Mineral Resources amenable to selective mining methods in these areas. This represents exploration upside for the Platreef Project.

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

14.5 Final Platreef Mineral Resource Model

The three Mineral Resource Models (UMT-TCU, UMT-BIK and UMT-FW) were combined into a final Platreef Mineral Resource Model.

14.6 Assumptions Made to Assess Reasonable Prospects of Eventual Economic Extraction

Amec Foster Wheeler undertook a conceptual analysis to assess reasonable prospects for eventual economic extraction for declaration of Mineral Resources. Underground mining methods considered are conventional, mechanised mining methods that have a reasonable safety factor. Assumptions made have been based on Base Data Template 20, received from Ivanhoe on 15 September 2015.

14.6.1 Commodity Prices

Amec Foster Wheeler considers that consensus long-term commodity prices should be used in declaration of Mineral Resources. For the Mineral Resource estimates, the following prices were used: $1,600/troy ounce for Pt, $815/troy ounce for Pd, $1,300/troy ounce for Au, $1,500/troy ounce for Rh, $3.00/lb for Cu and $8.90/lb for Ni.

14.6.2 On Site Operating Costs

For the selectively-mineable higher-grade scenario, a production rate of 4 Mtpa was assumed. Mining costs for some form of selective mining were estimated at $34.27/t. Process, concentrate transport and general and administrative (G+A) costs for this case were estimated at an average of $15.83/t of mill feed.

14.6.3 Process Recoveries

For the selective high-grade option, typical process recoveries are shown in Table 14.23. These recoveries were available from Base Data Template 20, provided by Ivanhoe on 15 September 2015.

Table 14.23 Typical Metallurgical Recoveries (15 September 2015, BDT20)

	Metallurgical Domain		
	Case 1	Case 2	Case 3
Mass Pull (%)	3.39	3.82	4.14
Typical Head			
Grades			
Pt (g/t)	1.13	1.68	2.11
Pd (g/t)	1.18	1.71	2.12
Au (g/t)	0.20	0.28	0.34
Rh (g/t)	0.08	0.11	0.14
4PGE (g/t)	2.59	3.78	4.65
Cu (%)	0.13	0.16	0.18
Ni (%)	0.26	0.32	0.35
Recoveries (%)			
Pt	79.9	84.6	87.7
Pd	80.6	84.3	85.8
Au	71.5	86.4	90.9
Rh	78.4	85.5	89.7
4PGE			
Cu	84.1	88.6	88.6
Ni	65.2	68.1	69.7

Mass Pull = percentage weight recovery to concentrates

14.6.4 Smelter Payables

Amec Foster Wheeler assumed that a smelter would pay for 82% of the metals contained in the concentrates. This assumption is based on a survey made by Kramer (2012). It is likely to cost an average of $39.77/t of concentrates (approximately $1.21/t of mineralised material) for road-freight to transport concentrates to a smelter, which for the purposes of assessing reasonable prospects, was assumed to be Rustenburg, in RSA.

There is some risk that if PGE concentrate grades are low, smelters would also levy treatment charges; on the other hand, it is envisioned that Platreef concentrates would be low in chromium, which might make them attractive to smelters whose feedstock primarily comes from Merensky and UG2 reef concentrates. Amec Foster Wheeler's conceptual analysis does not include treatment charges.

Platreef concentrates could also be marketed to smelters outside RSA.

14.6.5 Royalty

The royalty has been assumed as 5% of smelter payables.

14.7 Mineral Resource Statements

Mineral Resource statements for Mineral Resources amenable to underground mining methods are tabulated in this Section. The term base case has been used to indicate the tonnage and grade estimate that are considered by Amec Foster Wheeler to provide a starting point for feasibility studies. Other rows in the resource statements have been provided to show sensitivity of the estimated tonnages and grade to changes in cut-off criteria.

Mineral Resources are reported on a 100% basis. Attributable ownership is discussed in detail in Section 4.

14.7.1 UMT-TCU Mineral Resources Amenable to Underground Mining Methods

A selective mining scenario is considered the base case that could exploit mineralisation at depth within the Platreef. The selectively-mineable option is considered the base case Mineral Resource estimate for the purposes of this Report.

Mining costs have been considered in setting the cut-off ($34.27/t) for the selective mining case. Other considerations include process, concentrate transport and site G+A costs that must be covered for reporting Mineral Resources.

14.7.2 UMT-TCU Mineralisation Within and Adjacent to TCU Amenable to Underground Mining Methods (Estimate Assuming Underground Selective Mining Methods)

The TCU and adjacent blocks above T1, between T1 and T2 and below T2 contain higher-grade mineralisation that could be mined using underground selective methods such as longhole open-stoping, drift/cut and bench, bench-and-fill or drift-and-fill.

Table 14.24 shows Mineral Resources lying within and adjacent to the TCU mineralised zones.

In August 2017, Amec Foster Wheeler tested the Mineral Resources for reasonable prospects for eventual economic extraction (Parker, et al, 2017). The metals prices reviewed were: $1250/oz for platinum, $850/oz for palladium, $1300/oz for gold, $1000/oz for rhodium, $7.60/lb for nickel, and $3.00/lb for copper. At a 2 g/t 3PE+Au cut-off grade, >99% of the blocks will generate an NSR/t of $50 or higher, meaning they will pay mining, process, concentrate transport, and G&A costs. At the effective date of the estimate, an NSR/t of $50 was being considered by Ivanhoe, with longhole open stoping being the primary mining method. Approximately 90% of the blocks will generate an NSR/t of $70 or higher, which would be applicable to a drift-and-fill mining method. All of the blocks above a 1 g/t 3PE+Au cut-off generate an NSR of $15/t, meaning they will cover process, concentrate transport, and G&A costs. A check of the effect of implementing the metallurgical recoveries shown in Table 13.23 was also made, and given those recoveries were higher for Pt, Au and Ni and lower for Pd, Rh and Cu, the new metallurgical recovery equations do not meaningfully

affect block valuation compared to equations BDT20 (Table 14.23).

14.7.3 Bikkuri Reef Resource Estimate

Table 14.25 provides the total Mineral Resource estimate for mineralisation lying within and adjacent to 3PE+Au grade shells for the Bikkuri Reef.

Table 14.24 Mineral Resources Within and Adjacent to TCU (Base Case is Highlighted)

Indicated Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	202	2.11	2.12	0.34	0.14	4.71	0.18	0.35
2 g/t	339	1.68	1.71	0.28	0.11	3.79	0.16	0.32
1 g/t	685	1.13	1.18	0.20	0.08	2.59	0.13	0.26
Indicated Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t		13.7	13.7	2.2	0.9	30.6	788	1,576
2 g/t		18.4	18.7	3.1	1.2	41.3	1,197	2,386
1 g/t		24.9	26.1	4.3	1.8	57.19	1,977	3,938
Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3 g/t	212	1.93	1.95	0.32	0.13	4.33	0.17	0.35
2 g/t	459	1.45	1.48	0.27	0.10	3.29	0.16	0.31
1 g/t	1,213	0.91	0.96	0.18	0.07	2.12	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3 g/t	–	13.1	13.3	2.2	0.9	29.4	802	1,625
2 g/t	–	21.3	21.9	3.9	1.4	48.6	1,591	3,103
1 g/t	–	35.4	37.5	6.9	2.7	82.5	3,472	6,579

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

Table 14.25 Mineral Resources Within and Adjacent to Bikkuri (Base Case is Highlighted)

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

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

14.7.4 UMT- FW

Table 14.26 provides the total Mineral Resource estimate for mineralisation within the TCU-FW Model. The tabulation includes the FWcpx and the FWpnz resources.

Table 14.26 Mineral Resource Estimates for the TCU-FW Assuming Selective Underground Mining Methods (Base Case is Highlighted)

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

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

14.7.4.1 UMT-FWCPX

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

Table 14.27 Mineral Resource Estimates for the FW-CPX Assuming Underground Mining Methods (Base Case is Highlighted)

Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
2.5 g/t	4	1.34	1.60	0.21	0.09	3.24	0.19	0.40
2.0 g/t	10	1.08	1.29	0.17	0.08	2.63	0.16	0.34
1.5 g/t	27	0.82	1.00	0.14	0.07	2.03	0.14	0.29
1.0 g/t	58	0.64	0.78	0.11	0.06	1.60	0.11	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
2.5 g/t		0.17	0.20	0.03	0.01	0.41	16	35
2.0 g/t		0.33	0.40	0.05	0.02	0.81	34	73
1.5 g/t		0.71	0.86	0.12	0.06	1.74	80	170
1.0 g/t		1.19	1.47	0.21	0.12	2.99	144	317

1. Mineral Resources have an effective date of (22 April 2016). The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to 650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.
7. The FWcpx domain includes zones of low-grade mineralisation that may permit mass mining methods at a lower cut-off.
8. Mineral Resources in Table 14.27 are included in the tabulations in Table 14.26 and are not additive to that table.

14.7.4.2 UMT-FWPNZ

Table 14.28 provides the portion of the TCU-FW Mineral Resource estimate for mineralisation within the PNZ Domain.

Table 14.28 Mineral Resource Estimates for the FW-PNZ Assuming Selective Underground Mining Methods (Base Case is Highlighted)

Inferred Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
2.5 g/t	6	1.51	1.68	0.25	0.08	3.52	0.24	0.39
2.0 g/t	11	1.21	1.39	0.21	0.08	2.88	0.21	0.34
1.5 g/t	22	0.94	1.09	0.16	0.07	2.27	0.18	0.30
1.0 g/t	47	0.69	0.83	0.12	0.07	1.71	0.15	0.26
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
2.5 g/t		0.27	0.30	0.04	0.01	0.63	29	45
2.0 g/t		0.41	0.47	0.07	0.03	0.99	50	81
1.5 g/t		0.68	0.79	0.12	0.05	1.64	89	148
1.0 g/t		1.04	1.26	0.18	0.11	2.59	160	270

1. Mineral Resources have an effective date of (22 April 2016). The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to 650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.
7. Drill intercepts ≥2.0 g/t 3PE+Au suggest selective mining is required. Grade continuity best observed at a 1.0 to 1.5 g/t 3PE+Au cut-off. Discontinuous pods of mineralisation at a 2.0 g/t 3PE+Au are not well defined and additional drilling is required.
8. Mineral Resources in Table 14.28 are included in the tabulations in Table 14.26 and are not additive to that table.

14.7.4.3 Combined Mineral Resources

Table 14.29 provides a summary of the combined Platreef Mineral Resources for the UMT-TCU, UMT-BIK and UMT-FW models.

Table 14.29 Mineral Resources for All Platreef Mineralised Zones (Base Case is Highlighted)

Indicated Mineral Resources - Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3.0 g/t	204	2.11	2.11	0.34	0.14	4.70	0.18	0.35
2.0 g/t	346	1.68	1.70	0.28	0.11	3.77	0.16	0.32
1.0 g/t	716	1.11	1.16	0.19	0.08	2.55	0.13	0.26
Indicated Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3.0 g/t		13.86	13.86	2.23	0.92	30.86	800	1,597
2.0 g/t		18.66	18.94	3.12	1.23	41.95	1 226	2,438
1.0 g/t		25.63	26.81	4.49	1.82	58.75	2 076	4,108
Inferred Mineral Resources -Tonnage and Grades								
Cut-off 3PE+Au	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
3.0 g/t	225	1.91	1.93	0.32	0.13	4.29	0.17	0.35
2.0 g/t	506	1.42	1.46	0.26	0.10	3.24	0.16	0.31
1.0 g/t	1,431	0.88	0.94	0.17	0.07	2.05	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlbs)	Ni (Mlbs)
3.0 g/t		13.78	13.96	2.33	0.94	31.01	865	1,736
2.0 g/t		23.17	23.78	4.26	1.56	52.77	1,775	3,440
1.0 g/t		40.38	43.01	7.81	3.06	94.27	4,129	7,759

1. Mineral Resources have an effective date of (22 April 2016). The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The 2 g/t 3PE+Au cut-off is considered the base case estimate and is highlighted. The rows are not additive.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to 650 m elevation (from 500 m to 1,350 m depth). Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 x 400 m (locally to 400 x 200 m and 200 x 200 m) spacing.
4. Reasonable prospects for eventual economic extraction were determined using the following assumptions. Assumed commodity prices are Pt: $1,600/oz, Pd: $815/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: $3.00/lb and Ni: $8.90/lb. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $34.27/t) and process, G&A, and concentrate transport costs (average $15.83/t of mill feed for a 4 Mtpa operation) would be covered. The processing recoveries vary with block grade but typically would be 80%–90% for Pt, Pd and Rh; 70-90% for Au, 60-90% for Cu, and 65-75% for Ni.
5. 3PE+Au = Pt + Pd + Rh + Au.
6. Totals may not sum due to rounding.
7. Mineral Resources reported in Table 14.29 are included in Table 14.24, 14.25, 14.26 (note that Table 14.27 and 14.28 are included in Table 14.26) and are not additive to those tables.

14.8 Targets for Further Exploration

Beyond the current Mineral Resources, mineralisation is open to expansion to the south and west. Targets for further exploration (exploration targets) have been identified. Amec Foster Wheeler 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.

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.

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

Figure 14.26 Exploration Target Areas

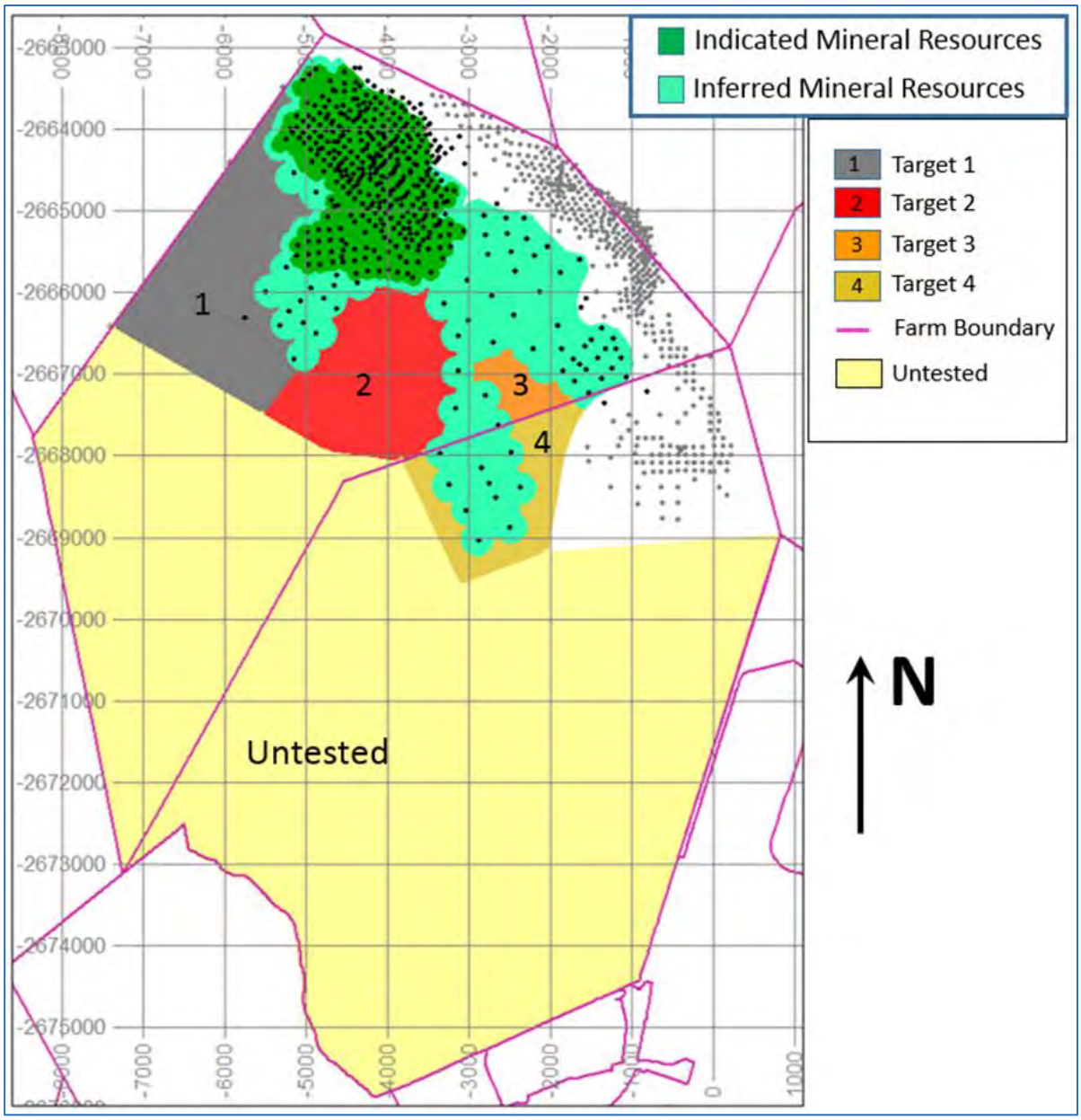


Figure prepared by Amec Foster Wheeler, 2016.

14.9 Comments on Section 14

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

Since the commencement of exploration in the UMT area, iterative Mineral Resource estimates between 2010 and 2015 have led to a progressive increase in the tonnage of Inferred Mineral Resources. With the inclusion of results from the 2014-2015 drill programme in the update of the block model reported herein, higher confidence category upgrades in the classification are supported, and will permit completion of more detailed mining studies.

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. Normal faults were used in the 2013 model to account for elevation changes.

Amec Foster Wheeler reviewed twin hole drill data, and notes that there are typically large differences in the positions of the top and bottom of the T1 and T2 in twin holes spaced less than 10 m apart. There is good correlation between the position of chromite stringers at the top of the T2. Other chromite stringers, granite veins, pegmatite veins that are actually formed from irregular bodies of intercumulus melt, and massive sulphides show weak or no correlation. There is fair correlation for length of the 1 g/t 3PE intercept, nickel grade and 3PE grade within the T1MZ. The T2MZ is thicker, and there are more assay intervals. The correlation is good to excellent for length of the 1 g/t 3PE intercept, nickel grade and 3PE+Au grade within the T2MZ. The implication for modelling is that the position of grade shell boundaries will be variable, and it will be difficult for mining to follow them.

14.9.1 Considerations for Next Model Update

Based on the available preliminary re-logging data and the provisional structural interpretations from seismic data, Amec Foster Wheeler notes the following considerations for the next model update:

- The T2U and T2L domains will be preserved, in case there are differences in metallurgical responses for these units.
- The presence of chrome stringers is known to enhance grade. Possibly distance from chrome stringers should be used in local domaining.
- The data on positions of grade shell boundaries should be examined to the extent possible to estimate their short-scale variability; the likely accuracies of down-hole surveys should be taken into account, and it is recognised a definitive answer may have to await exposures in underground workings.

Amec Foster Wheeler recommends the following:

- Re-logging ATS and AMK holes consistent with the new geological interpretation.
- Re-modelling ATS and AMK using the UMT litho-stratigraphic units and interpolation using total nickel and copper.

This will put all models on the same litho-stratigraphic and assay (total) basis.

14.9.2 Uncertainties Implicit in the Mineral Resource Estimate

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

- Permitting, environmental, legal and socio-economic assumptions.
- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction including:
  - Long-term commodity price assumptions.
  - Long-term exchange rate assumptions.
  - Assumed mining method.
  - Availability of water and power.
  - Operating and capital cost assumptions.
  - Metal recovery assumptions.
  - Concentrate grade and smelting/refining terms.
- Additional metallurgical sampling is planned once the updated geological interpretation has been validated; the ability to select samples from specific mineralisation layers may result in changes to the metallurgical recovery and smelter payables assumptions used to evaluate reasonable prospects of eventual economic extraction.
- Unmineralised GV dykes are not included in the geology model. These dykes may result in local over estimations of the volume of the mineralised material, however, any quantification of the dykes would require close-spaced drill data from underground drill stations.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. Dilution and mining recoveries will vary with the geometry (dip, thickness, faulting and or irregularities in contacts) of the mineralisation and the eventual mining method used. These factors can only be estimated after life of mine plans are prepared. Typically, dilution (low-grade or waste materials) ranges from 10–30%, and mining recoveries range from 70% to 100% using the mining methods considered for evaluation of reasonable prospects of eventual economic extraction.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

High-grade platinum group element (PGE) and copper-nickel mineralisation at Platreef occurs within a large area of layered mafic intrusion on a northern limb of the Bushveld Igneous Complex. Economic mineralisation occurs in two main zones within the Turfspruit cyclic unit: The Upper T1 pyroxenite unit and the Lower T2 pegmatoid unit. T1, with thicknesses of 10 m to 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 more weakly mineralised Lower T1 rock.

The current evaluation concentrates on mining the thicker T2 resource, as mineralisation in T1 is thinner and less continuous than in T2: 74% of the targeted reserves are contained in T2, while 19% are contained in T1. (The remaining 7% is dilution and other rock types.) While most the targeted reserves are contained within T2, T1 is occasionally mined where the grade of the T1 mineralisation justifies mining it along with T2. T1 is occasionally mined along with T2, where the grade of the T1 mineralisation justifies mining it along with the T2.

15.2 Resource Block Model

The probable mineral reserves available for mining at the Platreef were based on the resource block model (file name: prdf20c.bmf) developed by Ivanhoe with supervision by Amec Foster Wheeler and provided to Stantec in September 2015.

Model No. prf20c contains the following:

- 10 m × 10 m × 2 m parent blocks and 10 m × 10 m × 1 m sub-blocks.
- An updated base data template 20 (BDT20) has been used to calculate the NSR to use for the Platreef Feasibility Study initial mine planning.
- Grades- inverse distance (ID^3).
- Structural domains.
- Lithologic units.
- Classification.
- 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 case for the Mineral Reserve to assess the available Mineral Resource.

For the Platreef 2017 FS work, only the Indicated Resource is used. Inferred metal values are zeroed out in the block model for all resource calculations.

The Mineral Resource block model provided to Stantec also includes a variable called net smelter return (NSR). The NSR is the dollar value of the metals recovered from a unit of ore less the cost for transportation of concentrate to the smelter, smelting and refining charges, and other deductions at the smelter.

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.

Table 15.1 Metal Prices Used in NSR Calculations (NSR17 and NSR20)

Metal	Selling Price (US$)
Au	$1,300.00/oz
Pd	$815.00/oz
Pt	$1,600.00/oz
Rh	$1,500.00/oz
Cu	$3.00/lb
Ni	$8.90/lb

Table 15.2 Charges and Other Assumptions Used in NSR Calculations

Item	Assumption
Concentrator Recovery*	Varies by metal, rock type, and feed grade
Concentrate Transportation Charge	US$35.00/t
Payable Metal (smelting and refining)	82%
Royalty	5% of payable

* The only change between NSR17 and NSR20 is in the concentrator recoveries.

Validation of Model No. prf20c was compared to the interim model from the studies prior to feasibility. NSR17 and NSR20 used all the same metal prices and costs. The only difference between NSR17 to NSR20 is the concentrator recoveries. The comparison is illustrated below in Table 15.3 and Figure 15.1.

Table 15.3 NSR20 / NSR17 Comparison

Input	Grade Ratio	Set 1	Set 2	Set 3	Set 4	Set 5	Set 6	Set 7	Set 8	Set 9	Set 10
NSR20 Calculated	–	$0.00	-$0.87	$14.20	$55.57	$92.34	$130.16	$169.38	$207.91	$245.22	$263.49
NSR17 Calculated	–	$0.00	$14.22	$29.52	$61.91	$96.06	$131.58	$167.59	$204.32	$241.63	$260.47
Input Metal Grades	–	–	–	–	–	–	–	–	–	–	–
Cu (%)	0.10	0.00	0.02	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.30
Ni (%)	0.20	0.00	0.04	0.08	0.16	0.24	0.32	0.40	0.48	0.56	0.60
Pt (g/t)	1.00	0.00	0.20	0.40	0.80	1.20	1.60	2.00	2.40	2.80	3.00
Pd (g/t)	1.00	0.00	0.20	0.40	0.80	1.20	1.60	2.00	2.40	2.80	3.00
Au (g/t)	0.15	0.00	0.03	0.06	0.12	0.18	0.24	0.30	0.36	0.42	0.45
Rh (g/t)	0.070	0.000	0.014	0.028	0.056	0.084	0.112	0.140	0.168	0.196	0.210

Note: Typical metal ratios with respect to Pt were used to generate a test data set for an NSR20 & NSR17 calculation comparison.

Figure 15.1 NSR20 / NSR17 Comparison Graph

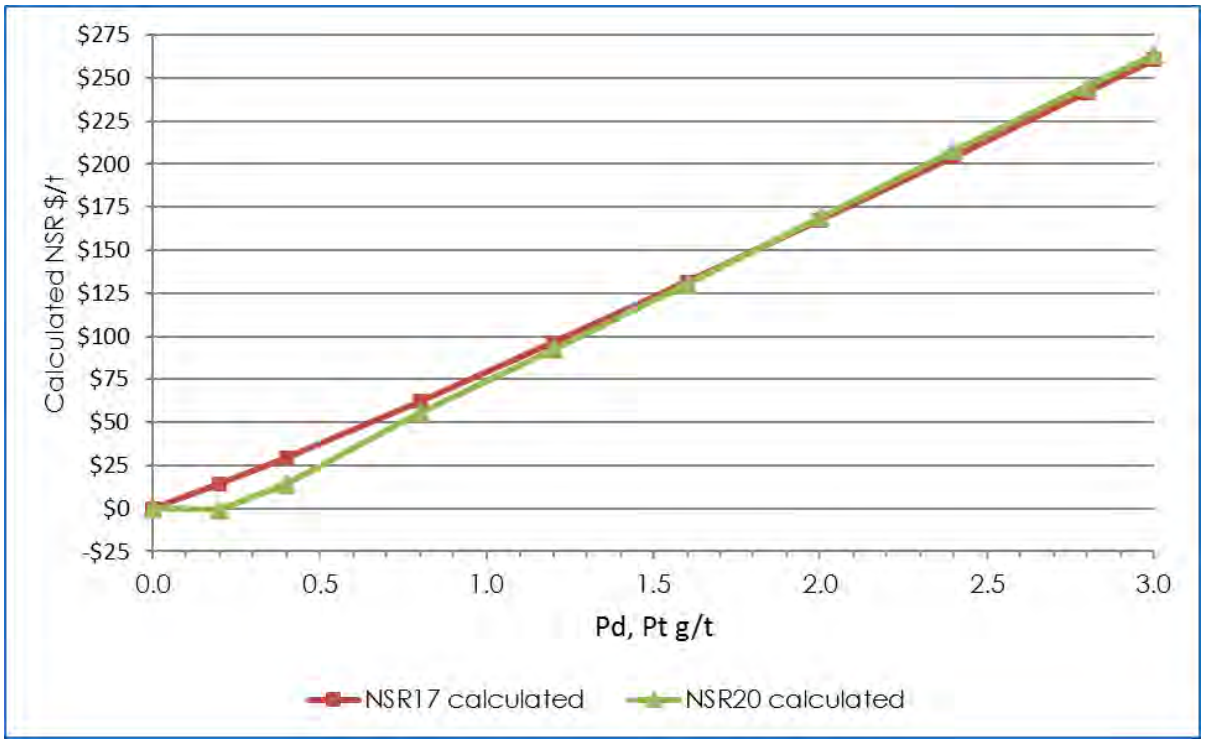


Figure by Stantec, 2017.

15.3 Economic Cut Off Grades

Global Net Smelter Return Evaluation

In the Platreef 2015 PFS, a breakeven cost analysis was completed by mining method. These costs ranged from $47/t to $58/t total cost. The break-even NSR cut-off must be high enough to pay for mining, processing, and on-site general and administration costs. Ivanplats' operational philosophy required additional allowances for capital and profit margin to be included in the NSR cut-off evaluation. As a starting point, and based on the Platreef 2015 PFS operating costs, NSR values of $80 and $100 were selected for evaluation. As part of the NSR evaluation further optimisation on a declining NSR cut-off approach was evaluated using NSR values from previous optimisation studies. These studies focused on the thickness of the deposit and the tonnage based on different cut-offs.

Based on a 30-year mine life and an achievable 4 Mtpa, the analysis was performed to determine the most economic tonnage to support the mine plan. The resource evaluation was performed on the grade shells and tonnages by height for each cut-off were determined. After this analysis, stope shapes were created for the high NSR cut-offs ($155/t and $130/t). The outliers that could not be economically accessed were removed, and to get the required tonnage for the 30-year mine life, an additional $100 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.

Table 15.4 summarises the global Indicated Resource tonnes, NSR, and 3PE+Au for the various NSR cut-offs.

Table 15.4 Resource Evaluation at Various Cut-Offs

NSR Cut-off	Indicated Resource (kt)	NSR20_2P ($/t)	3PE+Au (g/t)
$80/t	327,790	138.76	3.81
$100/t	239,450	158.36	4.36
$130/t	155,935	183.08	5.06
$155/t	101,848	204.42	5.69

15.4 Mineral Reserve

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in the Platreef 2017 FS. Only Indicated Mineral Resources have been used for determination of the Probable Mineral Reserve.

The Mineral Resource block model also includes the 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 Platreef 2017 FS Mineral Resource block model.

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

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

Prior to beginning stope design work and associated mineral reserve calculations, Stantec evaluated NSR cut-off values. The evaluation used updated mining cost estimates provided by Stantec as well as updated processing and G&A costs provided by Ivanhoe.

Economic cut-offs were established for each mining method and varied from $47.71/t - $58.53/t, excluding capital recovery and profit margin. For the production schedule and mineral reserve, a declining cut-off was chosen.

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

A definitive mine plan based on detailed stope layouts supports the mineral reserve. Due to irregularities in the geometry of the mineralised zones, not all material meeting cut-off grade can be mined without incurring some dilution. Due to inefficiencies in final mining recovery from the stopes, small amounts of mineralised material are lost during final stope cleanout, and additional losses may occur in transit from the stopes to the mill. Hence, a mining recovery factor is applied to the diluted resources to account for these losses.

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

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

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

Table 15.5 and Table 15.6 show the total diluted and recovered Probable Mineral Reserve for Platreef.

Table 15.5 Platreef Probable Mineral Reserve – Tonnage and Grades as at 24 May 2017

Method	(Mt)	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu %	Ni %
Ore Development	11.1	159.9	1.96	2.05	0.30	0.14	4.45	0.17	0.35
Longhole	93.1	152.1	1.88	1.95	0.29	0.13	4.25	0.16	0.33
Drift-and-Fill	20.4	182.0	2.28	2.23	0.37	0.15	5.03	0.18	0.37
Total	124.7	157.7	1.95	2.01	0.30	0.14	4.40	0.17	0.34

1. Mineral Reserves have an effective date of May 24, 2017. The Qualified Person for the estimate is Jon Treen (Stantec), P. Eng., with Professional Engineers of Ontario.
2. A declining NSR cut-off of $155/t to $80/t was used for the Mineral Reserve estimates.
3. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
4. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
5. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.
9. Metal prices used in the reserve estimate are as follows: Pt: $1,600/oz, Pd: $600/oz, Au: $1,300/oz, Rh: $1,500/oz, Cu: 3.00/lb, and Ni: $8.90/lb.
10. Tonnage and grade estimates include dilution and recovery allowances.
11. A declining NSR cut-off of $100–$80 was used in the mineral reserve estimates.
12. Total may not add due to the rounding.
13. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)

Table 15.6 Platreef Probable Mineral Reserve – Contained Metal as at 24 May 2017

Method	Mt	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Ore Development	11.1	0.7	0.7	0.1	0.05	1.6	42	85
Longhole	93.1	5.5	5.7	0.8	0.4	12.7	333	681
Drift-and-Fill	20.4	1.5	1.4	0.2	0.1	3.3	83	167
Total	124.7	7.6	7.8	1.2	0.5	17.6	457	932

1. Mineral Reserves have an effective date of May 24, 2017. The Qualified Person for the estimate is Jon Treen (Stantec), P. Eng., with Professional Engineers of Ontario.
2. A declining NSR cut-off of $155/t to $80/t was used for the Mineral Reserve estimates.
3. The NSR cut-off is an elevated cut-off above the marginal economic cut-off.
4. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
5. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.
6. Tonnage and grade estimates include dilution and mining recovery allowances.
7. Total may not add due to rounding.
8. 3PE+Au = platinum, palladium, rhodium and gold.

Based on the cut-off grade and mining criteria applied to the Platreef resource model, the Probable Mineral Reserve will support a 30-year mine life at a production rate of 4 Mtpa.

16 MINING METHODS

16.1 Mine Geotechnical

The primary aim of the Platreef 2017 FS investigation was to increase the level of confidence in the current geotechnical database and to undertake various analyses, based on data from the mine site and laboratory testing, to provide geotechnical design parameters and optimise the mine design going forward.

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

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

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

16.1.1 Study Methodology

The geotechnical investigation is based on an assessment of all the available geotechnical data, laboratory test data, the projects geological setting, major faulting, the proposed mine design and proposed stope orientations (as provided by Stantec).

Laboratory rock strength testing and stress measurement testing was conducted for the following purposes:

- Rock mass characterisation
- Empirical stope design
- Determination of Hoek-Brown parameters for numerical modelling

Significant input data were derived from 80 geologically and geotechnically logged boreholes, including 15 new geotechnical holes drilled to support the Platreef 2017 FS. Rock mass classification was conducted on all available geotechnical data and was assigned to a computer generated geotechnical block model. The geotechnical block model was validated against borehole data.

A structural analysis that focused on 34 boreholes was carried out to determine the major joint sets and their relationship to major faults in the project area. This was done to optimise mining directions by establishing structural domains and areas of potential instability based on major faults and the orientation of major joint sets.

The stability of the mining stopes was assessed using Mathews/Potvin's stability graph method, using the lower bound 20 percentile rock mass quality values. The stope height and width were determined in consultation with Ivanplats and Stantec. Following the analysis, maximum unfilled stope lengths were recommended for the different areas of the mine.

A risk assessment of the block failure potential of the back area and side walls of proposed transverse and longitudinal mining stopes was conducted using computer software JBlock. The failure potential of blocks was simulated in JBlock for the back, left wall and right wall of mining stopes of various proposed orientations (SOZ) considering Platreef's structural domains, for a vertically dipping rectangular excavation (Figure 16 2).

Figure 16.1 Proposed stope orientations by Stantec

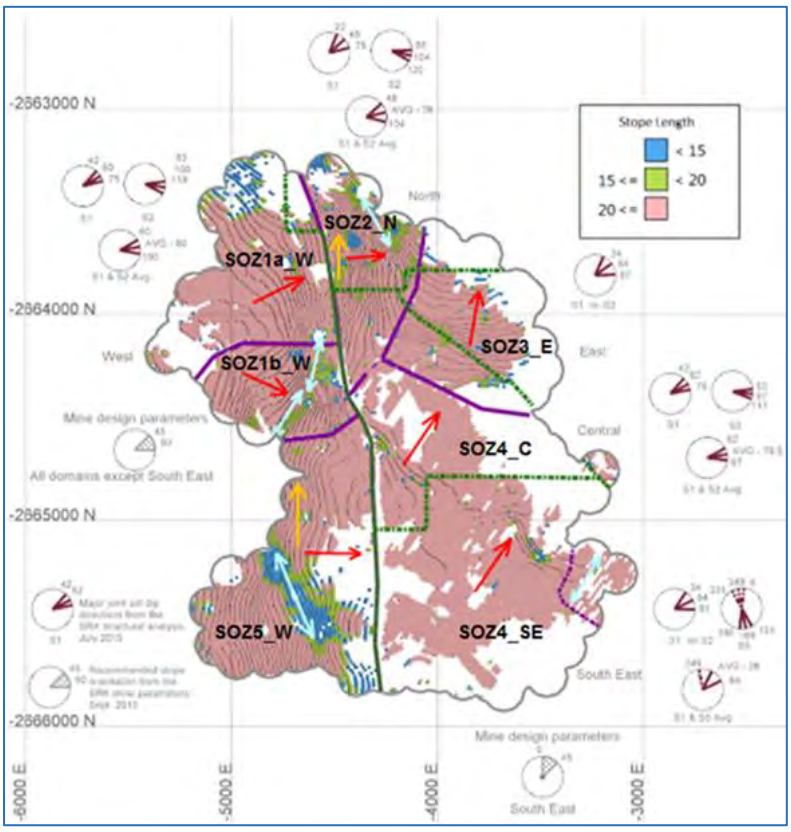


Figure by SRK, 2017.

A backfill strength analysis was performed and backfill strength requirements outlined. Bulkhead specifications were also provided.

A preliminary stress analysis was carried out using three dimensional elastic modelling of the Platreef 2015 PFS mining layout and sequence to determine the risk of stress damage. The results of the analysis were used to inform the geotechnical design parameters.

Geotechnical design parameters were derived to manage potential geotechnical risks that the mine may face. Parameters were outlined for the mine design and mine access design and include the backfill and support requirements.

The Platreef 2017 FS mine design (April 2016) was reviewed to ensure that it complied with the geotechnical design parameters and assessed using three dimensional elastic modelling.

16.1.2 Geotechnical Investigation Results

From the stress testing conducted it is observed that the pre-mining stress state at all locations tested across the project area is at a moderate to low level. While the major geological structures identified on the Platreef property have not been associated with any large magnitude tectonic earthquakes, it should be kept in mind that, the planned mining will cause significant stress changes along the major geological structures, which are likely to induce seismicity.

Rock strength properties have been determined for each stratigraphic unit within the project area, which form a comprehensive data set (Table 16.1 and Table 16.2). The results from testing indicate that the mean strength for the Main Zone (MZ), NC1, T1, T2U, T2L, NC2, FAZ and PNZ is 241MPa, 264 MPa, 189 MPa, 166 MPa, 163 MPa, 184 MPa, 173 MPa and 234 MPa, respectively.

Table 16.1 Design Rock Properties

Lithology	MZ	NC1	T1	T2U	T2L	NC2	FAZ	PNZ
No. of Samples	102	49	68	58	47	14	87	43
UCS Mean	241	264	189	166	163	184	173	234
UCS Mean – Std Dev	178	204	148	117	127	157	125	178
UCS Mean + Std Dev	304	325	230	214	199	212	222	290
Mi	20	21	10	10	10	14	11	17

Table 16.2 Intact Elastic Material Properties

Strat	No. of Tests	Results	Min.	Mean	Max.	Std. Dev.
MZ	17	Poisson's Ratio	0.16	0.30	0.37	0.05
		Young's Modulus (GPa)	52	89	103	13
NC1	12	Poisson's Ratio	0.27	0.33	0.39	0.03
		Young's Modulus (GPa)	86	96	130	12
T1	18	Poisson's Ratio	0.23	0.29	0.36	0.04
		Young's Modulus (GPa)	71	104	136	23
T2U	13	Poisson's Ratio	0.12	0.28	0.38	0.07
		Young's Modulus (GPa)	57	97	140	28
T2L	9	Poisson's Ratio	0.27	0.30	0.31	0.02
		Young's Modulus (GPa)	59	67	77	8
NC2	0	Poisson's Ratio	*	*	*	*
		Young's Modulus (GPa)	*	*	*	*
FAZ	6	Poisson's Ratio	0.24	0.31	0.43	0.07
		Young's Modulus (GPa)	62	102	131	29
PNZ	12	Poisson's Ratio	0.14	0.28	0.37	0.07
		Young's Modulus (GPa)	61	98	134	27

* No UCM test results

The Norwegian Geotechnical Institute's Q-system was utilised to facilitate the derivation of Q' values for the rock mass per geotechnical interval for each stratigraphic unit. Q' values were determined for use in the stope design and for the development support design. Based on the use of this rock mass classification system, it is observed that there is little variability in the overall quality of the rock mass, and that the rock mass can generally be described as "good". As Q' values are expressed on a log scale and are thus difficult to statistically analyse, Q' values were converted to Bieniawski's RMR for the purpose of statistical analysis and block modelling.

Following the conversion from Q' to RMR, a weighted average (compositing) of the RMR values was then determined for each borehole based on a 10 m interval. A summary of the rock mass classification results are presented in Table 16.3. Cumulative distribution function (CDF) curves and probability distributions (PDFs) for the results are presented in Figure 16.2 and Figure 16.3.

While the rock mass is generally of a "good" quality, geological faults and areas of poor ground do exist. A geotechnical block model has therefore been created to identify areas of "poor" ground Figure 16.4. Placement of permanent structures within these areas should be avoided.

Table 16.3 Summary of Rock Mass Classification Results

Strat Unit	No. of Geotechnical Intervals (Composited)	RMR (from Q')						
		Min.	Mean	Max.	50th Percentile (Median)	20th Percentile	80th Percentile	Std. Dev.
MZ	2,734	49	69	83	68	63	75	7
NC1	433	53	71	82	70	65	78	7
T1	832	48	65	82	63	61	68	5
T2U	351	52	67	82	66	61	72	6
T2L	287	46	68	82	67	63	74	7
NC2	22	58	64	72	63	61	67	4
FAZ	1,955	52	68	82	67	62	74	6
PNZ	689	42	64	82	64	60	70	8

Figure 16.2 Cumulative Distribution Curve of RMR Values (Converted From Q') for All Stratigraphic Units

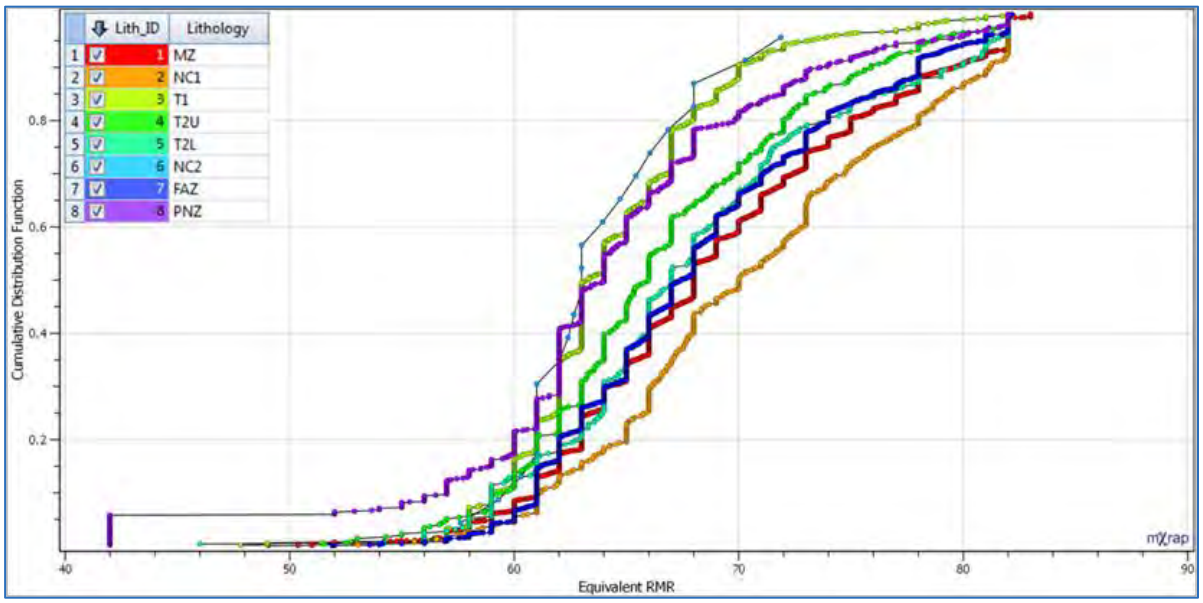


Figure by SRK, 2017.

Figure 16.3 Pdfs of RMR Values (Determined From Q') for All Stratigraphic Units

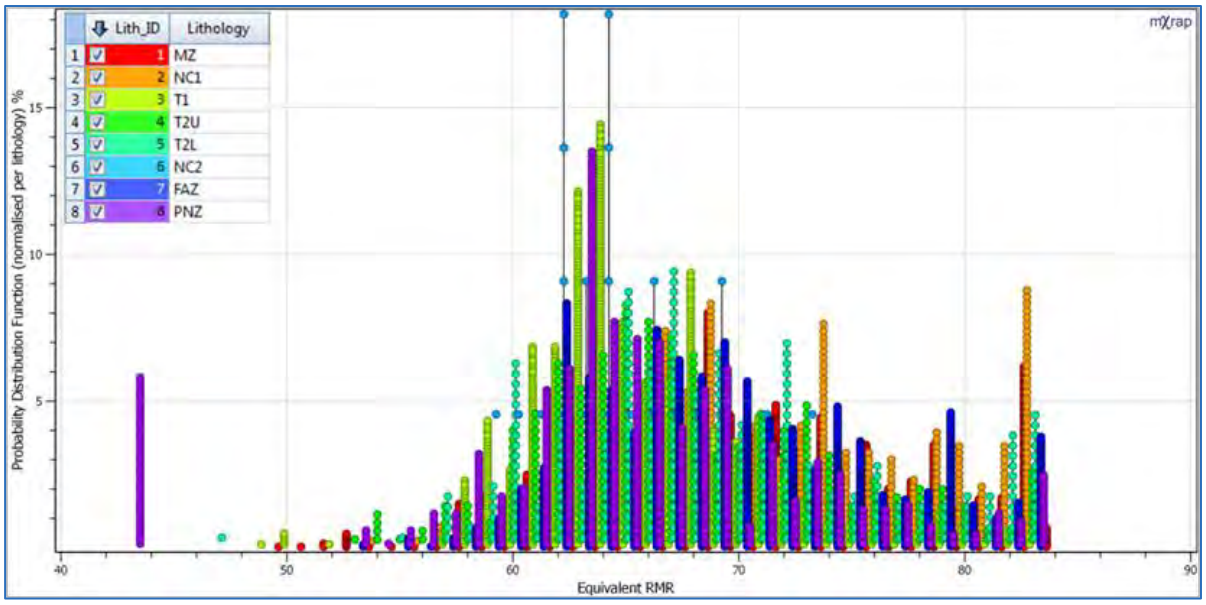


Figure by SRK, 2017.

Figure 16.4 Geotechnical Block Model

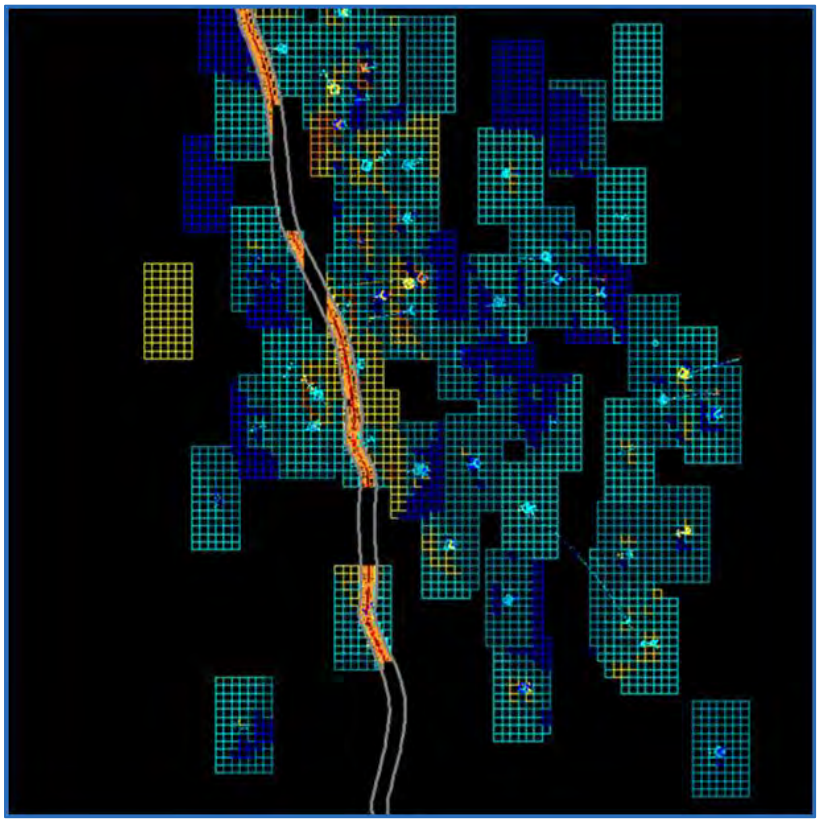


Figure by SRK, 2017.

From the structural analysis five structural domains (north, west, central, south-east and east) and six joint sets have been identified for the Platreef project (Figure 16.5). Based on the analysis the following is observed:

- Joint Set 1 strikes parallel to the Nkwe fault system, however Joint Set 1 is flatter dipping at 20°-65°.
- Joint Set 2 is an easterly dipping joint set present in the north, west and central domains. Joint Set 2 is parallel to the Kibaran-aged structures which form part of a system of closely spaced extensional faults that are developed across the central and western part of the project area. The Tshukudu and Nyati faults are included in this group.
- Due to the prominence of Joint Set 1 (across the project area) and Joint Set 2 (in the north, west and central domains), stopes mined perpendicular to these orientations will be more stable.

Figure 16.5 Stereographic projections per Structural Domain

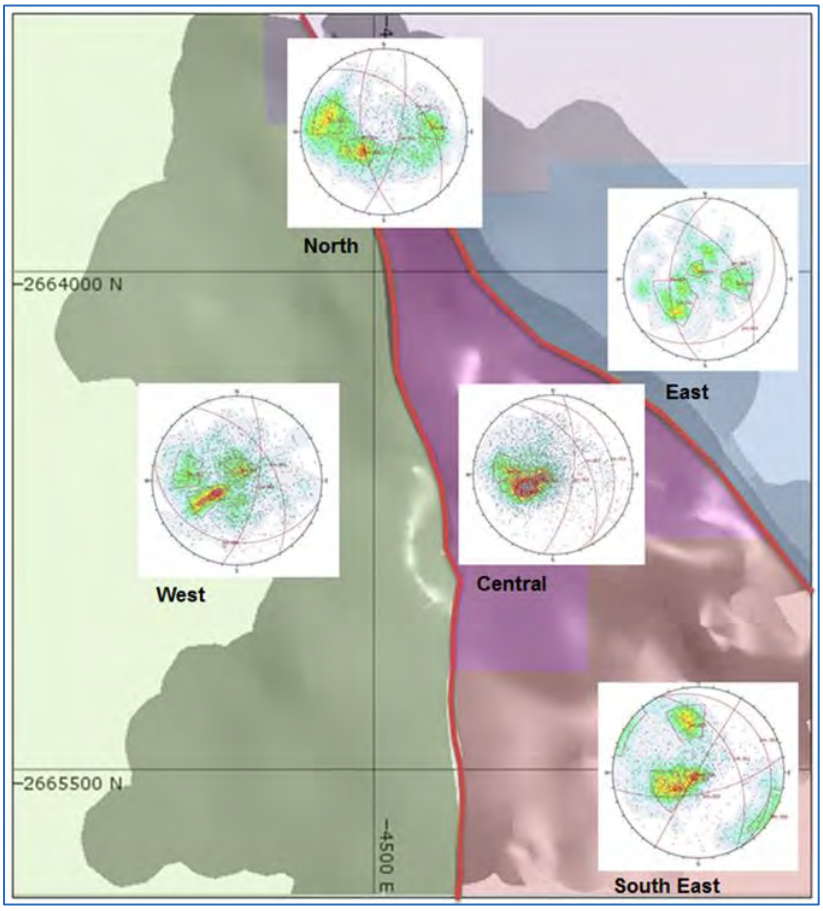


Figure by SRK, 2017.

16.1.3 Analysis and Design Results

From the keyblock analysis using JBlock, the results for the transverse stopes indicated that the expected linear overbreak from geological structures is low and is unlikely to influence the overall project. Based on the results, it is concluded that there will be a lower risk in excavating the transverse stopes in the orientations proposed compared with the longitudinal stopes (Table 16.4 and Table 16.5).

Table 16.4 JBlock Results – Transverse stopes

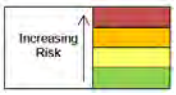
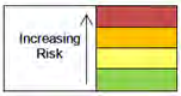
	Total No. of Blocks	No. of Failed Blocks	Failed Blocks (%)	Failed Block Area (%)	Expected Linear Overbreak (m)	Max Volume (m^3)	Max Area (m^2)	Max Height (m)
SOZ1a_W	751,515	1,754	0.23	0.53	0.004	344	90	8
SOZ1b_W	772,951	5,477	0.71	1.45	0.013	552	143	11
SOZ2_N	440,328	7,712	1.75	4.63	0.046	681	166	11
SOZ3_E	612,899	9,454	1.54	1.92	0.014	388	115	9
SOZ4_SE	663,308	13,129	1.98	4.57	0.034	644	178	9
SOZ4_SE no J6	978,307	7,884	0.81	2.86	0.017	1041	237	11
SOZ4_C	804,185	6,207	0.77	0.64	0.004	120	76	5
SOZ5_W	763,734	2,798	0.37	1.05	0.008	379	129	9

Table 16.5 JBlock Results – Longitudinal stopes

	Total No. of Blocks	No. of Failed Blocks	Failed Blocks (%)	Failed Block Area (%)	Expected Linear Overbreak (m)	Max Volume (m^3)	Max Area (m^2)	Max Height (m)
SOZ1b_W	761,415	34,018	4.47	4.52	0.021	255	179	8
SOZ2_N_150	465,636	26,157	5.62	7.74	0.056	625	193	9
SOZ2_N_000	512,056	42,259	8.25	7.63	0.048	463	198	9
SOZ5_W_155	759,584	46,778	6.16	6.41	0.038	438	139	10
SOZ5_W_000	512,465	18,506	3.61	3.16	0.024	369	122	7

The stability of the mining stopes was assessed using Mathews/Potvin's stability graph method, taking into consideration the rock mass quality (Q'), rock stress, rock strength, joint orientations and the orientations of the stope back and walls. The stope back is supported and therefore the interaction of joints with the stope walls has a significant influence on stope stability. Since the geotechnical block model represents Q' converted to RMR (Section 16.1.2), the recommended unfilled stope lengths are referenced to converted RMR (Table 16.6). As the median converted RMR value for the rock at Platreef is approximately 65, transverse stope lengths are likely to range between 25 m and 60 m for a stope height of 20 m (Table 16.6). It is therefore important that the geotechnical block model is used to verify the rock conditions (RMR) for the various stoping areas in the mine.

Table 16.6 Unfilled Stope Length

RMR	SOZ 1a W (65°)	SOZ 1b W (115°)	SOZ 2 N (85°)	SOZ 3 E (10°)	SOZ 4 C (35°)	SOZ 4 SE (35°)	SOZ 5 W (90°)	SOZ 5L W (0°)	SOZ 1bL W (35°)	SOZ 2L N (360°)
50-55	12	11	12	9	10	12	12	6	11	6
55-60	19	18	19	15	15	19	19	8	17	8
60-65	37	33	36	25	26	36	37	12	30	12
65-70	60	60	60	55	57	60	60	20	60	20
>70	60	60	60	60	60	60	60	39	60	39

Figure 16.6 indicates the backfill strength requirements, for both longhole stopes and drifts and benches. The bulkhead specifications are listed in Table 16.7.

Figure 16.6 Proposed Backfill Strength Requirements

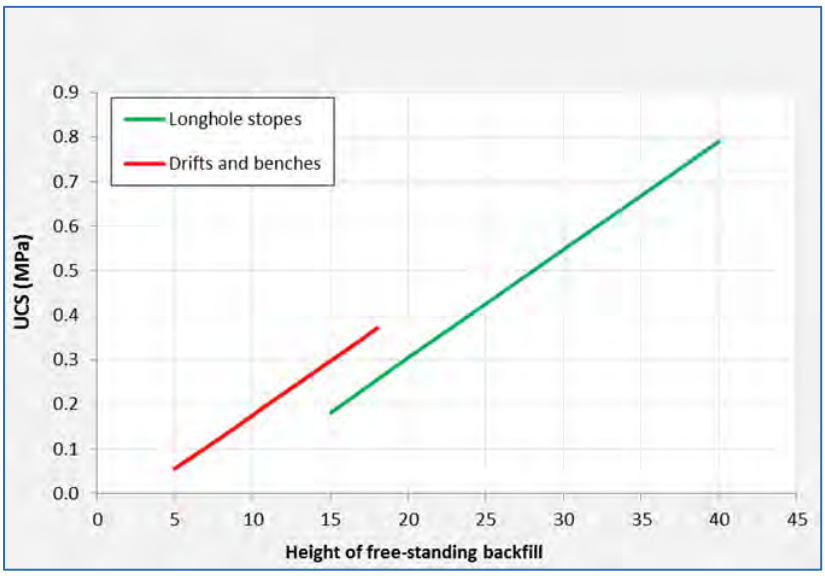


Figure by SRK, 2017.

Table 16.7 Fibre reinforced shotcrete bulkheads

Specifications	
Maximum stope entrance dimensions	6 m x 5 m
Minimum distance from brow	5 m
Minimum distance from intersection	2 m
Shotcrete thickness	0.3 m
Shotcrete compressive strength	25 MPa @ 7 days
Arched wall forming frame	5.0 m arc radius
Shotcrete curing time	48 hours
Plug height	6 m (1 m above stope drive)
Plug binder content	5%
Plug curing time	48 hours
Materials	
Arched wall frame kit	
Mesh 100 mm x 100 mm x 5 mm	
Hessian	
Fibre reinforced shotcrete	
Polypropylene drainage pipes	
Geotextile fabric for drain pipes	
Earth pressure cell	
Piezometer	

16.1.4 Geotechnical Design Parameters

Longhole stoping will be used where the thickness of the ore zones exceeds 20 m. Longhole stopes should not exceed a span of 15 m. Drifts should not be more than 6 m wide and should be developed in an arched profile to limit support requirements. The maximum height of drifts is dependent on safe operating practice.

Backfill will need to be placed in stopes before the maximum wall hydraulic radius is reached Table 16.6. RMR ratings can be obtained from the geotechnical block model. During mining, the unfilled stope length will vary depending on the assessment of local ground conditions and stope overbreak experienced.

Backfill in primary stopes must have a minimum strength as specified in Figure 16.6 to ensure that the backfill is stable during the mining of secondary stopes. There are no strength requirements for secondary stopes and either low strength paste fill of waste rock can be used.

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

A mine-wide seismic system should be installed within the first five years of mining to monitor the seismic response to mining.

Stope development (drilling access, mucking access, drilling levels and drifts) should be carried out "just in time". This is necessary to avoid damage due to changes in stress, particularly in secondary stopes and benches.

Stope brow support may be required to cater for stress and blast damage (see Table 16.8 and Table 16.9).

Excavations should be separated by twice the combined width to prevent excessive stress interaction.

Tunnels at depth will require S2 support during excavations and S3 support during stoping (see Table 16.8 and Table 16.9).

Long-term service excavations such as ramps need to be sited away from the isolated pillars.

The mining access sublevels must be horizontally offset at least 15 m away from mining. Longer term excavations such as ramps and main level haulages must be sited at least 30 m beneath planned mining.

During the mining of closing pillars, the mining access sublevels will be subjected to significant stress damage (0.5 m to 1.5 m depth of fracturing) and associated seismicity. In these instances, additional dynamic support (see Table 16.8 and Table 16.9) will be required, unless the closing pillar can be avoided through the mining sequence.

Isolated pillars will be highly stressed and excavations beneath these pillars will be subjected to intense stress damage (depth of fracturing > 1.5 m) and a high potential risk of seismicity. These should be minimised through improved mining sequences.

Shaft pillars must be large enough that the stress levels in the centre of the pillar do not cause damage to the shaft and critical excavations. The combined shaft pillar radius should be 250 m for Shafts 1 and 2 and 200 m for Ventilation Raise 1, Ventilation Raise 2 and Ventilation Raise 3.

The shaft pillars may be extracted at the end of the life of mine. Approximately 60% - 70% of the shaft pillar can be extracted using the infrastructure within the pillar with a controlled mining sequence and modification of the shaft steel work and lining. This can be increased to about 80% if alternate infrastructure can be used.

The Tshukudu fault has been identified as a major fault zone (8 m wide) with very poor quality rock and must be avoided. Development through the fault should be planned carefully to avoid delays and costs. There should be at least two main haulages developed though the fault, to ensure two means of egress.

A potential for seismicity exists on the geological structures in some stopes and is likely to affect nearby access developments. Additional S3 support will be required to cater for dynamic loading.

The proposed support standards and specifications for Platreef are listed in Table 16.8 and Table 16.9.

Table 16.8 Support Standards for excavations

Excavation Type	Support Standard
Shafts (blind sink)	Primary Support: Minimum 1.8 m long SS 33 split sets in a maximum 1.0 m x 1.5 m pattern with mesh. Secondary support: 300 mm concrete lining.
Vent shafts (raisebore)	Minimum 50 mm shotcrete or concrete lining.
Tunnel support for normal conditions (fracture depth < 0.5 m) (S1A)	Primary Support: 2.4 m long, tensioned resin rebars in a 1.8 mx1.8 m pattern with 50 mm shotcrete in crown and down to 1.5 m from floor (four rebars in back and two rebars in each sidewall per row). Support installed to face prior to face drilling. Required for main level haulages and ramps (long term).
Tunnel support for normal conditions (fracture depth < 0.5 m) (S1B)	Primary Support: 2.4 m long, tensioned resin rebars in a 1.8 mx1.8 m pattern with mesh in crown and down to 1.5 m 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 (fracture depth between 0.5 m and 1.5 m) (S2A)	Primary Support: 3.0 m long, tensioned resin grouted yielding bars in a 1.2 mx1.2 m pattern with 50 mm shotcrete in crown and down to 0.5 m from floor (five yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling. All mining access sublevels will require S2 support after ten years of mining. Shotcrete required for main level haulages and ramps (long term).
Tunnel support for high stress, dynamic conditions (fracture depth between 0.5 m and 1.5 m) (S2B)	Primary Support: 3.0 m long, tensioned resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh in crown and down to 0.5 m from floor (five yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling. All mining access sublevels will require S2 support after ten years of mining. Not in main level haulages and ramps (long term).
Tunnel support for extreme high stress, dynamic conditions and rehabilitation (fracture depth > 1.5 m) (S3)	Primary support installed during development. Secondary support: Minimum 3.0 m long grouted, resin yielding bars in a 1.0 m x 1.0m 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).
Support for hazardous geological structures	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 intersections	Primary support + minimum five 4.5m long grouted, cable anchors installed in crown at time of development.

Excavation Type	Support Standard
Development through Tshukudu fault	Spiling (12° to 15° to the horizontal) rings of 6.5 m long self-drilling anchors (SDAs) every 1.5 m (spacing 0.5 m to 1.0 m). Resin injection through the SDAs for consolidation. Steel arches spaced 0.5 m to 1.8 m apart depending on rock quality (expected 1.0 m), void filling between arches and tunnel walls. Resin consolidation and reinforcement may also be required in the face.
Large excavation support	Primary support + pre-tensioned, grouted, cable bolts (minimum length = half excavation span), maximum spacing = 0.5 x length.
Large excavation brow support	Primary support + three rows of pre-tensioned, grouted, cable bolts, in a 1.5 m x 1.5 m pattern or denser, depending on mass to be supported. Length = brow height. Installed within 1.0m of the brow.
Stope drive support	Primary support: 3 m long resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh or 50 mm shotcrete in crown and down to 0.5 m from floor (seven yielding bars in back and three yielding bars in each sidewall per row). Support installed to face prior to face drilling.
Stope brow support (where necessary)	Primary support + three rows (1.0 m apart) of three 6.5 m long grouted, cable anchors installed within 1.0 m of planned brow position.

Table 16.9 Support Specifications

Support Type	Minimum Specification
Rebar	Minimum yield strength 500 MPa black steel, minimum 20 mm diameter, hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing must be demonstrated through approved testing).
Yielding bar	Minimum yield strength 500 MPa black steel, minimum 20 mm diameter, minimum energy absorption 30 kJ within 300 mm, hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing must be demonstrated through approved testing).
Split set (SS-33)	Minimum outer diameter 33.5 mm to 34.2 mm, minimum 420 MPa (yield stress) black Supraform steel, minimum steel thickness 2.3 mm, hole size 30 mm to 32 mm
Self drilling anchor	Minimum 32 mm diameter black steel, 360 kN ultimate load, hollow self-drilling anchors (R32s).
Cable anchor	Minimum 18 mm diameter black steel, 380 kN ultimate load.
Mesh	Black weld mesh, minimum 5 mm gauge, maximum 100 mm aperture, blast resistant.
Shotcrete	Minimum 25 MPa (28 day strength) fibre reinforced shotcrete.
Osro straps	300 mm wide straps with five 10 mm rods, minimum 500 MPa black steel.
Capsule resin	Two component urethane silicate resin capsules. Fast (< 30s) and slow (5-10 min) setting.
Injection resin	Two component urethane silicate injection resin with water sealing properties.
Cable grout	Minimum 40 MPa Ordinary Portland Cement, water cement ratio 0.35 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 Review of the Mine Design

A three-dimensional elastic numerical modelling exercise was conducted to assess the updated FS mining layout (April 2016) and 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 few areas where it will be necessary to undermine backfill placed in the stope above. Where this occurs, it is necessary to plan ahead and include additional cement binder to maintain the stability of the backfill sill above the active stope.

16.1.6 Geotechnical Risks

The Tshukudu fault has been identified as a major fault which traverses the Project area. This fault presents a major geotechnical hazard as it is often characterised by very poor quality rock. Development through the fault should thus be planned carefully to avoid delays and costs. As the Tshukudu fault strikes across the entire lease area, some development through the Tshukudu fault will be essential to provide access to ore to the west of the fault. There should be at least two main haulages developed though the fault, to ensure two means of egress. Specialised support comprising resin injection, arch sets and void filling (Table 16.8 and Table 16.9) to be carried out by a specialist contractor is recommended for this case. An indication of the slow rate of this development is also provided.

The ingress of water through the Tshukudu fault is another potential risk as the presence of water is likely to negatively influence stability. Modelling by Golder Associates indicates that 80% of the groundwater inflow into the underground workings will be derived from the faults traversing the mine. The Tshukudu fault will produce a greater inflow since it covers more mining area than the other faults (see DRA Report Number: DRA-J0283-STU-REP-909 S04).

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

Overall the Tshukudu fault has been taken into consideration in the mine design, however mining through the fault will need to be monitored and managed carefully. Geotechnical drilling will be required to delineate and characterise the Tshukudu fault during implementation.

16.2 Underground Mining

16.2.1 Introduction

Stantec began with the mining methods, mine design, and schedule recommended in the Platreef 2015 PFS and optimised them based on the new block model and other factors. Mining method selection focused on mechanized mining with high productivities. Mine design work aimed to maximise production grades and reduce operating costs. Mine schedule work focused on optimising stope sequencing, maximising grades in the early years, and removing development from the critical path. The sub-sections below discuss considerations taken in the mining methods selection process and in the final mine designs and schedules.

16.2.2 Mine Design Parameters

The following mine design criteria were used for the Platreef 2017 FS.

Ore and Waste Properties

- Ore in Situ Density
  - T1 = 3.18 t/m³
  - T2U = 3.19 t/m³
  - T2L = 3.04 t/m³
  - Average =3.14 t/m³
- Waste in Situ Density (Pyroxinites)
  - Hanging Wall = 2.91 t/m³
  - Footwall = 3.10 t/m³
- Swell for Development = 40%
- Swell for Production (Longhole Stoping) = 40%
- Swell for Production (Drift-and-Fill) = 40%

Mine Planning

- Lateral Development
  - Maximum grade will be 15%.
  - Ramps will level off at sublevels to reduce risk of rollovers.
  - Ore passes will be spaced 400 m apart on the levels.
  - Electrical bays will be spaced 400 m apart.
  - Level development will be inclined slightly to have water drain to the sumps (actual incline to be designed during detail design).

- Vertical Development
  - All raises except Shaft 1 and Shaft 2 will be raisebored.
  - Maximum raisebore diameter is 6 m.
- Longhole Stoping
  - Bottom-up Longhole Stoping will be used.
  - Minimum inclination of stopes on the footwall side will be 55°.
  - The unfilled stope length for the 20 m high Longhole Stopes should not exceed 60 m.
  - As much as possible, stopes should be aligned to minimise the risk of failure.
  - A primary-secondary stoping sequence is recommended.
  - Adjacent secondary stopes are not mined simultaneously.
  - Secondary stope development can be carried out at any stage, regardless of whether nearby stopes have been backfilled.
  - Overbreak thickness for top sides, hanging wall, footwall, end sides, and secondary stope sides is 0.5 m.
  - Overbreak thickness for the floor is 0.3 m.
- Drift-and-Fill
  - Overhand Drift-and-Fill will be used.
  - Each horizontal slice is mined completely before mining of the slice above begins.
  - Drifts will be driven with flat backs for recovery purposes.
  - Mining widths should be limited to 5.5 m to reduce the length of ground support in the back and walls to 2.4 m. If mining widths exceed 5.5 m, additional support will be required.
  - Assuming a mining width of no more than 5.5 m, the mining sequence should be primary-secondary-tertiary to ensure that slender backfill ribs are not formed. For this method a 5.0 m width was utilised.
    - An average of 0.15 m of overbreak will be used to estimate the overall dilution from each paste fill rib.

The stoping designs were based on the Net Smelter Return (NSR), the inputs of which are defined in Section 14.6 Assumptions Made to Assess Reasonable Prospects of Eventual Economic Extraction.

Geotechnical information was used for development designs, ground support, and stope layouts. The ventilation parameters defined minimum air velocities, air quantities, and cooling requirements. Shaft parameters were used to determine the production rate along with the production ramp up profile. The underground infrastructure parameters defined required mine facilities, excavation sizes, and equipment requirements (mobile and fixed).

16.2.3 Mining Method Selection

As was concluded in the Platreef 2014 PEA and Platreef 2015 PFS, to minimise mining costs and achieve the highest productivity, most production will use Sublevel Blasthole (Longhole) Stoping with cemented backfill/rock fill, supplemented with development waste rock, where possible, to fill open stopes in the thicker ore zones. The remainder of production may come from thinner high-grade zones. Drift-and-Fill methods will be used in these zones.

Two longhole mining scenarios were studied and evaluated for the Platreef 2017 FS: Transverse Longhole Stoping and Longitudinal Longhole Stoping. Longitudinal Longhole Stoping was considered specifically where deposit thickness could not support stope lengths of 15 m or more, normal to the strike, where Transverse Longhole Stoping would not be a suitable option. From the Vertical Miner software (VM) results, three major areas (Zone 1 north section, Zone 4, and Zone 2 central locations), were identified as the ideal candidate for this mining method. Stope shapes were created accordingly. After reviewing the longitudinal stope shapes created by MSO, the entire methodology had to be dismissed due to geotechnical considerations, such as hydraulic radius and mining direction. The ribs of the stope were parallel to the fault planes and the hydraulic radius provided would not support such stope shapes with meaningful sizes.

For the thinner and high-grade areas, two mining methods were evaluated: Cut and-Fill and Drift-and-Fill. After reviewing the created mining shapes (pancakes), it was concluded that Drift-and-Fill mining would be possible and is the best solution due to higher productivities and performance rate.

As Drift-and-Bench mining methods were studied in the Platreef 2015 PFS, this method was also reviewed and evaluated for the Platreef 2017 FS. However, due to low productivity, operational complications, and potential costs associated with this method, it was no longer considered viable and was dismissed.

16.2.4 Mining Shapes Design

Stantec considered the following criteria when determining the best method(s) for ore extraction.

- Safety.
- Maximise productivities by incorporating bulk mining methods and operational flexibility.
- Operating costs.
- Maximise metal grades.
- Maintain high overall recovery rates.
- Minimise overall dilution.
- Prevent surface subsidence.
- Equipment selection and compatibility.

The following mining methods were evaluated for use in the Platreef 2017 FS. All three methods use cemented paste backfill (paste fill) supplemented with a minor amount of development waste rock to fill open stopes.

- Longhole Stoping – For ore zones with vertical thicknesses exceeding 15 m, longhole Stoping methods include the following.
  - Transverse Longhole Stoping
  - Longitudinal Longhole Stoping
- Drift-and-Bench Stoping – For ore zones of intermediate vertical thickness (10-15 m thick).
- Drift-and-Fill Mining (Mechanised Cut-and-Fill) – For thinner portions of the ore zones (less than 10 m thick).

16.2.4.1 Longhole Stoping

The primary mining method selected for the Platreef 2017 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.

Transverse Longhole Stoping

Transverse Longhole Stoping extracts ore in blocks oriented perpendicular to the strike of the mineralised zones. Mining access levels are driven subparallel to the strike of the ore zones in the footwall of the deposit. Drilling and mucking access drifts are then driven from the levels to the top and bottom of the stope, respectively. The levels and access drifts up to the ore contact at the stope boundary are driven at 5 m W × 5 m H arched.

The designs assume a stope design width of 15 m for all transverse Longhole Stopes. They also assume that a minimum vertical height of 15 m is required for transverse Longhole Stopes to ensure the stability of the ground between the drill drift and the mucking drift. Stope lengths vary depending on the thickness of the ore zone from a minimum of 5 m to a maximum of 60 m, as governed by the geotechnical hydraulic radius (provided by SRK). The drilling level at the top of the stope is developed at 6 m W × 5 m H arched. The mucking drift along the stope bottom is also developed at 6 m W × 5 m H arched. Finally, the stopes will be designed with a minimum hanging wall and footwall angle of 55°. Figure 16.7 illustrates the concept for a typical transverse stope, demonstrating the fan and 55° wall drilling pattern design.

Figure 16.7 Typical Transverse Drill Design

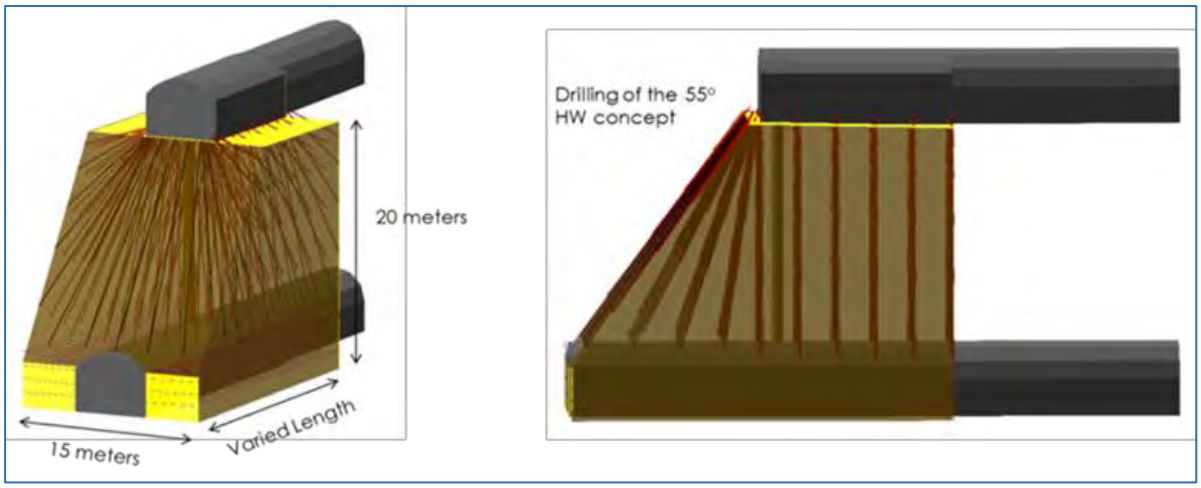


Figure by Stantec, 2017.

Transverse Longhole Stoping is considered a bottom-up method, whereby the lowest stopes are removed first. In primary stopes, the drill drift and mucking drift will be driven perpendicular to the strike of the ore body. The initial bottom drift will be slashed to the full 15 m stope design width immediately in advance of each stope bench blast. Walls will be slashed using a Jumbo drill. The holes will be angled into the wall such that they slash out to the centre drift and will be drilled at least 22° from perpendicular to the drill drift. Once the slash is taken, no personnel shall be permitted in the area since it does not include entry-quality ground support.

Stopes above the bottom stopes only require a drill drift to be developed; the mucking drift will already be established by the stope below and will subsequently be mined to the required 15 m width with the stope blast. Secondary stopes will be mined similarly to primary stopes; however, they will require that the primary stopes on both sides be mined out and paste filled prior to blasting. Figure 16.8 illustrates the concept of mining the transverse stope with primary and secondary stopes.

Figure 16.8 Mine Sequence Concept

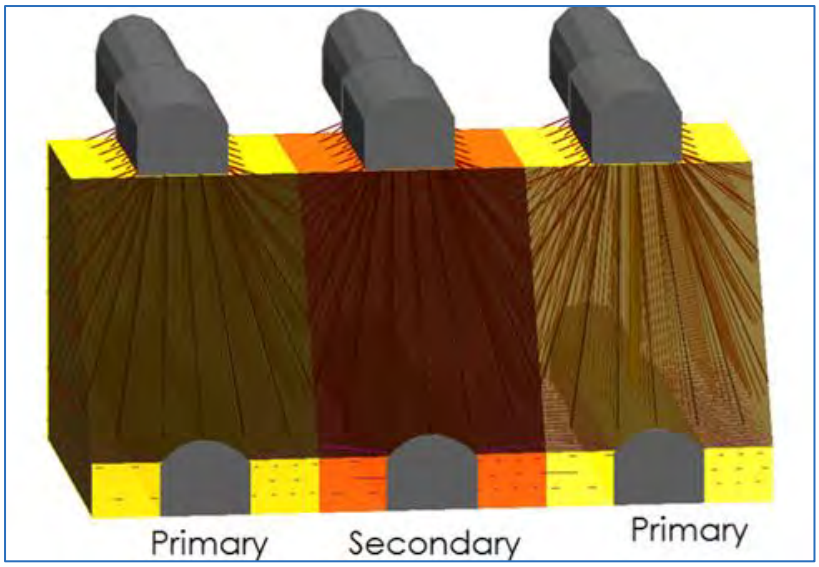


Figure by Stantec, 2017.

After completion of the development, a slot void must be created prior to blasting the entire stope. These slot rings, when blasted, will provide a large enough void to blast the remaining stope. The slot will consist of three production rings located at the end of the planned stope. These rings will be vertical. Any additional fan production rings will be blasted with the cap blast of the slot. The remainder of the stope will be drilled and blasted on retreat from the brow of the initial slot rings to the stope entrance.

To begin the slot sequence, a drop raise will be centred on the second ring in the centre hole. This raise will be 2.4 m × 2.4 m and will consist of 17 holes, including five centre holes. The five centre holes will be reamed to a larger diameter and will serve as cut holes. These holes will be uncharged. (Note an alternative to this type of a drop raise is a large diameter raisebore slot.) The drop raise will be blasted in two lifts with the first being half of the length of the drilled raise. Prior to blasting the final drop raise shot, the remaining rings on the three slot rings will be taken to the same height of the blasted portion of the drop raise. The remaining cap of the slot rings will be removed with the drop raise and any fan holes that are drilled out at the end of the stope. This process will end the slot production sequence and will begin the sequence to extract the remaining stope.

After mining of the stope is completed, it will be backfilled with paste except when waste rock is available for inclusion in the secondary stope backfill mix. In the initial production ramp-up, CRF will be used for backfill until paste fill is available. During the backfill portion of the stope cycle, an engineered bulkhead will be constructed at the stope entrance on the lower drift. Paste fill will then be poured from the top drift into the open stope. Once the paste fill is poured, it will require a cure time of 28 days prior to blasting against the backfill. Other activities such as development and drilling in the adjacent stopes may continue during the filling cycle.

Longitudinal Longhole Stoping

Similar to the Transverse Longhole Stoping method, Longitudinal Longhole Stoping consists of a drill development drift and a mucking drift. These drifts are driven parallel to the strike of the ore body. The width of the drift varies depending on the width of the ore body. The drifts are a minimum of 5 m and up to a maximum of 10 m in width. Stopes that are wider than 10 m are evaluated for Transverse Longhole Stoping or Drift-and-Fill mining methods.

This method requires the drifts be driven to a defined boundary; this boundary is normally either a change in mining method or the end of the mineralised deposit. Then, the stopes are mined out in sequence from the end of the drift to the access. Similar in process to Transverse Longhole Stoping, each stope has a drop raise as a point of beginning for the extraction retreat to the access crosscut. Depending on the length of the stope and in accordance with geotechnical design parameters, should the stope length be excessive, it will be necessary to stop, backfill, and re-establish a new drop raise for the recommencement of extraction. Upon completion, all voids will be paste filled and supplemented with development waste rock when applicable.

Longitudinal Longhole Stoping was rejected as viable for the following reasons.

- Geotechnical constraints dictated by the major faults, subparallel subsets, and hydraulic radius.
- Poor rock mass rating in these areas; stope length would be too short to justify.
- The risk of potential hanging wall failure.
- Low productivities due to the small stopes and the limited number that can be in production.
- High costs due to the above factors.

Longhole Stoping Design

Longhole stope orientation zones (SOZs) are based on the topographic surface created by the top of the T2 mineralisation. As shown by the red arrows in Figure 16.9, transverse stopes are mined up-dip and perpendicular to the contours of the ore body surface (illustrated by the blue lines). The map shows six SOZs (1a, 1b, 2, 3, 4, and 5) created by areas that have similar ore body dip directions.

Figure 16.9 Longhole Stope Orientation Zones

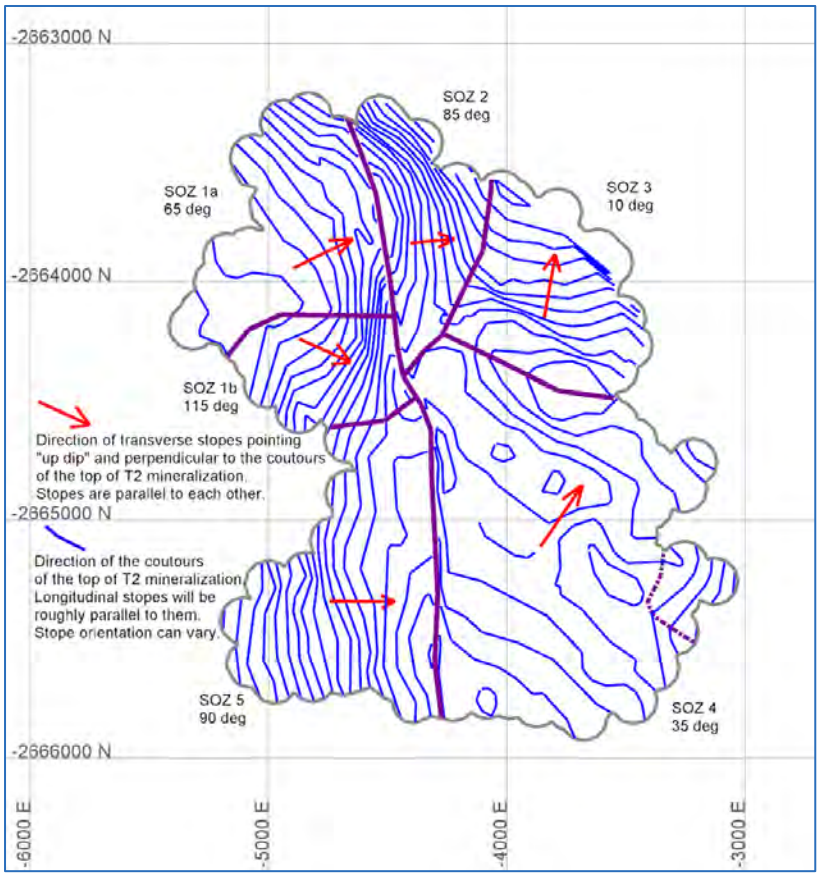


Figure by Stantec, 2017.

Figure 16.10 illustrates Maptek's Stope Optimiser (MSO) stopes that were generated using a $130/t NSR cut-off for a 20 m minimum ore thickness. Longhole Stopes are 20 m H × 15 m W × a minimum stope length of 5 m. The $130 and $100 NSR cut-off stopes are coloured by stope length in Figure 16.10 and Figure 16.11 according to the following.

- Blue: <15 m length
- Green: ≥15 m and <20 m length
- Pink: ≥20 m length

Figure 16.10 Longhole MSO Stopes – $130/T NSR Cut-off

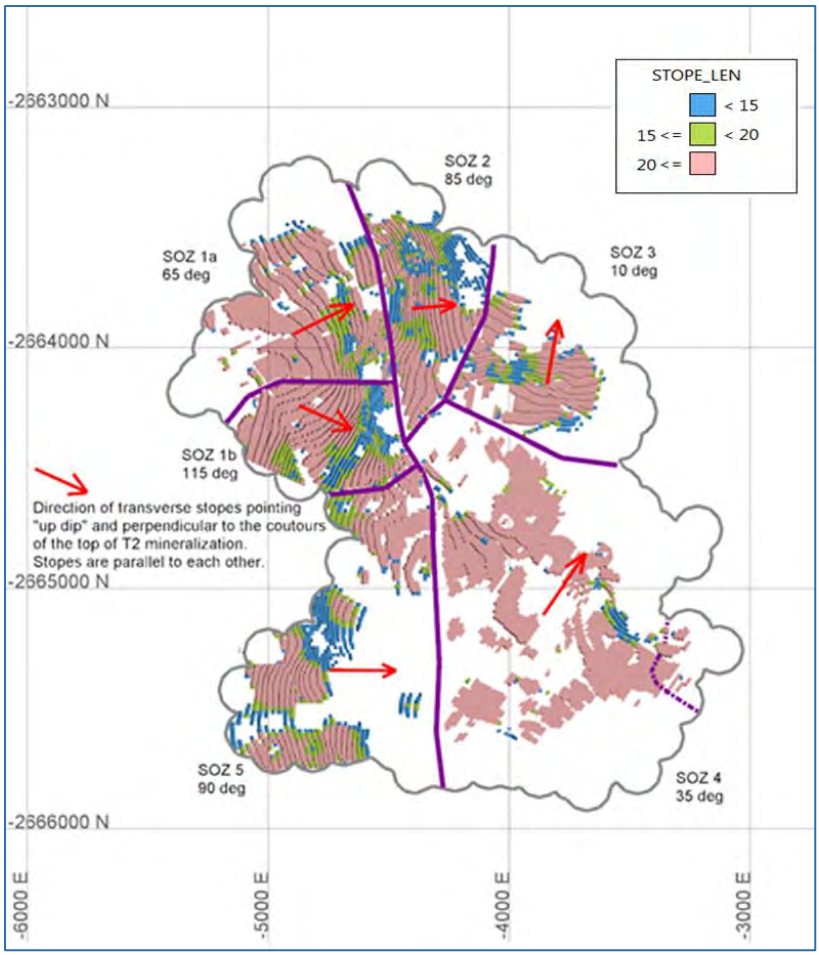


Figure by Stantec, 2017.

MSO Longhole Stopes generated using a $100/t NSR cut-off are shown in Figure 16.11. (Except for the change in cut-off, stope parameters are the same as in Figure 16.10.)

Figure 16.11 MSO Stopes – $100/T NSR Cut-off

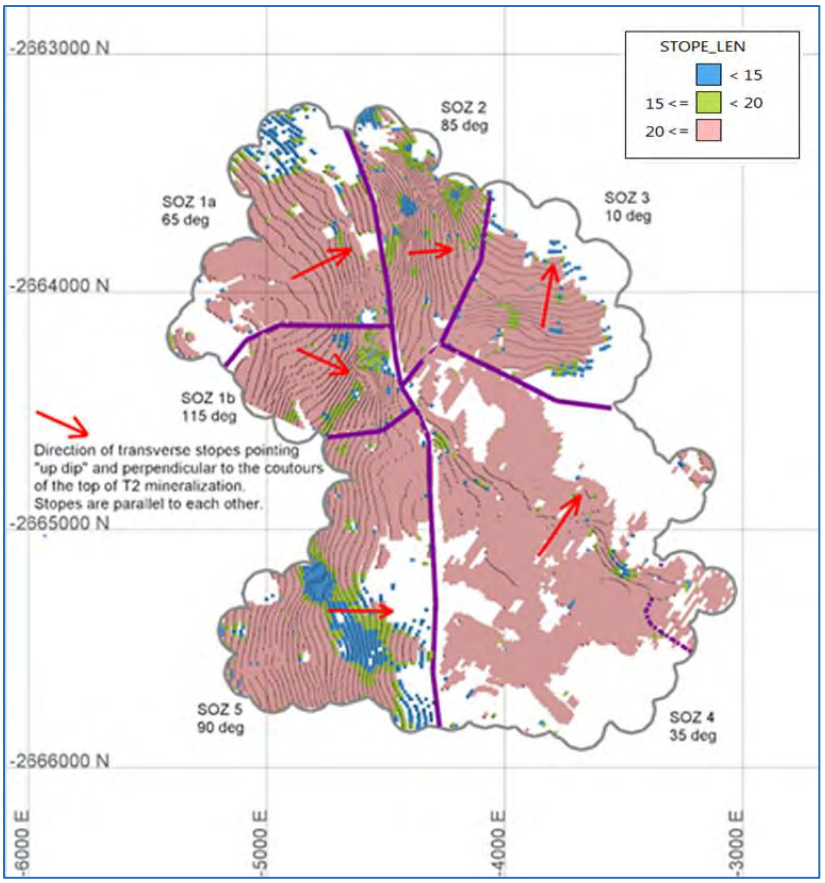


Figure by Stantec, 2017.

Figure 16.12 shows SRK geotechnical domains with green dashed borders. The SRK geotechnical report lists the major joint sets that would affect stope orientation as S1, S2, and S5, as well as domain averages in areas of multiple joint sets. Maximum, minimum, and mean dip directions are shown for each joint set in the circles. Since dip direction is normal to the strike of the joint set, dip direction is the preferred azimuth for stope orientation. In the cases where two joint sets are present, the average dip direction for both structures is shown in a circle below the two joint set circles, along with the new average dip azimuth for both joint sets. SRK's recommended stope orientations are also shown; an orientation of 45° to 90° is recommended for all domains except the southeast, which has a preferred orientation of 0° to 45°.

Figure 16.12 SRK Structural Domains and Preferred Stope Orientations

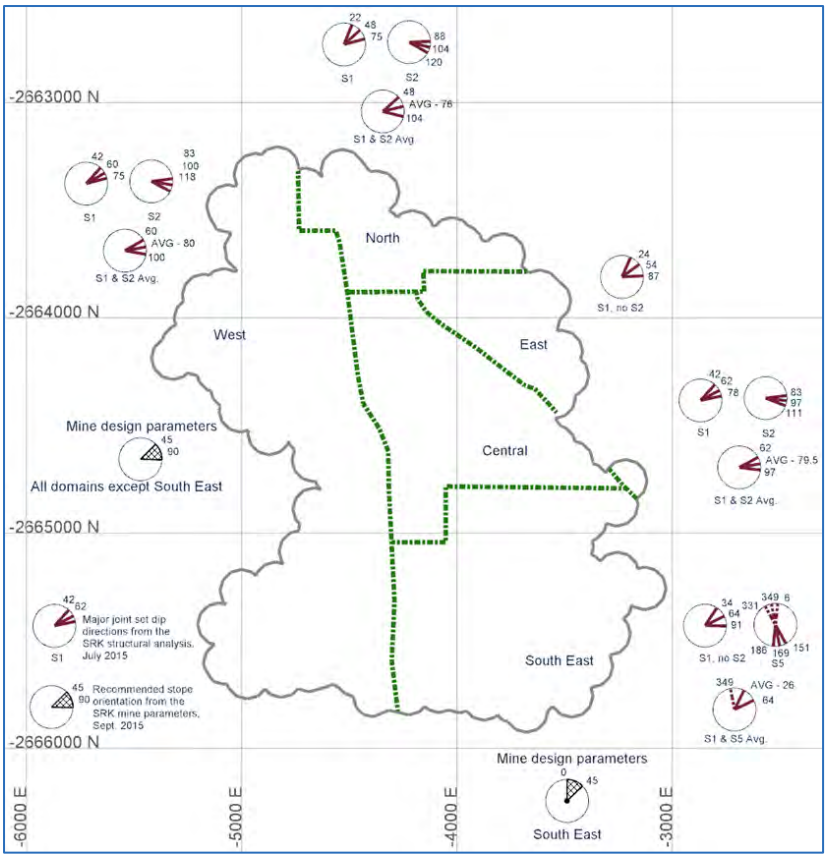


Figure by Stantec, 2017.

16.2.4.2 Transverse Longhole Stoping

In Figure 16.13, the SRK structural domain and preferred stope orientations are combined with the Stantec transverse longhole SOZs (Figure 16.9). For each SOZ, "OK" means that the transverse stope azimuth is within the SRK-recommended stope orientation limits. For the SOZ orientations that are not compliant, a blue arrow shows the corrected azimuth that is required to comply with the limits of the SRK geotechnical parameters (Stantec in red). Most areas are "OK," except for SOZ 1b, SOZ 3, and the northern part of SOZ 4. In these cases, the Stantec stope orientations were rotated 10° to 35° to make them compliant with SRK recommendations.

Figure 16.13 Possible Transverse Stope Orientation Conflicts

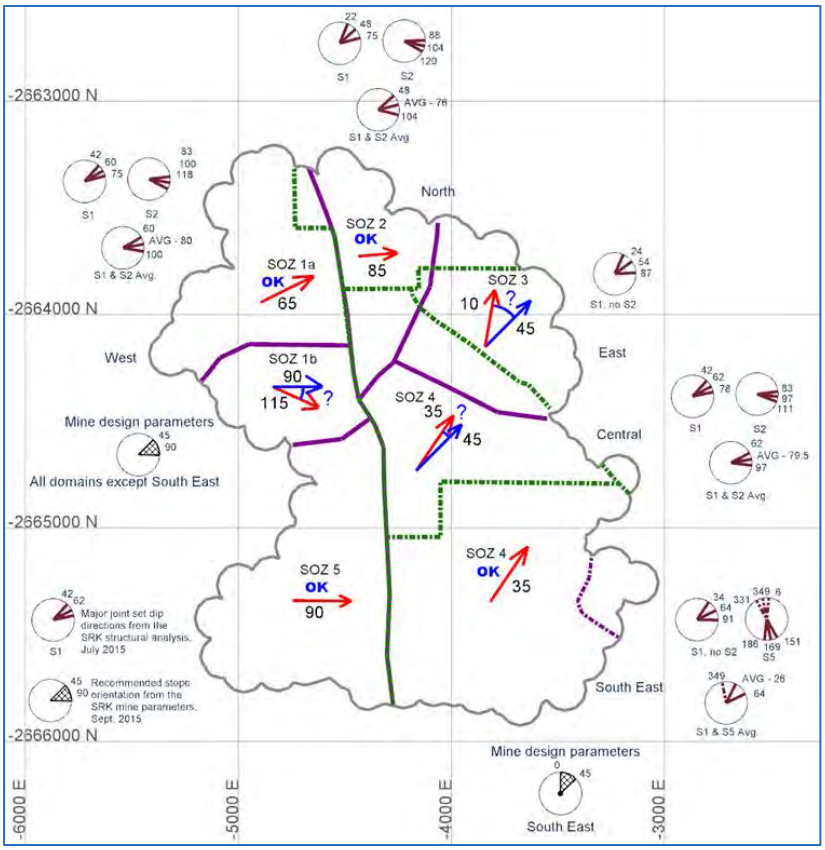


Figure by Stantec, 2017.

16.2.4.3 Drift-and-Fill Mining

In Drift-and-Fill mining, the ore zone will be divided into 5 m high horizontal slices (or lifts), and 5 m wide ore drifts will be mined and backfilled adjacent to one another in a repeating fashion. Upon completion of each drift, a bulkhead will be constructed and the void backfilled with paste fill. After the paste fill sets sufficiently to the required strength (28 days), another drift will be driven next to the fill. Mining will progress in this manner in a chevron pattern until the entire slice of ore is depleted. Where ground conditions permit, mining can be performed using a primary secondary or primary-secondary-tertiary sequence, enabling access to multiple mining faces at all times and allowing greater productivity from an individual ore slice. Figure 16.14 shows a typical primary-secondary-tertiary Drift-and-Fill mining level layout.

Figure 16.14 Typical Drift-and-Fill Mining Level Layout

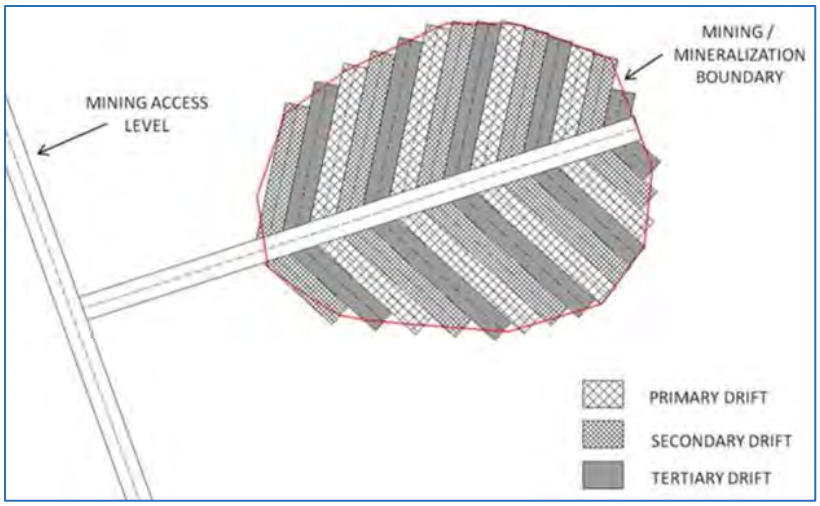


Figure by Stantec, 2017.

Drift-and-Fill mining can be performed in either an overhand or underhand fashion. In overhand mining, as each horizontal slice of ore is exhausted, mining progresses to the slice above the previous section. In underhand mining, after each slice is mined and backfilled, mining progresses to the ore slice below and mining takes place beneath the paste fill. Stantec recommends the use of overhand methods for the Platreef 2017 FS because the lower cement content in paste fill and also the ability to downbreak the development onto the paste fill. Drift-and-Fill mining is a flexible mining method that allows near-complete recovery of the ore zone. Mining is completed with the same equipment used for mine development, and dilution from waste external to the ore zone is minimal. Negatively, productivity is lower than Longhole Stoping due to the smaller blast sizes and sequencing of backfill. Good control of drilling, blasting, and mucking is also necessary to minimise backfill dilution.

16.2.4.4 Drift-and-Bench Stoping

This method was not selected for Platreef due to the ore geometries in the areas with a thickness less than 15 m. Mining using 5 m lifts of Drift-and-Fill and mining bottom-up provided for a better grade profile. The use of Drift-and-Fill also improved the geotechnical and safety considerations related to potential excess hanging wall exposure.

16.2.5 Dilution and Recovery Factors

The following sub-sections summarise the procedures and methodologies used to calculate the dilution and recoveries by mining method.

16.2.5.1 Transverse Longhole Stoping

Dilution

The methods described below were used to determine dilution volumes and grades for Longhole Stoping. For the calculated dilution tonnage, Stantec used the following parameters for the overbreak of the hanging wall and the paste overbreak of the ribs. Primary stopes will have a floor dilution of 0.3 m paste and a hanging wall dilution of 0.5 m of overbreak. Primary stopes rib overbreak is assumed to be ore and therefore will not be calculated as part of the dilution. Secondary stopes will have the same estimated hanging wall and floor dilution as a longhole stope; however, it will also contain rib dilution. Secondary Longhole Stopes will be mined adjacent to paste walls; therefore, the resulting paste wall overbreak will be added to the secondary stope dilution. Table 16.10 lists the different stope sizes and the dilution percentages that will be applied to each primary and secondary stope.

Table 16.10 Dilution Percentage by Stope Length

Stope Length (m)	Primary Stope Dilution Factor (%)	Secondary Stope Dilution Factor (%)
10	8.2	14.0
15	5.9	11.7
20	4.1	9.9
25	3.6	9.5
30	3.3	9.1
35	3.1	8.9
40	2.7	8.6
45	2.6	8.4
50	2.5	8.3
55	2.4	8.2
60	2.3	8.1

The calculated thickness of the overbreak for the paste walls was increased from 0.3 m in the Platreef 2015 PFS to 0.5 m in the Platreef 2017 FS. The change is due to a change in the drill pattern. In the Platreef 2015 PFS, the wall holes were designed parallel to the wall, thus distributing the energy from the blast evenly along the wall in a 15.0 m wide drill drift. In the current design, the holes will be fanned from a centre drift that is only 6.0 m wide. The result is a pattern that drills toward the paste wall, thus increasing the chance that energy from the blast will be directed into the paste fill, thus increasing the chance for dilution. The increase of 0.2 m accounts for this change. A typical drill pattern is illustrated in Figure 16.15.

Figure 16.15 Typical Drill Pattern for Longhole Stoping

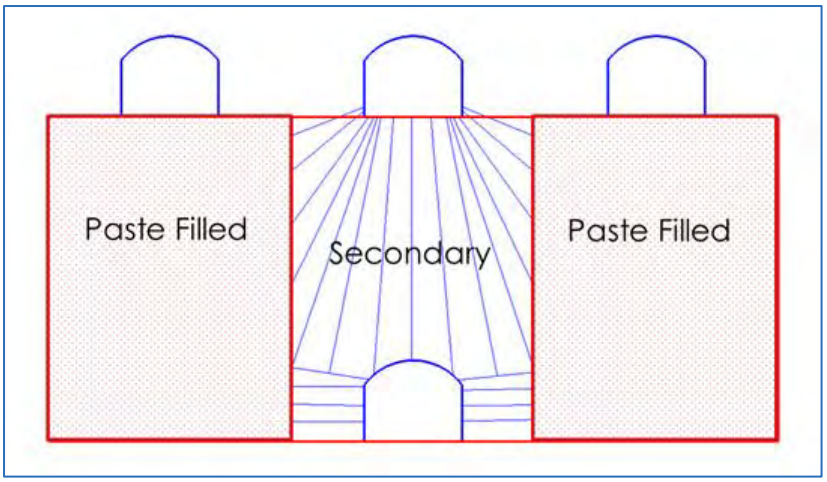


Figure by Stantec, 2017.

To generate an appropriate grade, a Vulcan macro was then developed to create 1 m thick tabular shapes around each stope.

For overbreak of the secondary stopes into the paste fill of the primary stopes, zero grade was assigned to the paste fill dilution. The tonnes for the paste fill overbreak were calculated from the following formula.

One Side Paste Fill Tonnes = length of the stope x overbreak width x stope height x fill density

For the Platreef 2017 FS, stope shapes were generated in Vulcan using MSO and were subsequently imported into Studio 5DP for scheduling in enhanced production scheduling (EPS). Although initial mined resource estimates were run in Vulcan, the final scheduled tonnes and grade were reported from EPS by querying the mined shapes against the Datamine version of the block model. The only exception to this was the stope dilution, which was estimated in Vulcan using the method described above. The dilution grade values were attached as attributes to each Vulcan stope triangulation, and the triangulation attributes were imported into Studio 5DP along with the stope triangulations.

Recovery

The methods described below were used to determine recoveries for Transverse Longhole Stoping.

Development drifts through ore in Transverse Longhole Stopes will be driven 5 m H × 6 m W. All overbreak is expected to be ore and will be included as part of the drift development ore tally. Drift recovery will be 100% since any unrecovered development ore will be extracted, and therefore included, as part of the longhole stope ore recovery.

Ore recovery from Transverse Longhole Stopes was based on a typical stope dimension of 15 m W × 20 m H × 50 m L. It is assumed that the maximum angle the LHD can operate at from the drawpoint will be approximately 45°. Also, cleanup at the back of the stope will be difficult to gauge and will result in additional lost ore recovery.

Production blasting in large excavations presents issues that affect ore recovery, such as oversized rock and un-blasted rock left on the walls of primary stopes without adjacent secondary stopes. For the Platreef 2017 FS, the losses by these factors represent an average ore loss of 2.3%.

Design complications are the final factor in determining how much ore is recovered from a stope. The designed blasted shape does not necessarily recover all the ore. Restrictions on the drill angle and therefore the design blast have a slight difference in shape when compared with the planned stope.

Mucking complications, blasting limitations, and unplanned ore losses result in an overall mining recovery from Transverse Longhole Stopes of 96%.

16.2.5.2 Drift-and-Fill Mining

Dilution

The methods described below were used to determine dilution volumes and grades for Drift-and-Fill mining.

The dilution from Drift-and Fill mining is an average between the primary, secondary, and tertiary drifts. The primary drift will have zero rib dilution, as the overbreak 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 overbreak will be used to estimate the overall dilution from each paste fill rib.

The back dilution for Drift-and-Fill mining depends on whether there is a lift above the stope. If there is no lift above the stope, then the grade will be calculated assuming 0.15 m of overbreak dilution. These dilution tonnes will be added to the overall paste fill dilution to accurately reflect the total dilution for a Drift-and-Fill shape.

For estimating purposes, an average of one-third of the total Drift-and-Fill tonnes is assumed to be overlapping when viewed in cross-section. Additionally, Drift and Fill stopes that overlap also incur a floor paste dilution due to mucking, which is estimated to be approximately one-third of the total development, with an average depth of 0.15 m.

Final dilution percentages will be applied based on the shape size and the individual Drift and-Fill areas. In Table 16.11, the 4.9% represents the average dilution that was applied to the Drift-and-Fill mineable resource. All values will be reflected in the final LOM schedule.

Table 16.11 Drift-and-Fill Stope Dilution Percentages

Drift Type	Paste Dilution (%)	Paste Floor Dilution (%)	Overbreak 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

The dilution grade for the Drift-and-Fill shapes were calculated using slabs 0.15 m thick, created in Vulcan to represent the back overbreak and the floor paste dilution. To determine the difference between the paste dilution and back dilution, the same principal was applied as in the Longhole Stopes. The estimated dilution grade will be calculated by combining the back dilution tonnage with the paste tonnage from the floor and the ribs. The internal rib dilution tonnage will be calculated using the same method described above for the rib dilution. The metal content will then be averaged over the total estimated dilution tonnage for an overall dilution grade.

Recovery

Drift-and-Fill production headings will be driven 5 m H × 5 m W. Mining losses are expected to occur when rock cannot be removed from the floor. This typically happens when the floor is uneven and some of the rock settles in to smooth the floor or is lost into the underlying paste fill. Non-recoverable ore is assumed to be 0.1 m in depth. The resulting ore recovery for Drift-and-Fill mining is 98% of the planned excavation.

16.2.6 Mine Access Designs

Mining zones included in the mine plans for the Platreef 2017 FS occur at depths ranging from approximately 570 m to 1,170 m below surface. Shaft 1 and Shaft 2 are the primary access points to the mine, along with three ventilation raises.

Access from Shaft 1 and Shaft 2 to the mining areas will be via the three principal access/ haulage levels (the 750, 850, and 950 Levels) and a series of interconnecting ramps. Additional mining sublevels will be developed as necessary from the ramps.

16.2.7 Shafts

Shaft functions and design parameters are summarized in Table 16.12. Shaft locations are shown in Figure 16.16.

Table 16.12 Shaft Functions and Design Parameters

Shaft	Function	Diameter (m)	Depth (m)
Shaft 1	Early Mine Development / Escape / Ventilation Intake	7.25	980
Shaft 2	Production – 6 Mt/a (Labour and Material / Mine Services / Ventilation Intake)	10.0	1,104
Vent. Raise 1	Ventilation Exhaust	6.0	950
Vent. Raise 2	Ventilation Exhaust	6.0	800
Vent. Raise 3	Ventilation Exhaust	6.0	725

Figure 16.16 Shaft Locations

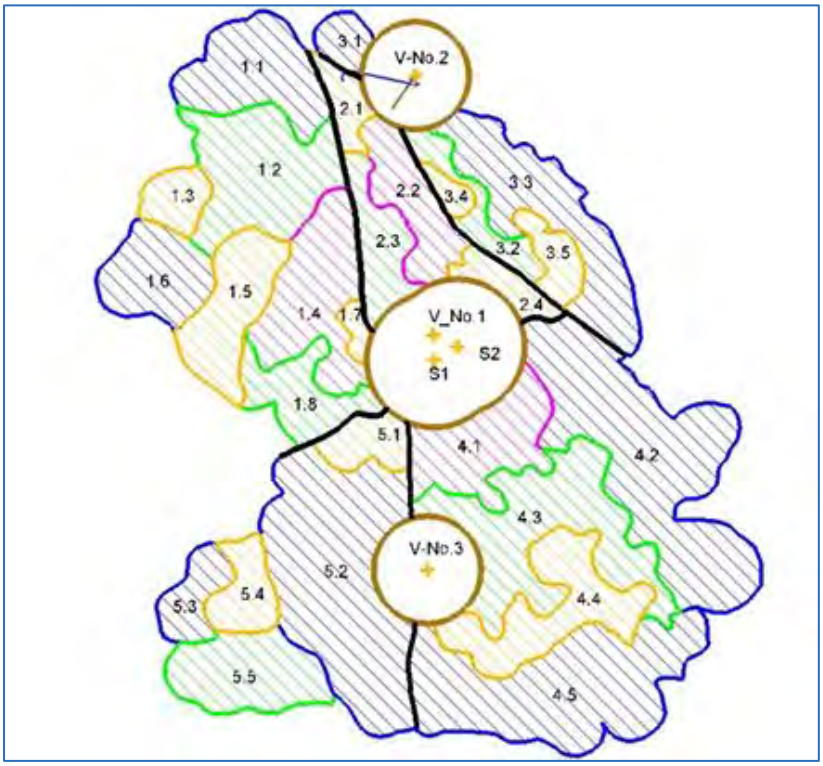


Figure by Stantec, 2017.

16.2.7.1 Shaft 1

Shaft 1 has a 7.25 m diameter, and sinking commenced in October 2015 and is expected to reach a planned depth of 980 m below surface in 2019. Essential mine services and auxiliary cage guides, for emergency egress, will be extended into the mine as the sinking progresses. This auxiliary cage will support initial mine development and later be used for emergency egress during mine production. Shaft services will be installed to supply the mine with essential utilities, including a temporary fuel line for early mine development. Shaft 1 mine services will provide a redundant backup once Shaft 2 mine services are installed and functional and will provide chilled intake ventilation to the deeper and hotter portions of the mine.

Shaft 1 sinking will include initial lateral development at the 450, 750, 850, and 950 Levels. Once shaft sinking is complete, there will be a changeover from sinking buckets to a loading arrangement that will allow for 2,500 t/d of hoisting.

A raise borehole muck pass will be pulled to the 750 Level through the 850 Level from the 950 Level to support fast track development to the high-grade ore zones.

Development will be scheduled to tie into Shaft 2 completion as soon as is practical.

Shaft 1 will be the dedicated cooling and refrigeration ventilation intake that will predominantly service the lower mining zones and have a maximum velocity of 15 m/s (620 m^3/s) to limit losses.

16.2.7.2 Shaft 2

Shaft 2 will serve as the production shaft and primary access to the mine. It will be a 10.0 m diameter, 1,104 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 Mt/a. 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 1,104 m. All equipment and materials will be caged (no slinging). The cage is designed to be chaired with drive on / drive-off capabilities and will be balanced via a counterweight. Shaft 2 includes a 20-person, two-deck auxiliary cage with a 3 t payload.

There will be two main ventilation intake shafts: Shaft 1 and Shaft 2, which will supply fresh air to the mine. Intake from Shaft 2 will be designed for a maximum velocity of 11 m/s (860 m^3/s).

16.2.7.3 Ventilation Raise 1

Ventilation Raise 1 will serve as the primary exhaust for the preproduction development, ramp-up development, and early-stage mine production. In full production, it will also serve as a direct exhaust for the workshops, fueling stations, and mine production. It will be located central to the deposit near Shaft 1 and will have a maximum velocity of 20 m/s (570 m^3/s) to limit losses. It will be developed in two stages. One raisebore hole will be developed from the 750 Level to surface and the other from the 950 Level to the 750 Level. The raise will be 6.0 m in diameter and 950 m deep.

16.2.7.4 Ventilation Raise 2

Ventilation Raise 2 will serve as a dedicated ventilation exhaust during mine production. It will be located near the northern edge of the mining area. The raise will be 6.0 m in diameter, 800 m deep, and designed with a maximum velocity of 20 m/s (570 m^3/s).

16.2.7.5 Ventilation Raise 3

Ventilation Raise 3 will serve as a dedicated ventilation exhaust. It will be located near the southern edge of the mining area. The raise will be 6.0 m in diameter, 725 m deep, and designed with a maximum velocity of 20 m/s (570 m^3/s).

16.2.8 Main Access Levels

Three main access levels are established as primary haulage levels. These are the 750, 850, and 950 Levels. These levels provide air distribution systems and haulage drifts to the mine production areas. All the ramps to the upper and lower ore bodies originate from these levels. The 750, 850, and 950 Levels will be designed flat, and most of the designed ore passes terminate on or near one of these levels. These levels will also tie back into the central infrastructure, where the central grizzlies will allow truck dumping to occur.

16.2.9 Access Ramps

Access ramps will be designed at an overall gradient of 13.5%. They will connect the main haulage levels to the mining sublevels and their associated infrastructure.

16.2.10 Mining Sublevels

The mining sublevels will be designed horizontally at 20 m vertical intervals and are typically driven from access ramps. The 20 m vertical interval was selected based on results from the longhole stoping optimisation study.

Drilling and extraction levels will be driven from the sublevels to the individual stopes. Typically, vent raises are incorporated at the terminus of these drifts to allow for adequate and efficient ventilation during the production phases.

16.2.11 Mine Backfill

The mining methods selected for Platreef require backfill as a support medium. The following three backfill products will be used:

- Cemented Rock Fill (CRF), referred to as Interim Backfill;
- Cemented Paste Fill (CPF), referred to as Long Term Backfill; and
- Waste Rock (from mine development used with CRF and CPF).

CRF will be used until the concentrator is producing sufficient quantities of tailings to produce CPF. The transition from CRF to CPF is scheduled for 01 January 2023. This allows 9 months after the initial mill startup for the mill to achieve stabile operation. Once this occurs, CPF will become the primary backfill method; however, the CRF plant will remain as a backup for the life of the mine. Development waste rock will be used to supplement backfill. Table 16.13 presents the type and quantity of backfill required for the life of mine.

Table 16.13 Fill Requirements by Year (Tonnes ×1,000)

Backfill Method	2019	2020	2021	2022	2023	2024	2025
CRF	–	–	22.5	799.1	–	–	–
CPF	–	–	–	–	2,029.7	2,297.3	2359.5
Waste Rock	–	–	–	127.6	304.4	275.9	301.3
Total Backfill	–	–	22.5	926.7	2,334.1	2,573.2	2,660.8
Backfill Method	2026	2027	2028	2029	2030	2031	2032
CRF	–	–	–	–	–	–	–
CPF	2,294.0	2,247.3	2116.9	2316.9	2278.8	2,105.4	2337.8
Waste Rock	381.5	427.7	415.2	448.4	482.0	425.5	381.2
Total Backfill	2,675.5	2,775.0	2,532.1	2,765.3	2,760.8	2,530.9	2,719.0
Backfill Method	2033	2034	2035	2036	2037	2038	2039
CRF	–	–	–	–	–	–	–
CPF	2,206.9	2,192.9	2,307.0	2,125.0	2,216.4	2138.4	2,222.0
Waste Rock	439.0	463.2	450.4	483.1	521.6	432.2	485.0
Total Backfill	2,645.9	2,656.1	2,757.4	2,608.1	2,738.0	2,570.6	2,707.0
Backfill Method	2040	2041	2042	2043	2044	2045	2046
CRF	–	–	–	–	–	–	–
CPF	2,338.3	2,094.2	2,236.9	2,313.9	2,311.6	2,178.0	2,343.2
Waste Rock	505.4	458.6	432.8	394.4	425.6	391.6	387.8
Total Backfill	2,843.7	2,552.8	2,669.7	2,708.3	2,737.2	2,569.6	2,731.0
Backfill Method	2047	2048	2049	2050	2051	2052	2053
CRF	–	–	–	–	–	–	–
CPF	2,472.3	2,384.6	2,237.11	2,359.0	2,273.9	2,294.5	1,976.5
Waste Rock	480.7	437.9	471.2	488.3	415.9	367.1	306.4
Total Backfill	2,953.0	2,822.5	2,708.3	2,847.3	2,689.8	2,661.6	2282.9

16.2.11.1 Assumptions and Basis of Design

The backfill feasibility study completed is based on certain key assumptions which were defined through collaboration with Ivanplats throughout development of the feasibility study.

- The unconfined compressive strength (UCS) target for the CRF and CPF was provided by SRK, and Golder determined the binder content from CRF test work undertaken by Geopractica, and CPF test work undertaken by Golder.
- The tonnages for the placement of CRF are calculated based on the replacement factor values derived in the design criteria.
- Uncemented fill refers to waste rock without the addition of binder.
- Paste Fill will always require binder for placement underground. For secondary stopes, there is a minimum amount of binder required to mitigate liquefaction.
- For the purpose of estimating binder requirements, the backfill requirements are based on the largest stope sizes for both CRF and CPF.
- The placement of CRF is based on 100% CRF in the primary stopes. The secondary stopes will be filled with 50% waste rock and a combination of uncemented rock fill (RF) and cemented rock fill at the top of the stope to provide a solid floor when mining the stope above. 100% CRF is required for ore development and drift and fill mining prior to commissioning of the CPF plant.
- Once the CPF plant is in operation, the CRF plant may continue to operate and is maintained as an alternative backfill source if required.
- Internal mine boreholes can be drilled up to 300 m in length (if required) to service the paste underground distribution system (UDS). This length of borehole is anticipated to be the maximum required for the UDS.
- All boreholes for the CPF UDS are drilled to a target dip angle of 70º but some variance occurs within the UDS.
- When not backfilling, the concentrator tailings will be diverted to the tailings storage facility (TSF) through the concentrator's discharge system developed by DRA.
- During the CRF period, secondary stopes will be filled with 50% CRF and 50% waste rock. During the CPF period, secondary Longhole Stopes will be filled with 67% CPF and 33% waste rock and the Drift-and-Fill Stopes will be filled with 75% CPF and 25% waste rock.

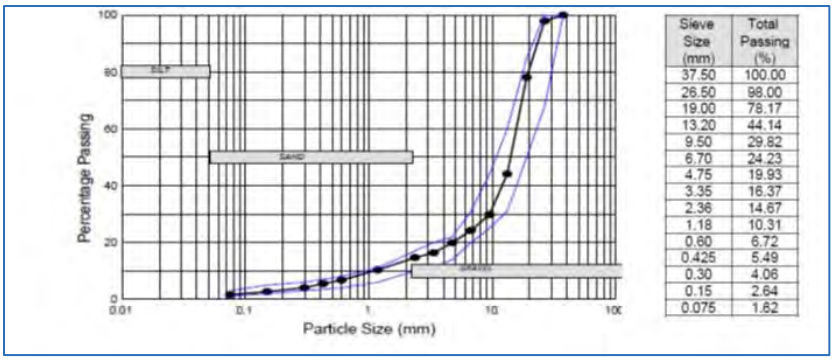
16.2.11.2 Test work

Laboratory scale test work has been undertaken to determine material characterization and the Unconfined Compressive Strength (UCS) for both CRF and CPF and the settling, filtration and rheological parameters for CPF.

16.2.11.3 Cemented Rock Fill (CRF)

Waste rock samples were obtained and graded for UCS testing. Figure 16.17 presents the particle size distribution (PSD) of the test sample.

Figure 16.17 Waste Rock Sample Gradation for Tests



Sieve Size (mm)	Total Passing (%)
37.50	100.00
26.50	98.00
19.00	78.17
13.20	44.14
9.50	29.82
6.70	24.23
4.75	19.93
3.35	16.37
2.36	14.67
1.18	10.31
0.60	6.72
0.425	5.49
0.30	4.06
0.15	2.64
0.075	1.62

Figure by Golder 2017

The UCS results obtained are presented in Table 16.14. Lafarge Powercrete 42.5R cement was used in the test work. The results were found to be suitable for the application. Based on the test work, a binder addition rate of 4.5% would provide the calculated UCS of 420 kPa

Table 16.14 CRF UCS Results

Test Core ref	Age (days cured)	Cement content (%)	UCS (kPa)
1 (3)	7	6	934
2 (4)	7	6	651
3 (5)	7	6	594
4 (9)	28	6	2,433
5 (10)	28	6	2,660
6 (11)	28	6	2,207

16.2.11.4 Cemented Paste Fill (CPF)

Test work was undertaken by Golder on Platreef Mine (Platreef) tailings during the Platreef 2017 FS paste fill study to allow the paste plant design to proceed to a feasibility level. The test work for settling and filtration have been reported in Section13.9 of this report and need not be repeated.

Viscocity and Dynamic Yield Stress

The Platreef tailings exhibit a normal increase in both viscosity and dynamic yield stress as they dewater into the paste range. There is negligible evidence of shear thinning.

The viscosity and dynamic yield stress for the feasibility sample (1530678) have increased from the pre-feasibility sample (13-6154-22). The values obtained in both the feasibility and pre-feasibility are presented for comparison purposes in Figure 16.18 and Figure 16.19. The viscosity and dynamic yield stress values are two key factors used to calculate friction losses for that material while it travels through UDS pipelines. With the increase in viscosity and dynamic yield stress, the friction loss values will increase from those in the pre-feasibility study but are still in a range acceptable for paste. For Platreef, those friction loss values are used for the flow model calculations for the UDS for cemented pastefill.

Due to the extents of the mine it is anticipated that a higher slump paste will need to be delivered to certain areas in order to manage the pressures encounter in the UDS. Depending on the results of the flow model calculations, intermediate booster pump stations may be required to deliver paste to the extents of the ore body without over pressurizing the system.

Figure 16.18 Bingham Yield Stress Comparison

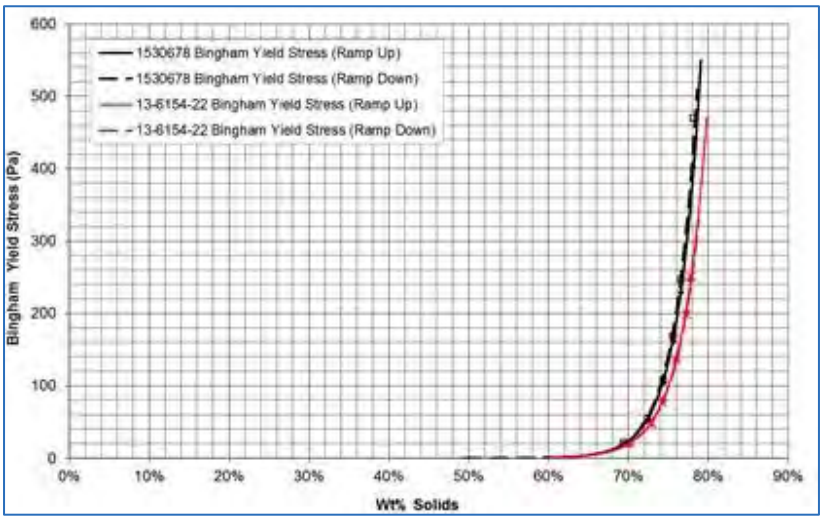


Figure by Golder 2017

Figure 16.19 Bingham Viscosity Comparison

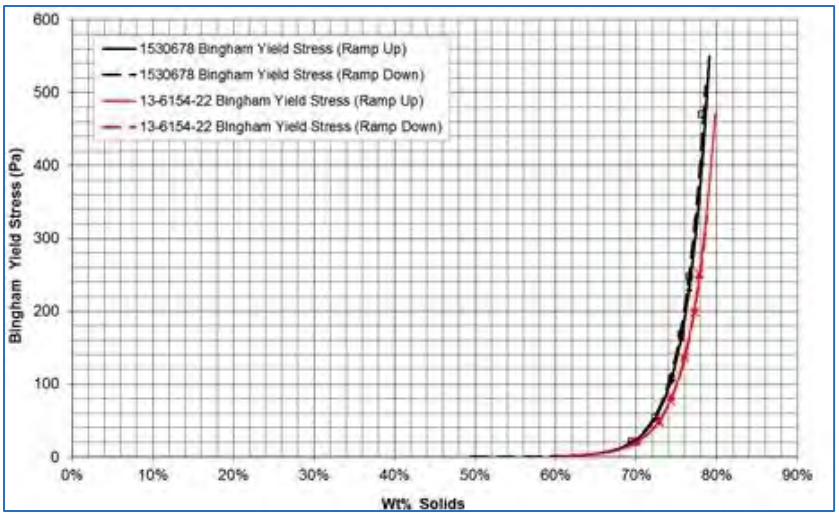


Figure by Golder 2017

UCS Testing

The amount of binder added (solids wt%) for the UCS testing is based on achieving calculated mining backfill UCS requirements provided to Golder by SRK.

Generally speaking, the binder addition rates are on the higher end of the spectrum for the observed UCS values when compared to tailings with similar properties contained in Golder's lab database of actual results.

Figure 16.20 and Figure 16.21 present the relationship between binder addition rate and ultimate UCS for backfill strength design per the calculated mining backfill UCS requirements.

Figure 16.20 UCS vs. Binder Content - 7" (178 mm) Slump

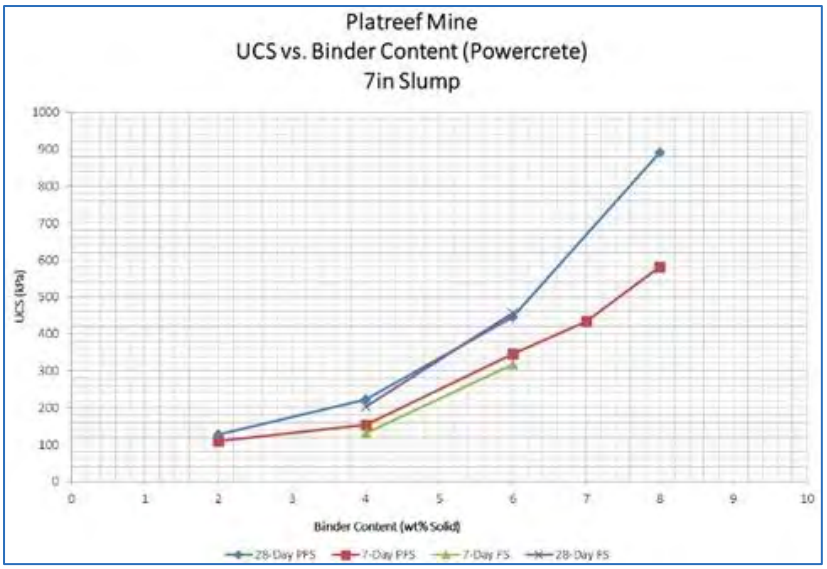


Figure by Golder 2017

Figure 16.21 UCS vs. Binder Content - 10" (254 mm) Slump

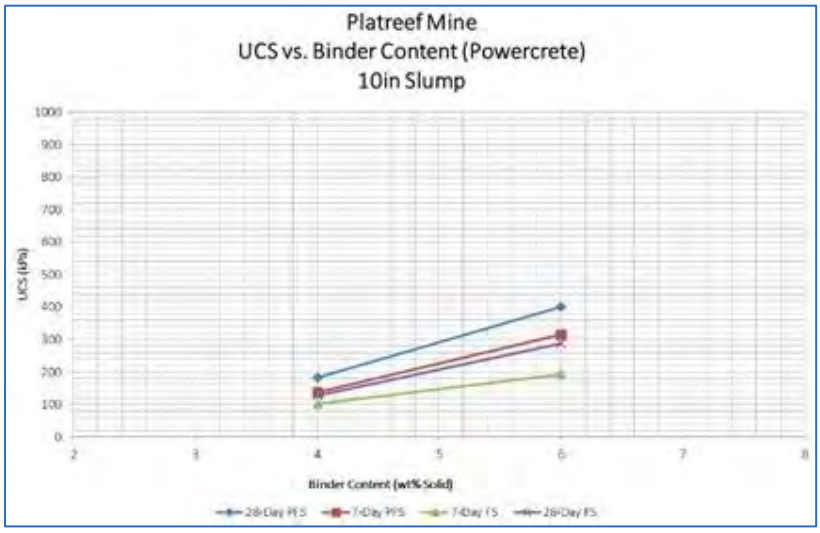


Figure by Golder 2017

16.2.11.5 Laboratory Materials Testing Conclusions and Recommendations

The following conclusions can be drawn from the laboratory assessment conducted on the feasibility samples:

- The feasibility tailings can be initially dewatered through the use of available thickener technology and the same design of thickener is still applicable as was selected in the pre-feasibility study.

- The filtration of tailings needed to produce a consistent paste consistency can still be accomplished with vacuum disc filtration. However, the filtration rate has decreased in the feasibility sample therefore a larger disc filter will need to be introduced into the process.
- The tailings are still amenable to the formulation of paste using the aforementioned technologies.
- The static and dynamic yield stresses and dynamic viscosity for the feasibility samples will still require the use of positive displacement pumps in the paste distribution system. The values observed and the extents of the ore body, will require the slump to be varied between 178 and 254 mm (7 to 10") dependent upon the proximity of the stope being filled. It is quite possible that booster pump stations will be required in the underground distribution system in order to control overall distribution system pressures.

The binder addition rate for cemented pastefill in the primary stopes is estimated to be 5.0 wt%, for a 178 mm (7") slump paste for the bulk pour (main body of the stope), in order to attain a UCS of 0.3 MPa after 28 days curing. The binder addition rate for cemented pastefill in the secondary stopes is estimated to be 4.4 wt% in order to attain a UCS of 0.17 MPa after 7 days curing. The details of the binder addition rates for 178 mm (7") and 254 mm (10") slumps of paste, including the binder additions rates for different types of pours (plug, body, cap, etc.) are included in Golder's technical memo issued to Ivanplats on September 2, 2016 (Golder memo "Estimated Binder Addition Rates for Platreef Paste Backfill - Feasibility Study").

16.2.11.6 Binder Addition Rates

Binder addition rates were measured from the UCS results for CPF obtained on the samples tested during the feasibility study. The target strength of 0.3 MPa was provided to Golder by SRK Report, "482446 M5 Platreef FS Mine Geotechnical Parameters rev2" and based on designed stope dimensions of 15 x 20 x 60 m (W x H x L).

A summary of the binder addition rates for CPF at 178 mm (7") and 254 mm (10") slumps, and for CRF are presented in Table 16.15, Table 16.16 and Table 16.17.

Table 16.15 Longhole Stopes Paste 178mm Slump – Target freestanding strength - 0.3 MPa

Binder Addition Rate (wt%)	Days Cured	Primary Purpose	Description
5.8	7	Paste plug Primary and Secondary Stopes	Poured to 1.5 m height above the undercut brow
5.0	28	Bulk pour	Remaining volume in stope
6.5	28	Mucking Floor	Capping of bulk pour for mobile equipment target 0.5 MPa – 1 to 1.5 m thickness
4.4	7	Liquefaction Mitigation Secondary Stopes	To attain 170 kPa
3.5	28	Liquefaction Mitigation Secondary Stopes	To attain 170 kPa

Table 16.16 Longhole Stopes Paste 254mm Slump – Target freestanding strength - 0.3 MPa

Binder Addition Rate (wt%)	Days Cured	Primary Purpose	Description
7.4	7	Paste plug Primary and Secondary Stopes	Poured to 1.5 m height above the undercut brow
6.4	28	Bulk pour	Remaining volume in stope
8.0 – 8.5	28	Mucking Floor	Capping of bulk pour for mobile equipment target 0.5 MPa – 1 to 1.5 m thickness
5.5	7	Liquefaction Mitigation Secondary Stopes	To attain 170 kPa
4.9	28	Liquefaction Mitigation Secondary Stopes	To attain 170 kPa

Table 16.17 Longhole Stopes CRF – Target freestanding strength - 0.3 MPa

Binder Addition Rate (wt%)	Days Cured	Primary Purpose	Description
4	28	Bulk pour	Minimum recommended binder addition rate for longhole stopes

16.2.11.7 Cemented Rock Fill

The basis of the CRF design criteria is the maximum monthly backfill demand until paste backfill is utilized.

A summary of the design criteria used in the engineering design for CRF equipment sizing considerations is presented in Table 16.18.

Table 16.18 CRF Process Design Criteria Summary

Description	Tailings	Units
Ore specific gravity	3.14	
Maximum long hole stope dimensions	15 x 20 x 60	m (W x H x L)
Maximum long hole stope volume	18000	m3
Maximum monthly ore production - 12/1/2022 (1)	202381	tonnes
Maximum daily ore production – 12/1/2022 (1)	6746	tonnes per day
Average daily CRF production – 12/1/2022(2)	3384	tonnes per day
Average daily RF production – 12/1/2022 (2)	1128	tonnes per day
Peak daily CRF production (plant boiler plate)	4000	tonnes per day

Notes:

1. Prior to full production of the CPF plant per Stantec "Primary_Secondary_LH Tonnes_rev0_Sept22_16".
2. Based on fill replacement factor and all values are calculated based on 30 operating days per month.

The following Sections described the CRF preparation processes.

The colloidal mixing plant on surface combines clean water from a clean water tank, cement from cement silos, and a chemical hydration retardant from a chemical retardant tank into a slurry consistency and it then discharges this slurry to an underground agitated tank via a borehole. The slurry is stored underground in an agitated tank until being pumped into a twin shaft continuous mixer containing crushed and screened waste rock.

The surface components for the cemented rock fill slurry preparation plant will be permanently installed inside a standard 9.1 m "high-cube" shipping container and shipped to site pre-tested. The plant is supplied complete with motor starters, PLC system, instrument air dryer, hydration retarder addition system, and pumps. This design minimises the installation and commissioning time required on site. An air receiver, clean water storage tank, clean water pump, and sump pump will be supplied separately and installed outside the "high-cube". Water and compressed air are used in the flushing process for surface and underground lines and borehole. This system has been successfully used in other CRF applications internationally.

The cement slurry is transferred underground from the colloidal mixer on surface through a 76 mm (3") borehole to the underground agitated storage tank. The waste rock travels down a 1.5 m diameter bored raise. Both the slurry and waste rock go from surface to the 850 level. In the underground facility aggregate is received from the aggregate pass, transferred along a conveyor into a mixer where the slurried cement is added. The mixed CRF is then loaded into trucks for transport to the stopes.

16.2.11.8 Cemented Paste Fill Design

The process design of the CPF plant to be used as the Long Term Backfill System was developed during pre-feasibility and was refined in the feasibility. It was determined that the overall process remains adequate to deliver CPF for the Life of Mine.

A summary of the design criteria used in the engineering design for CPF equipment sizing considerations can be found in Table 16.19.

Table 16.19 CPF Process Design Criteria Summary

Description	Quantity	Units
Years of mine production (LOM)	32	years
Ore specific gravity (1)	3.14	
Tailings solids specific gravity (1)	3.19	
Average ore production annually	4	Mega tonnes per year
Average ore production daily	11111	tonnes per day
Paste plant utilization	53	%
Concentrator tailings discharge density (1)	27	wt% solids
wt% passing 20 µm (2)	30	wt%
PSD (D80) of tailings (2)	105	µm
Paste backfill solids content at 178mm (7") slump	77.5	wt% solids
Paste backfill solids content at 254mm (10") slump	76.6	wt% solids
Paste bulk density at 178mm (7") slump	2.14	tonnes per m3
Paste bulk density at 254mm (10") slump	2.1	tonnes per m3
Average daily binder consumption (3)	395	tonnes per day
Maximum daily binder consumption (3)	463	tonnes per day
Maximum daily CPF production (3)	15888	tonnes per day
Flocculant addition rate	25	grams per tonne solids

Notes:

1. As provided by DRA. Value reported here is an average of the ore type SG's Tailings SG is as per DRA PFD. Concentrator tailings % solids per DRA PFD.
2. As measured in Golder lab testing. DRA measured a PSD in the 80-85% passing 75micron range.
3. Paste at 77.5 wt% solids and 6 wt% binder addition rate (nominal). Maximum daily consumption considers filling secondary stope with low binder content paste as well as waste rock disposal as an opportunity.

The paste plant process has been designed to prepare a backfill product made from concentrator tailings alone and no additional components, such as sand or aggregate, have been included in the process. During pre-feasibility, it was established that the paste preparation process would be split into two separate and somewhat independent preparation streams and underground distribution systems.

The reasons for this were twofold:

- Ivanplats has requested that the plant have the flexibility to backfill two independent areas (stopes) underground simultaneously. These areas may be a significant geographical distance apart and may have different unconfined compressive strength requirement, therefore would require different binder addition rates; and

- The volumetric throughput for the paste preparation process exceeds the capacity of any high shear twin shaft mixer available on the market today. Splitting the plant's throughput into two largely independent streams allows the use of the largest mixer on the market for each of the two streams.

The following processes are included in the design of the paste plant:

- Thickening
- Filtration Dewatering
- Mixing
- Paste Pumping
- Binder System
- Air System

16.2.11.9 Underground Paste Distribution System

The pipeline friction losses applied in the flow model evaluation have been estimated based on the results of the lab testing program on the Platreef tailings and Golder's database of tailings with similar properties. As a result, the friction factors for the paste at a 178 mm (7") and 254 mm (10") slump have been estimated at 11.5 and 7.9 kPa /m respectively. The paste will be delivered in 200 mm (8") boreholes and pipelines. It is important to note that the Platreef tailings tested during this study have different characteristics than those tailings samples tested in pre-feasibility. The change in tailings characteristics have resulted in higher friction factors that those estimated during pre-feasibility.

Flow models and distribution system design, as well as the pressure profiles were completed to the extents of all the main ore zones for the life of mine of the underground distribution system.

Based on the results of the flow model it is estimated that 90 bar positive displacement pumps will be required over the life of the mine. In all areas of the mine, 178 mm (7") and 254 mm (10") slump pastes will be delivered underground. Generally speaking, the 254 mm (10") slump occurs at the very bottom sections of each area and is required because pressures in the line are above the design limit of the pipe when delivering 178 mm (7") slump. Therefore, as the paste distribution gets further away from the paste plant, the higher the slump will be necessary. It should be noted that as the slump increases, so does the amount of binder required to maintain the same target backfill strength. Therefore, during plant operation, the slump requirement should be determined on a stope by stope basis.

The positive displacement pumps deliver paste to the collars of two in-service boreholes that dip at 70 degrees from horizontal. In total there are 4 boreholes: two that can service the North area of the mine and two that can service the South area. Redundancy has been built in by providing a standby borehole to each area. The boreholes from surface are fitted with a 200 mm (8") ceramic lined casing for longevity as well as to mitigate the ingress of water into the paste which would detrimentally effect paste quality. The ceramic lining is 6 mm, resulting in a total internal diameter (ID) reduction of 12 mm from the 200 mm (8") pipe. The life of the ceramic lining is estimated to be 10 Mt and boreholes will be drilled to a nominal 300 mm diameter in order to insert the ceramic lined pipe.

The flow model indicates that underground booster stations will be required. As a minimum, one servicing the North area and one servicing the South area. The booster station in the North is required to service the upper part of Zone 3. Zone 3 is located above the grade level of the main access drifts. As these pump stations are expensive to construct and operate, the next phase of rheology testing will be used to determine whether a mitigation method can be introduced to remove the necessity for the pump stations. Thus, paste will be delivered upwards and will require a booster station. Underground booster stations are not desirable as they consume a large amount of power, are expensive to maintain and are difficult to operate. Additional flow loop testing is recommended when the pilot plant is commissioned to provide the required sample quantity for testing. This test work will validate the existing rheological design as well as investigate other mitigating measures to remove the booster stations, such as rheology modifiers.

As has been stated, the pipeline friction losses have been estimated and should be validated through lab based flow loop testing, and flow models adjusted accordingly, in the next phase of the project (basic or detailed engineering). Likewise, the pipeline routing should be updated in the next phase to reflect any changes in the mining areas and the paste slumps should be determined on a stope by stope basis during operations.

Cameras will be located at the paste discharge point as well as at the backfill barricade underground. These cameras will link into the proposed leaky feeder system within the underground workings.

16.2.11.10 Further Backfill Work and Studies

The combination of cemented rockfill (CRF) as an intermediate backfill method, and cemented paste fill (CPF) as the long term backfill method, provides the most cost effective backfilling processes for the Platreef project when compared to: other backfill methods; backfill material availability; and mine backfill requirements and based on the assessments completed by Golder in the pre-feasibility and feasibility level assessment for the Platreef Project.

Since CPF is an "engineered" backfill, and can be designed to accommodate the required UCS target strengths based on the required target UCS, the economic returns associated with its use in particular mining methods should be assessed for viability on a case by case basis.

The design and use of paste backfill barricades varies between conventional shotcrete and waste rock. The latter provides significant cost savings and have been in successful operation, for example at the Zinkgruven Mine in Sweden for longhole open stope mining. Consideration for their use at the Platreef Project should be investigated to reduce operating costs. The waste rock barricades may be most applicable to the "short cycle time" drift and fill mining sequence.

The binder requirements for CRF have been estimated based on UCS results obtained during the pre-feasibility stage of study. It is recommended that UCS testing be conducted at various binder quantities to verify the assumptions.

The placement of CRF is based on 100% CRF in the primary stopes and 50% CRF / 50% Waste Rock in the secondary stopes for longhole mining. At the next stage of engineering, the amount of CRF and Waste Rock for secondary stopes should be evaluated more accurately based on the mining sequence. This presents an opportunity for Platreef to reduce the cost of backfilling.

The binder requirements for CPF have been estimated based on the UCS results measured in the Sudbury Golder's laboratory during Platreef 2017 FS. At the next stage of engineering, binder requirements for primary and secondary stopes, along with binder requirements for the different stages of pouring (plug, body, cap, etc.) should be defined more accurately for the opportunity to reduce the cost of backfill.

Although it is not believed to be currently available in the mine's immediate area, initial indications are that a binder consisting of a blend of 90% ground iron blast furnace slag and 10% normal Portland cement (90/10) could be beneficial in reducing binder consumption, and associated cost, over the life of the mine. The economic merits for the use of this binder should be evaluated in the next phase of laboratory testing. In a previous paste plant design, the availability of this binder was also thought to be non-existent until the economic merits of its supply were evaluated and its use was implemented.

The pipeline friction losses applied in the flow model evaluation have been estimated based on the results of the lab testing program on the Platreef tailings and Golder's database of tailings with similar properties. The conclusion was that booster stations are required to backfill the extremities of the orebody as well as Zone 3 since it is above the 750 level. It is recommended that flow loop testing be conducted in the next phase of the Platreef project, being when the pilot plant has been commissioned in order to obtain a new tailings samples. This will allow a better quantification of the anticipated pipeline friction losses and provide a better indication of if, when, and where booster stations for paste distribution are required.

In some ore zones, long paste delivery times are observed in the flow models. It is recommended to investigate the addition of a binder hydration retardant be added to the paste product on surface to prevent hydration during transport. It is also recommended that testing be done with a retardant to determine the yield stress over time, validating the maximum time that the paste can be in transit from the plant to the stope.

In the North, part of Zone 3 is above elevation (up-grade) from the main access. Golder is recommending that a booster station be added to deliver paste to these upper stopes. However, it is recommended that Platreef examine the possibility of drilling a separate borehole from surface to the top part of Zone 3 to determine if there is an economic benefit.

In most areas of the mine, a 254 mm (10") slump is required to deliver the paste fill, along with booster stations in some of the areas. At the next stage of engineering, the location of the surface boreholes should be re-examined. Locating the boreholes in separate areas above both the North and South sections of the orebody should be considered.

The UDS is based on the installation of 200 mm (8") Sch 80 carbon steel (CS) pipe throughout the distribution system, with the exception of branch lines going in to the stopes. In the next stage of engineering, the use of 200 mm (8") Sch 40 CS pipe should be considered for the installation on the levels (i.e. Level Piping). The Sch 40 pipe, due to its slightly larger internal diameter (ID), will reduce the friction losses and pressure in the system and will also present an opportunity for cost savings. This can be examined once the UDS routing is further defined.

16.2.12 Mine Equipment Requirements

16.2.12.1 Introduction

The mobile equipment for the mine design focused on primary development and production equipment. The secondary fleet to support the production and development was simulated and evaluated as a separate exercise prepared by DRA.

The equipment was selected to meet mine design and production requirements. The identification of the specific manufacture(s) and the costing of the equipment was determined through a tendering process. Equipment quantities were determined based on operating hours required and location. Swing units were then added to account for overall availability.

Primary equipment is best described as equipment that is needed to either develop drifts or Longhole Stopes. The absence of any of this equipment would stop the advancement of the mine. Secondary equipment is best described as equipment that supports the crews on primary equipment. The absence of this equipment would not stop mine advancement but would affect the performance of the primary equipment.

16.2.12.2 Mobile Equipment

The primary mobile equipment determined for the mine design covers both development and production. The secondary equipment to support these activities are covered by others.

The development equipment is based on a fleet of Jumbos, mechanised rock bolters, and explosives trucks. Also included in the development fleet are cable bolters for large spans and intersections. A complete equipment list is in Table 16.20.

The Jumbos will be two-boom units with a 6.49 m drill boom. These units will be mechanised and allow for computer automated drilling and alignment. The rock bolters will be designed to allow for the variety of bolts listed in the geotechnical portion of the mine design criteria, 2.4 m resin rebar, 3.0 m resin rebar, and other miscellaneous support types. The bolters will have carousels to load a quantity of bolts to minimise delays or worker exposure during the bolting process. The emulsion loaders are one of the units that will require operators to work most the time outside of the unit's operator's compartments. The certified explosive loaders will be required to be in a lifting basket at the work face to load the round. These lifting baskets are to be integrated into the design of the emulsion trucks.

The production equipment includes a longhole drill and an emulsion loading truck. Due to the depth of the holes, less than 20 m, a top-hammer drill is recommended. This drill will provide the required accuracy and will have a much faster penetration rate than an in-the-hole drill. The emulsion-loading trucks for the production stopes do not require lifting baskets, as the holes to be loaded are not elevated.

For each operating area, the initial equipment quantities were rounded up. The operating areas for development were by level (750, 850, 950) and for production were by zone (Zone 1, 2, 3, 4, and 5).

All units will have closed, air-conditioned cabs as risk mitigation for heat rest regime and to reduce work exposure to dust-borne particles while operating. Although the refrigeration design will provide for sufficient cooling for the average temperature of the hottest month, infrequent peaks may occur when the mine is operating in the warmest zones. As such, closed cabs will prevent the equipment operators for having to use a work rest regime if temperatures exceed 27.5 °C WB.

Table 16.20 Mobile Equipment Types

Equipment Type	Quantity Base	Maximum Operating Quantity
Two-Boom Jumbo	All Lateral Ore (including Drift-and-Fill) and Waste Development	9
Rock Bolter	All Lateral Ore (including Drift-and-Fill) and Waste Development	15
Cable Bolter	Waste Development	1
Waste Development LHD	All Lateral Waste Development	7
Production LHD	All Lateral Ore Development and Ore Produced from Stopes	13
Mobile Rock Breaker	LHD only Hammer by others	2
Production Drills	Ore Produced from Stopes	6
Haulage Trucks	All Waste tonnes and CRF	4
Haulage Trucks	All Ore Tonnes	12
Explosives Loaders	Mine Wide	8
Scissor Lifts	Mine Wide	9
Telehandlers	Mine Wide	9

16.2.12.3 Fixed Equipment

A list of major fixed equipment by category for Platreef is presented in Table 16.21. This list includes fixed equipment, water handling, electrical, material handling, ventilation, and miscellaneous.

Table 16.21 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	9
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	8
Tsurumi LH430W	2

16.2.13 Personnel

Stantec's scope included the estimation of direct operating personnel required for development, production, and construction activities.

Direct labour requirements were established to suit the selected mining method, direct support systems requirements during mine development and production. Personnel requirements are based on an operating schedule of 12 hours per shift and two shifts per day for 360 days per year. A baseline was created with all crews being owner-operator and was modified based on contractor bids and compared to the baseline. The productivities did not change when compared to the baseline. The current resourcing strategy is that initial development and production will be performed by contractors. The contractor personnel will be transitioned into owner crews.

Training of the workforce is an extremely important requirement for the success of the project. The numbers provided for the development and production crews assume the initial contractor crews are ready and able. As the contractor crew size increases, they are providing training so the crews are capable. The transition of this workforce to the owner provides the same trained workforce. That training process is developed to ensure that trained operators are ready to fill the positions when required.

16.2.13.1 Classification Descriptions

The following is the breakdown of labour based on equipment operation for matching skill level. The breakdowns used are presented below in Table 16.22 to Table 16.24.

Table 16.22 Development Equipment

Development Equipment	Operators per Unit
Jumbo	Drill Rig Operator
	Drill Rig Assistant
Explosives Truck	Explosives Team Operator
	Explosives Team Assistant
	Explosives Team Assistant
Bolter	Roof Bolt Operator
	Roof Bolt Assistant
Cable Bolter	Cable Bolt Operator
	Cable Bolt Assistant
LHD	LHD Operator
Truck	Truck Driver
Telehandler (nipper delivering bolts, screen, pipe, etc.)	Transport Crew
	Transport Crew
Scissor Lift (vent, pipe installation, etc.)	Transport Crew
	Transport Crew
	Transport Crew

Table 16.23 Production Equipment

Production Equipment	Operators per Equipment
Longhole Drill	Drill Rig Operator
	Drill Rig Assistant
Explosives Truck	Explosives Team Operator
	Explosives Team Assistant
	Explosives Team Assistant
LHD	LHD Operator
LHD Jammer	LHD Operator
Truck	Truck Driver
Telehandler (nipper delivering bolts, screen, pipe, etc.)	Transport Crew
	Transport Crew
Scissor Lift (vent, pipe installation, etc.)	Transport Crew
	Transport Crew
	Transport Crew

Table 16.24 Miscellaneous Equipment

Miscellaneous Support	Operators per Equipment
Fixed Rock Breaker	Rock Breaker Operator
Mobile Rock Breaker	LHD Operator
Shotcrete Jumbo	Shotcrete Jumbo Operator

No allocation to an Owner's project team is included in the mine design portion, as it is accounted for in the overall labour plan.

No allocation for a Contractor's project team is included the mine design portion, as it is accounted for in the overall labour plan.

An overall carrying complement of additional people for sick time, absenteeism, training, etc., is accounted for in other Sections of the Platreef 2017 FS. The quantities referenced above only include direct labour requirements.

16.3 Underground Effective Times

Underground shifts were determined to be 11.5-hour shifts, 2 shifts/day, 7 days/week, 360 days per year; for work for underground development, construction, and production.

Fixed (Nonproductive) Time

Fixed nonproductive time estimates are based on prior experience with similar projects. The 11.5-hour shift is measured "collar to collar," meaning time starts when the individual enters the cage to go underground and ends when he gets off the cage on surface at the end of their shift. Fixed nonproductive time includes the following.

- Pre-shift Lineup Meetings.
- Equipment Inspection.
- Lunch, Government-Mandated Breaks, and Additional Rest Periods.
- Safety Meetings.

Table 16.25 presents the durations of fixed nonproductive activities.

Table 16.25 Fixed Nonproductive Time Hours

Description	Hours
Shift Change / Travel Time	1.50
Lunch Break	0.50
Safety Talks/Equipment Inspection/Reports	0.25
Subtotal Nonproductive Time	2.25

Available Hours

Available time is total shift time minus fixed nonproductive time. Table 16.26 summarises the available hours per shift that have been calculated for the project.

Table 16.26 Available Hours per Shift

Description	Hours
Shift Length	11.5
Subtotal Nonproductive Time	2.25
Available Work Time per Shift	9.25

Efficiencies

An average of 8.13 Effective Hours per Shift has been applied across all mining functions. Efficiency factors are applied to performance during available hours to account for unexpected underground mining delays and environmental conditions. The basis for efficiency includes, but is not limited to, the following factors.

- No Services (process water, power, compressed air, ventilation, etc.)
- Logistical Issues.

- Underground Housekeeping.
- Bathroom and Other Unscheduled Breaks.
- Delays Caused by Supervision, Engineering, Tours, and Senior Management.

The overall efficiency applied is 87.9%. Table 16.27 presents the estimated effective work time per shift.

Table 16.27 Effective Work Time

Description	Unit
Work Time per Shift	9.25 h
Efficiency Factor	88%
Effective Work Time per Shift	8.13 h

16.3.1 Productivity Rates

16.3.1.1 Waste Development

Waste development will be done through various types of ground conditions with different support requirements. The SRK recommended support requirements for waste development are as follows.

- S1A – Tunnel support for main haulages and ramps in normal conditions (shotcrete)
- S1B – Tunnel support for other haulages and ramps in normal conditions (mesh)
- S2 – Tunnel support for high stress, dynamic conditions
- S3 – Tunnel support for extreme high stress, dynamic conditions, and rehabilitation

The waste development support design was applied accordingly in the mine design based on the RMR value of the area in which the development will be positioned. The development advance rates are then based on the designed support type. The waste development rates are summarised in Table 16.28.

Table 16.28 Waste Development Rates Summary

5 × 5 Arched Development		Support Type			
		S1a	S1b	S2	S3
Effective Working Hours per Day	Min	16.25	16.25	16.25	16.25
Avg. Advance per Round (1)	m	4.511	4.51	4.51	2.9
Overall Efficiency	%	100%	100%	100%	100%
Performances					
Single Heading (single end availability)	m/day	5.16	5.83	4.26	1.93
Single Heading	m/mo	155	175	128	58
Double Heading – Increase from Single Heading – 35 % (twin end availability)	m/day	6.97	7.87	5.75	2.61
Double Heading – Increase from Single Heading – 35 %	m/mo	209	236	173	78
Multiple Heading – Increase from Single Heading – 70 % (multi end availability)	m/day	8.78	9.91	7.24	3.28
Multiple Heading – Increase from Single Heading – 70 %	m/mo	263	297	217	99

(1) 150 mm Bootlegs (Sockets)

16.3.1.2 Ore Development

The first principles rate calculation for ore development applies to 6 m wide × 5 m high arched ore access drifts. Ore development comprises loading drift development, drill drift development, and bottom drift slashing. The support requirement for all ore access development is S2 tunnel support for high stress, dynamic conditions (SRK). The ore development rates are summarised in Table 16.29.

Table 16.29 Ore Development Performance Data

6 × 5 Arched Development	Units	Quantity
Effective working hours per day	Min	16.25
Average advance per round (1)	m	4.51
Overall efficiency	%	1
Performances		
Single Heading (single end availability)	m/day	4.72
Single heading	m/mo	143
Double Heading – Increase from Single Heading – 35 % (twin end availability)	m/day	6.37
Double Heading – Increase from Single Heading – 35 %	m/mo	194
Multiple Heading – Increase from Single Heading – 70 % (multi end availability)	m/day	8.02
Multiple Heading – Increase from Single Heading – 70 %	m/mo	241

(1) 150 mm Bootlegs (Sockets)

16.3.1.3 Longhole Stoping

Longhole Stoping is the predominant source of ore production for the operation. This method is used in areas where the ore thickness exceeds 15 m. The Transverse Longhole Stoping sequence is as follows.

- Drilling will be done by the production drill in one pass, starting with the drop raise and slot sequence followed by stope blast hole drilling.
- Loading and blasting will follow a set sequence that is designed to systematically increase the open area of the stope, named the slot sequence.
- Mucking will be done after every blast to load the stope area empty to create the necessary open area for the following blasts.
- The bottom drift will be slashed wider to 15 m by the ore development crew and mucked, before the slot sequence is started and will be blasted in-time to ensure that each following bench from the longhole stoping can break into the available area below.
- Cemented rock fill or paste fill will be used as post extraction support.

Three longhole stoping production rates were calculated that will be applied in the mine design to the appropriate stope lengths.

- 15 m Long Stopes (applied to all stopes between 10 m and 20 m long)
- 30 m Long Stopes (applied to all stopes between 21 m and 35 m long)
- 45 m Long Stopes (applied to all stopes between 36 m and 60 m long)

The Longhole Stopes have changed from the Platreef 2015 PFS to improve grades, efficiency, and safety, and to remove development from the critical path. The first change is the change in height to 20 m high stopes. This change provided for higher-grade stopes and contributed to the overall grade increase in the LOM plan. The next step was a reduction in the widths of the top and bottom cuts from 15 m to 6 m. This change improved development performance, as the volume of development in a top cut is reduced by 60%. A further advantage is that the adjacent secondary top cuts can advance prior to the primary top cut being filled. This provides much more flexibility in the development schedule and improves the stope availability. However, secondary top cuts must be examined to determine if deterioration is occurring from the blasting of adjacent stopes. The final change to maintain recoveries is slashing the bottom cuts with the development drill rig. This provides for a smooth mucking floor to maintain the recoveries identified in the Platreef 2015 PFS. This also has the advantage that the backfill barricade in the bottom cuts must only span 6 m instead of 15 m, which reduces the construction size and thickness of the barricades.

The LHS rate calculation summary is illustrated in Table 16.30.

Table 16.30 Longhole Stoping Rates

Stope Length	Cycle Components (Days)								Total Tonnes (avg.)	Performance
	Drill	Load	Muck	Fence Constr.	Paste Plug	Fill Stope	Set	Total Days		
10–20 m	5.9	4.4	9.5	3.0	3.6	3.6	24.2	54.1	16,250	300 t/d
21–35 m	7.8	5.3	15.9	3.0	4.0	6.0	22.3	64.4	26,850	417 t/d
36–60 m	10.7	6.8	25.3	3.0	5.3	8.6	19.8	79.6	46,400	538 t/d

All rates in Table 16.30 encompass the paste fill rate listed in Table 16.31. The difference in the performance is due to the concurrent drilling activity occurring in an adjacent stope. Larger stopes require a longer drilling time therefore that delay associated to filling of the stope is reduced giving the larger stopes an overall higher performance.

Table 16.31 represents the key data used to determine the paste delay factor for each stope length in Table 16.30. Paste fill rates are not affected by the mine operating hours. The paste fill lines will operate continuously 24 hours a day with one scheduled day per month for preventative maintenance. This provides an overall utilisation of 96.4% for the system. Using the pour performance from Golder, 291 paste t/h (85% pump efficiency), the stopes will receive 6, 730 paste t/d. The overall utilisation accounts for downtime due to changing pour locations, stopes not being ready to be poured, line issues, and water quantities in stopes preventing pouring. This reduction results in the stopes being filled from the plug to the top in 3.6, 6.0, or 8.6 days based on stope lengths of shown in Table 16.30.

Table 16.31 Paste Fill Parameters

Stope Length	Backfill Barricade Construction Delay	Pour Hours/Day	Pour Performance (paste t/h)
10 m to 60 m	3 Days	23.125	291

Several mining zones are established based on geological structures, dip of the ore body, and overall value, while considering the location of main infrastructure at the 750, 850, and 950 Levels. The main zones were then divided into subzones.

The number of stopes and levels per subzone were counted to calculate the number of stopes per sublevel. Based on the number of stopes per sublevel, the amount of active stopes was determined based on the criteria illustrated in Table 16.32.

Table 16.32 Active Stope Criteria per Subzone

Average Stopes per Level	Max Active
0–7	x/4
8–15	4
16–23	6
>24	8

A production factor was assigned based on the number of sub-levels that are within a subzone based on the criteria illustrated in Table 16.33 to account for the improved efficiency of mining over multiple sublevels.

Table 16.33 Level Production Factor Criteria

Number of Levels	Production Factor
1–4	1
5–8	1.25
>9	1.5

An average production rate per stope of 417 t/d was used to calculate the maximum production per zone. As a result of these factors, the maximum production rate for each zone in the Platreef Project was determined, as illustrated in Table 16.34.

Table 16.34 Production Rate Estimation per Mining Subzone

Subzone	Number of Levels	Total Stopes	Average Stopes per Level	Active Stopes per Level	Total Active Stopes per Zone	Number of Active Levels	Production Factor	Max (t/d)
12	12	444	37	8	12	3	1.5	5,004
14	17	465	27	8	12	3	1.5	5,004
15	10	117	12	4	6	3	1.5	2,502
21	12	320	27	8	12	3	1.5	5,004
22	6	131	22	6	8	2	1.25	3,128
23	4	45	11	4	4	1	1	1,668
24	4	43	11	4	4	1	1	1,668
32	6	131	22	6	8	2	1.25	3,128
33	11	257	23	8	12	3	1.5	5,004
35	3	46	15	8	8	1	1	3,336
41	5	109	22	6	8	2	1.25	3,128
43	4	155	39	8	8	1	1	3,336
45	6	136	23	6	8	2	1.25	3,128
51	10	186	19	6	9	3	1.5	3,753
52	12	295	25	8	12	3	1.5	5,004
53	13	338	26	8	12	3	1.5	5,004
70	6	157	26	8	10	2	1.25	4,170
80	13	145	11	4	6	3	1.5	2,502

Platreef is a large mine with a high mining rate. Maintaining the high mining rate requires having several stopes in cycle at once. The stope cycle includes longhole drilling, loading, blasting, mucking, and backfilling; it does not include the development of the top cuts or bottom cuts. Figure 16.22 shows the average daily number of Longhole Stopes and Drift-and-Fill accesses in production by year.

Figure 16.22 Average Daily Stopes

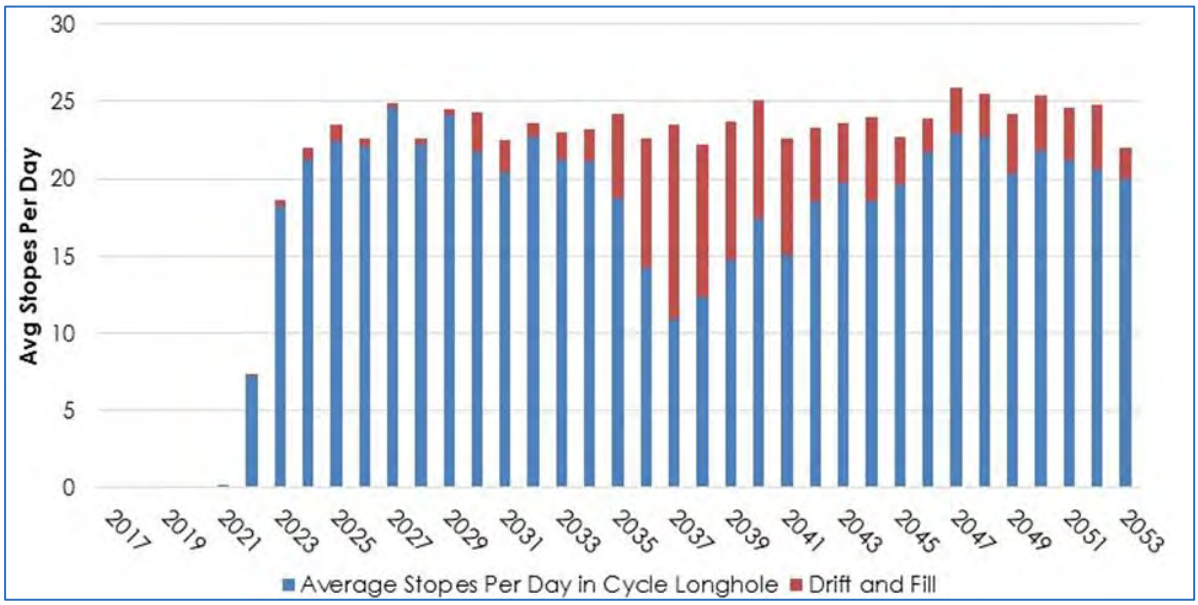


Figure by Stantec, 2017.

16.3.1.4 Drift-and-Fill Mining

Drift-and-Fill mining is a flexible mining method that allows near complete recovery of the ore zones. This method is development intense, resulting in lower productivity rate. This is due to the small blast size and small tonnages per cycle. The other impact is a higher mining cost per tonne, but the method allows for less dilution and more selective, higher grade mining. Good control of drilling and blasting is also necessary to minimise dilution from backfill. Mining is completed with the same equipment used for mine development.

Drift-and-Fill mining is a variation of the Cut-and-Fill mining method. In Drift-and-Fill mining, the ore zone is divided into horizontal slices, or lifts, and ore drifts are mined and backfilled adjacent to one another in a repeating fashion. Upon completion of each drift, a bulkhead is constructed, and the void is backfilled with cemented paste fill. After the backfill sets sufficiently to achieve the required strength, another drift is driven next to the fill.

For the Platreef ore body, Stantec chose two methods of Drift-and-Fill mining: transverse and chevron. The size and shape of the ore body will dictate the method used. For larger wide shapes (pancake-style shapes), the chevron method will be incorporated. For narrower stopes, the transverse method will be the applied to determine the production rates.

Drift-and-Fill: Transverse Stoping

Transverse mining is perpendicular to the strike of the ore body in situations where the average stope is 15 m perpendicular to strike (refer to Figure 16.23). Stopes are extracted as primary, secondary, or tertiary. 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 until adjacent primary and secondary stope paste fill set times are complete. Except for 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 were used to establish the production rate for this mining method.

- Each stope is 5 m × 5 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 1,177.5 t.
- Using a single heading production rate of 354 t/d (4.5 m/day advance rate), one stope will be mined in approximately 3.5 days.
- The cure time for the paste fill is 28 days; before advancing to the adjacent secondary stope then the total active primaries faces required for continuous mining will be 8 faces (28 days/3.5 days/stope + the initial panel ~9 panels).
- Each panel (defined as the combination of primary, secondary, and tertiary) is 15 m wide (5 m/stope × 3 stopes), and 9 available panels translate to 135 m required for the total length of one active area, including the 5 m wide access drift (42,390 tonnes).
- The maximum allowed rate is based on the amount of accesses that are available to a series of active areas. Each access from the ramp will allow for 354 t/d.

Figure 16.23 Typical Drift-and-Fill Transverse Layout

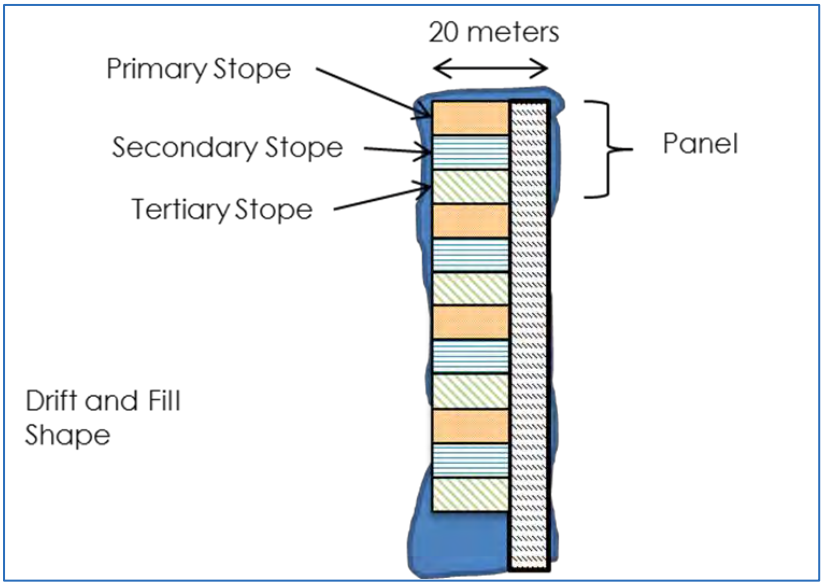


Figure by Stantec, 2017.

Drift-and-Fill: Chevron Stoping

Chevron Drift-and-Fill will use a similar primary-secondary-tertiary sequence until the entire slice of ore is depleted (Figure 16.24). The average length of the stope in this sequence for Platreef is 150 m, with each panel having an average length of 46 m; therefore, there are sixty 5 m × 5 m stopes for each 70 m wide mining block (164,850 tonnes per mining block). The total tonnage of the pancake will determine the number of mining blocks.

Figure 16.24 Typical Drift-and-Fill Level Layout

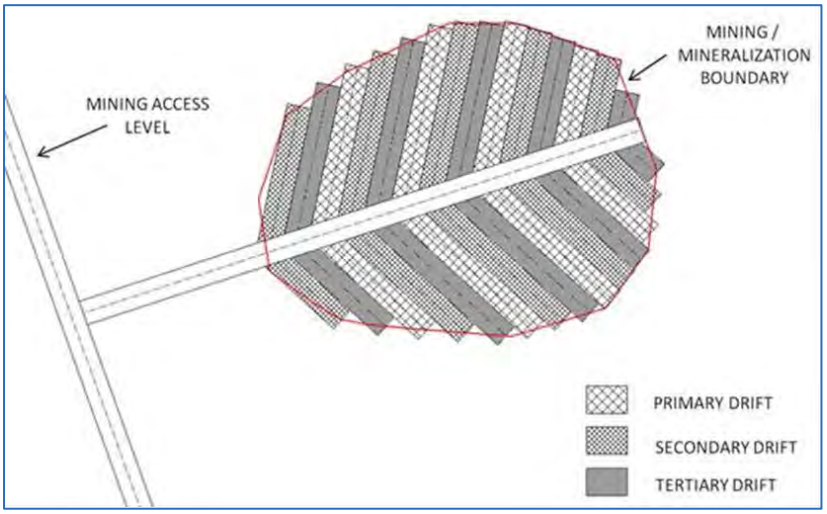


Figure by Stantec, 2017.

In these areas, the ore zones will be divided into 5 m high horizontal slices, or lifts. Within each of these lifts, individual drift cuts 5 m wide × 5 m high are driven across the ore zone.

Due to the single-access nature of this mining sequence, the ability to have an LHD and Jumbo working independent of each other is greatly reduced. Congestion in the headings as a result of a single access reduces the production rate to a double heading rate. The maximum ventilation supplied to a level is 142 m^3/s, limiting the production rate to a maximum of three centre drift headings at a given time period per level. Each centre drift will supply 47.2 m^3/s to a mining block as defined above. Based on the dual heading rate, the estimated performance of the chevron Drift and Fill method is 488 t/d per centre drift, or a maximum of 1,464 t/d per level.

An opportunity to increase the overall performance exists in the paste fill portion of the stoping. The tertiary stope drifts can be backfilled with waste, thus reducing the time to construct a fence and cure. For the Platreef 2017 FS, it is assumed that all drifts will be paste filled with waste rock placed in secondaries and tertiaries prior to filling. This results in approximately 25% of the entire drift and fill stopes being filled with waste rock.

16.3.1.5 Mass Excavations

First principles rate calculations for lateral development drifting for 5 m wide × 5 m high arched drifts in S1A ground were the basis for determining an aggregate development rate for mass excavations. To determine the performance rate, the number of days required to develop these excavations was determined. There are two types of mass excavations.

- Unique Excavations
- General Mass Excavations

Unique excavations require slashing or additional cuts to complete the development. The list of unique excavations for the Platreef Project is listed in Table 16.35. Full-face top and bottom cuts were used in high profile or rectangular designs such the crusher rooms and crane bays. The performance of irregular shapes such as the grizzly stations and tool cribs consists of full-face advance, with slashing to the extents of design. The aggregate amount of days with the aforementioned development methodologies applied against the centreline distance results in the performance rates listed in Table 16.35.

Table 16.35 Unique Mass Excavation Rates

Excavation Name	W (m)	H (m)	No. of Passes	Rate	Unit	Delay for Cable Bolts	Unit
Shop Large Profile	7.0	5.0	1	3.87	m/day	3.00	days
Shop Tool Crib	2.0	5.0	1	1.80	m/day	0.25	days
Grizzly Acc Standard	6.4	5.0	1	4.06	m/day	0.50	days
Grizzly Acc Wide	9.0	5.0	1	3.39	m/day	0.50	days
Ore Pass Temp	5.0	7.3	2	2.29 (shaft crew)	m/day	0.50	days
Transverse Stope Drift	6.0	5.0	1	4.72	m/day	–	–
Crane Bay Large	7.6	8.8	4	1.43	m/day	0.05	days/m
Crane Bay Medium	7.6	7.0	4	1.60	m/day	0.05	days/m

Schedule durations assigned to individual heading components were derived by dividing length or volume by the unit advance rate.

16.3.2 Mine Development Plan and Scheduling

Mine development has been broken down into four main phases. Phase 1 involves lateral development off Shaft 1 during shaft sinking. Phases 2 and 3 involve lateral development after Shaft 1 sinking is complete and prior to Shaft 2 commissioning. In Phase 2, lateral development is limited until Ventilation Raise 1 is completed; therefore, the priority is to commission Ventilation Raise 1 so that ventilation can be increased and additional development crews can be added. In Phase 3, the development rate off Shaft 1 is increased after Ventilation Raise 1 is commissioned. Phase 3 is no longer limited by ventilation but is limited by the 2,500 t/d hoisting capacity of Shaft 1. Phase 4 occurs in November 2021 when Shaft 2 is commissioned, and hoisting is no longer a bottleneck.

16.3.2.1 Preproduction Development Schedule

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

- Minimal equipment availability since there will be limited development off Shaft 1.
- Limited ventilation in the overall mine design. Advance rates increase when Ventilation Raise 1 comes into operation as more mining crews can be added.

This phase 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 to ensure that mainly critical development was completed due to the limited hoisting capacity.

Figure 16.25 shows the monthly rock production during the preproduction period and Figure 16.26 shows the annual rock production for the life of mine.

Figure 16.25 Monthly Rock Production

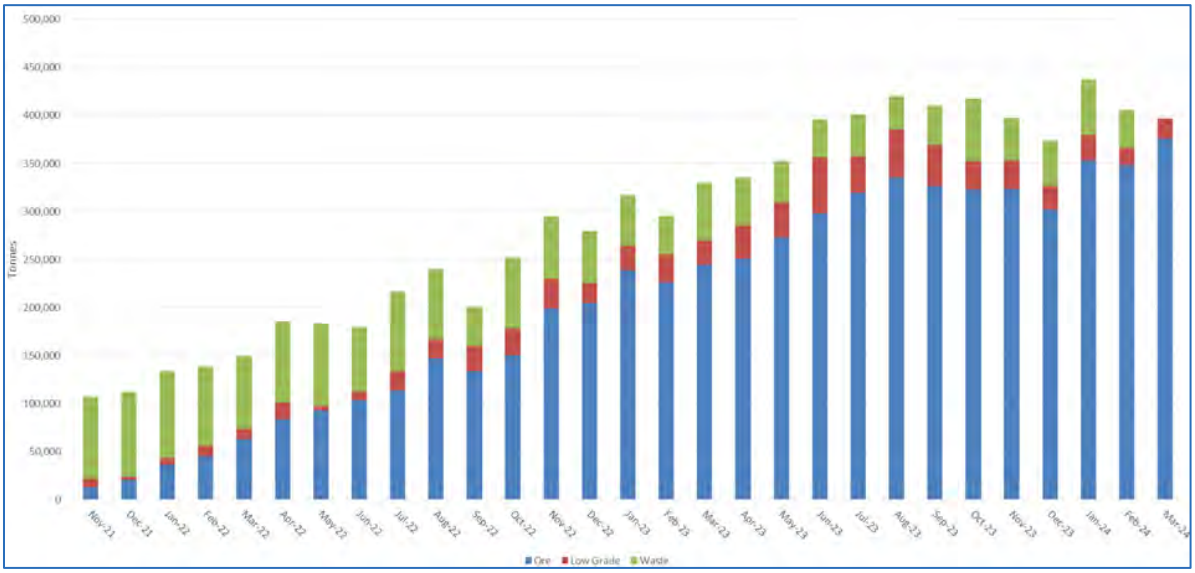


Figure by Stantec, 2017.

Figure 16.26 Yearly Waste Rock Production

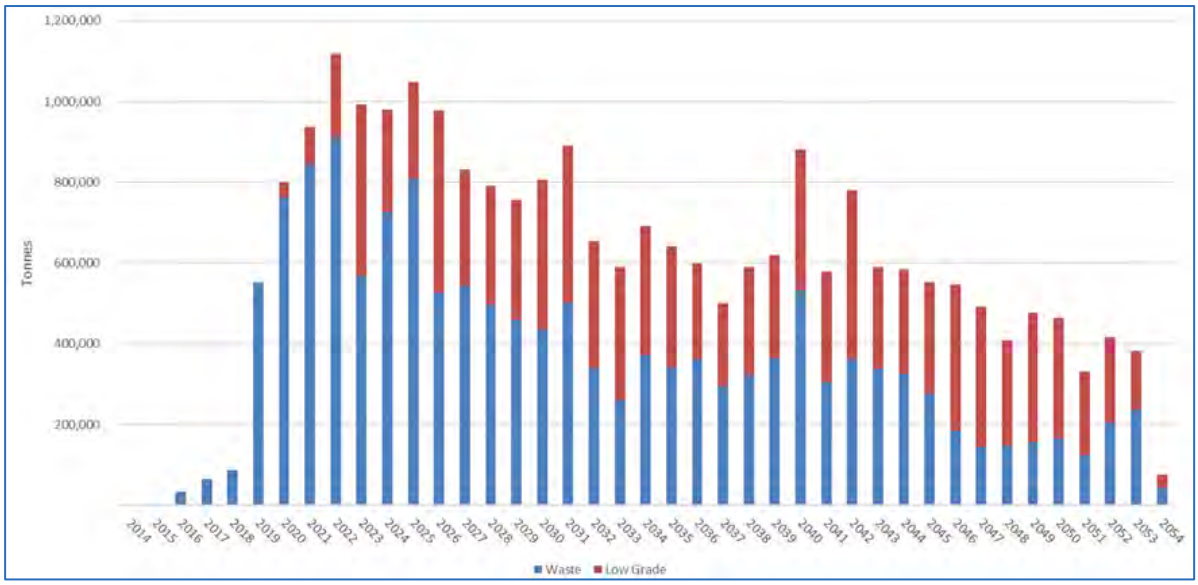


Figure by Stantec, 2017.

16.3.2.2 Life of Mine Development Schedule

Once the preproduction development was defined, the overall LOM development was scheduled, (see Table 16.36 and Table 16.37). 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 180,000 m of waste development, or 14.5 Mt of waste material, is mined. The overall waste development is summarised in Table 16.38.

Table 16.36 Life-of-Mine Development Schedule to 2034

Description	Units	Totals	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Shafts																					
Advance	m	2,125	267	442	343	686	387	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste	kt	408	35	60	52	165	96	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Vertical Development																					
Advance	m	9,076	-	-	-	1,140	799	421	759	229	678	1,093	280	20	203	88	280	286	146	35	391
Waste	kt	395	-	-	-	61	70	31	17	5	49	75	6	0	5	2	6	6	3	0	8
Lateral Waste Development																					
Advance	m	168,673	-	40	228	3,452	7,311	10,444	11,118	6,977	8,580	9,289	6,612	6,533	6,212	5,696	5,342	6,164	4,021	3,131	4,537
Waste	kt	13,690	-	5	39	326	603	812	894	563	677	733	519	542	492	457	430	495	339	261	363
Low Grade Development																					
Low Grade Waste Advance	m	114,890	-	-	-	-	499	1,219	2,507	5,066	3,094	2,904	5,609	3,477	3,602	3,589	4,539	4,773	3,775	3,933	3,897
Low Grade Waste	kt	9,563	-	-	-	-	38	94	207	425	253	240	452	289	295	298	371	390	313	329	320
Total Waste and Low Grade Development																					
Total Advance	m	294,764	267	482	571	5,278	8,995	12,084	14,384	12,272	12,353	13,286	12,502	10,031	10,017	9,373	10,161	11,223	7,943	7,099	8,825
Total Waste	kt	24,054	35	65	90	552	808	937	1,118	993	980	1,048	977	832	792	757	807	891	654	591	691
Ore Development																					
Advance	m	131,861					197	349	3,248	7,758	6,432	5,116	6,684	4,378	4,473	5,015	4,690	4,167	5,230	4,923	4,545
Ore	kt	11,107	-	-	-	-	15	44	275	653	538	429	564	368	374	421	390	346	439	418	383
Total Development Advance	m	426,624	267	482	571	5,278	9,192	12,433	17,632	20,030	18,785	18,402	19,186	14,409	14,490	14,387	14,851	15,390	13,173	12,021	13,371

Table 16.37 Life-of-Mine Development Schedule to 2053

Description	Units		2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Shafts																					
Advance	m		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste	kt		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Vertical Development																					
Advance	m		52	13	-	388	179	622	193	253	73	53	193	50	-	-	103	53	-	-	-
Waste	kt		1	0	-	8	4	16	4	6	2	1	4	1	-	-	2	1	-	-	-
Lateral Waste Development																					
Advance	m		4,254	4,613	3,802	3,962	4,489	6,323	3,658	4,364	4,117	3,944	3,286	2,189	1,715	1,760	1,876	1,940	1,475	2,465	2,753
Waste	kt		341	362	295	314	360	516	301	356	338	324	272	184	145	148	155	164	123	206	236
Low Grade Development																					
Low Grade Waste Advance	m		3,638	2,905	2,516	3,232	3,094	4,200	3,261	5,067	2,941	3,164	3,341	4,245	4,000	3,030	3,750	3,503	2,404	2,452	1,665
Low Grade Waste	kt		299	236	206	267	255	350	275	418	250	259	277	362	347	261	320	300	207	211	146
Total Waste and Low Grade Development																					
Total Advance	m		7,945	7,531	6,318	7,582	7,763	11,145	7,112	9,684	7,131	7,160	6,820	6,485	5,716	4,790	5,729	5,496	3,879	4,917	4,419
Total Waste	kt		641	598	500	590	619	882	579	780	589	584	553	547	492	409	477	465	331	417	382
Ore Development																					
Advance	m		3,815	3,370	2,595	2,959	2,754	3,002	2,480	4,249	4,603	3,966	3,851	4,630	4,741	2,987	3,005	3,603	2,250	3,126	2,671
Ore	kt		321	281	218	246	229	251	207	356	388	334	324	393	404	255	257	305	191	265	226
Total Development Advance	m		11,760	10,901	8,913	10,540	10,516	14,148	9,592	13,933	11,734	11,126	10,672	11,114	10,456	7,777	8,734	9,098	6,129	8,042	7,089

Table 16.38 Life-Of-Mine Waste Development Summary

Description	Units	Total
Shafts		
Advance	km	2
Waste	kt	408
Lateral Development		
Advance	km	169
Waste	kt	13,690
Vertical Development		
Advance	km	9
Waste	kt	395
Low Grade Development		
Low Grade Advance	km	115
Low Grade Tonnage	kt	9,563
Total Waste Development		
Total Advance	km	295
Total Waste	kt	24,055

Note that the approximately 9.6 Mt of low-grade material has an NSR greater than $50 NSR but less than $100 NSR and could be stockpiled during the LOM.

16.3.2.3 Low-Grade Development

Low-grade development has been separated in the schedule and is defined as any development with an NSR value of greater than $50 NSR but less than $100 NSR. This material will be identified and stockpiled separately on surface. For the Platreef 2017 FS, low grade development is not considered reserve and has no economic value. However, during the life of the mine, metal prices may result in NSR values providing an opportunity to process this material.

For this to occur, it is necessary to separate the material after development. With the large number of active headings underground, separate low-grade storages will be available during the development cycle. Once the material is stored underground, batch hoisting of the low-grade stockpile will be required.

With the excess capacity in the ore handling system and the availability of the waste pass as a backup, this separation is possible. With a maximum production of less than 1,000 t/d of low-grade development, only 1 hour per day of hoisting is required. See Figure 16.27 for the annual low-grade development tonnages. Batch hoisting of the low-grade material is therefore possible once or twice a week. This batching could occur and still allow the required time for ore hoisting, since Shaft 2 has a capacity of over 6 Mt/a.

While low-grade material is being batched through one line of the ore handling system, ore movement can continue in the other line through the pass, crusher, and into the fine ore bin. Then, once the low-grade hoisting is completed, the system should have a significant amount of ore in the other pass and bin to resume ore hoisting.

Figure 16.27 Annual Low-Grade Production with NSR

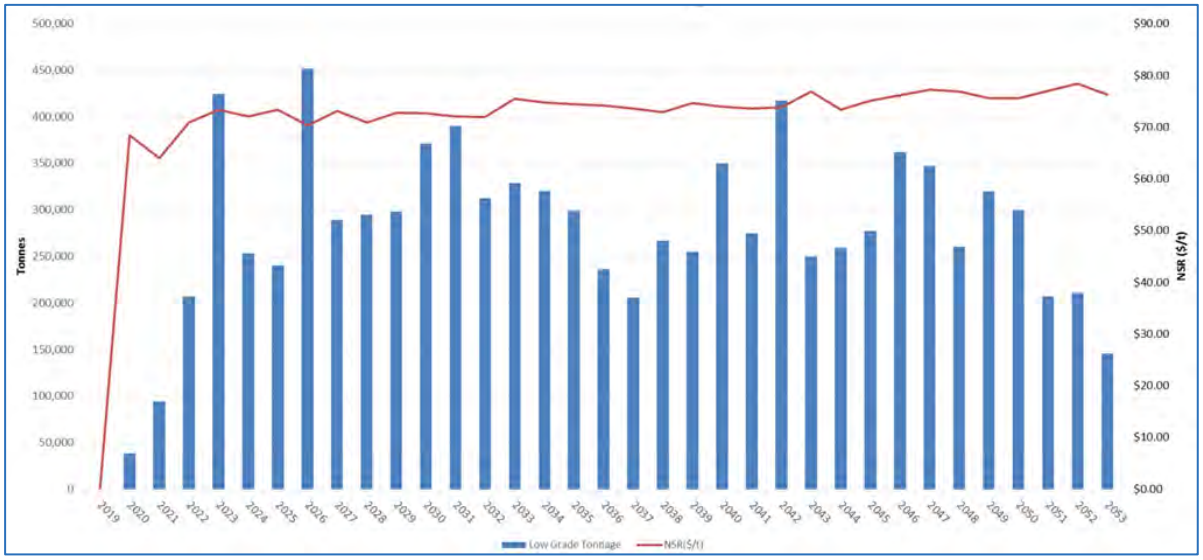


Figure by Stantec, 2017.

16.3.3 Production Planning Criteria

Although the resource has approximately 330 Mt above an $80 NSR cut-off, the reserve focused on maximising the grade profile for a 4 Mt/a production rate for 30 years. As such, higher NSR cut-offs were used to develop the reserve statement. This strategy provides opportunity for either a longer mine life or ramping up to higher production rates to utilise more of the resource.

The LOM production plan focuses on maximising higher grade areas. The ore body was targeted to get approximately 130 Mt at the highest NSR. This resulted in using declining cut-off grades, decreasing the NSR from $155 to $100.

A further focus on optimising NPV targeted the higher grade stopes in the LOM plan for mining in the early years. An optimisation was performed based on stope locations, stope grades, mining method, and subzone productivities. Through this analysis, the grades in the first 10 full years had a significantly higher 3PE+Au than the Platreef 2015 PFS. Table 16.39 shows the first 10 full years and the remaining life of mine.

Table 16.39 3PE+Au in Early Production

Item	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032–53
Ore (t)	1,369	3,457	3,866	3,987	3,971	4,104	3,785	4,112	4,112	3,775	88,058
NSR ($/t)	188.71	184.56	180.31	170.30	165.70	165.76	163.59	171.76	170.15	163.26	152.08
3PE+Au (g/t)	5.63	5.35	5.08	4.76	4.64	4.67	4.58	4.92	4.87	4.58	4.21

Four subzones (14, 22, 24, and 41) were identified and prioritized for initial / early development. Once Shaft 2 is commissioned, one zone per quarter will be accessed for production ramp up. Stantec's mineable resource estimates include detail by mining methods for each of the mining zones and subzones. The following additional criteria were applied over the mine life:

- Proximity to shafts and early development.
- High grade.
- Highly productive.
- Shaft pillars are not extracted until the end of the mine life.
- Low operating cost.

16.3.3.1 Production Ramp-Up Schedule

Production ramp-up begins in November 2021 and ramps up approximately 1,500 t/d each quarter. After seven quarters, or 21 months, full production is achieved. This ramp up is achieved through four subzones coming into production during the ramp up period.

Prior to production beginning in late 2021, the development ramp-up strategy for the production ramp-up is achieved by accessing three of the four subzones, with the fourth subzone accessed soon after. These subzones continue to ramp up their production rate until the 4.0 Mt/a production rate is achieved.

The ramp-up begins with mining in one of the subzones. Even though three subzones are ready, only one subzone is started, and there is a 1–2 quarter delay before starting the next subzone. This methodology provides flexibility and contingency in the schedule. Also over the ramp-up period, 20% of the ore tonnes comes from development, which is a common method in the area and will have been occurring on the project for over 2 years.

The early ramp-up is achievable based on the number of primary stopes and development headings. Compliance to the development schedule is required to ensure that additional ore development is not taken and impacts the waste development schedule. This is common practice in operating mines and will be important during the project ramp-up. This is important at Platreef, as there are a significant number of primary and secondary top and bottom cuts available once Shaft 2 is commissioned.

As for the Longhole Stoping sequence, there will be a significant number of stopes in the three subzones available for early production. The key to maintain the scheduled ramp-up will be prioritising CRF to ensure that secondary stopes can be brought online. Flexibility is possible due to the number of available subzones and primary stopes available within each subzone, but proper focus on backfilling is critical in a primary-secondary longhole stope sequence.

16.3.3.2 Life of Mine Production Schedule

The first 10 years of full production were analysed to maximize the ore grade profile. During this period, constraints such as stope availability, ventilation requirements, and hoisting capacity were taken into consideration. Figure 16.28 and Figure 16.29 show the NSR and 3PE+Au grade ranges of the stopes. These values were taken into consideration when developing the LOM production schedule.

Figure 16.28 NSR Values by Stope

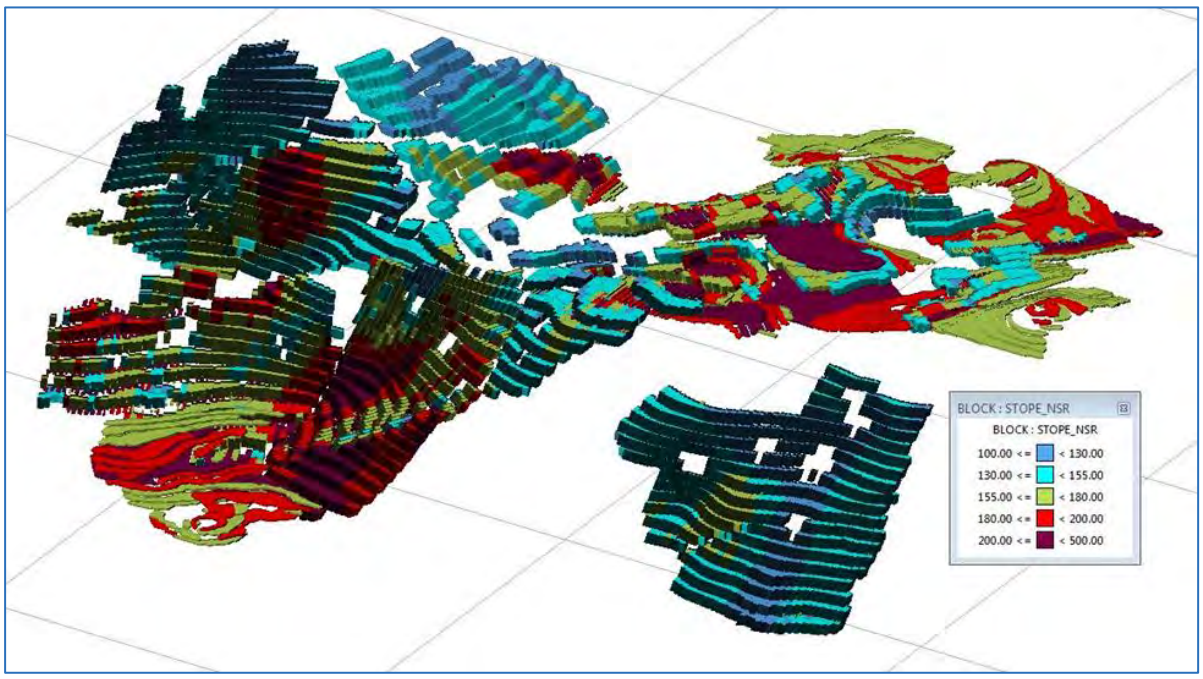


Figure by Stantec, 2017.

Figure 16.29 3PE+Au Value by Mining Shapes

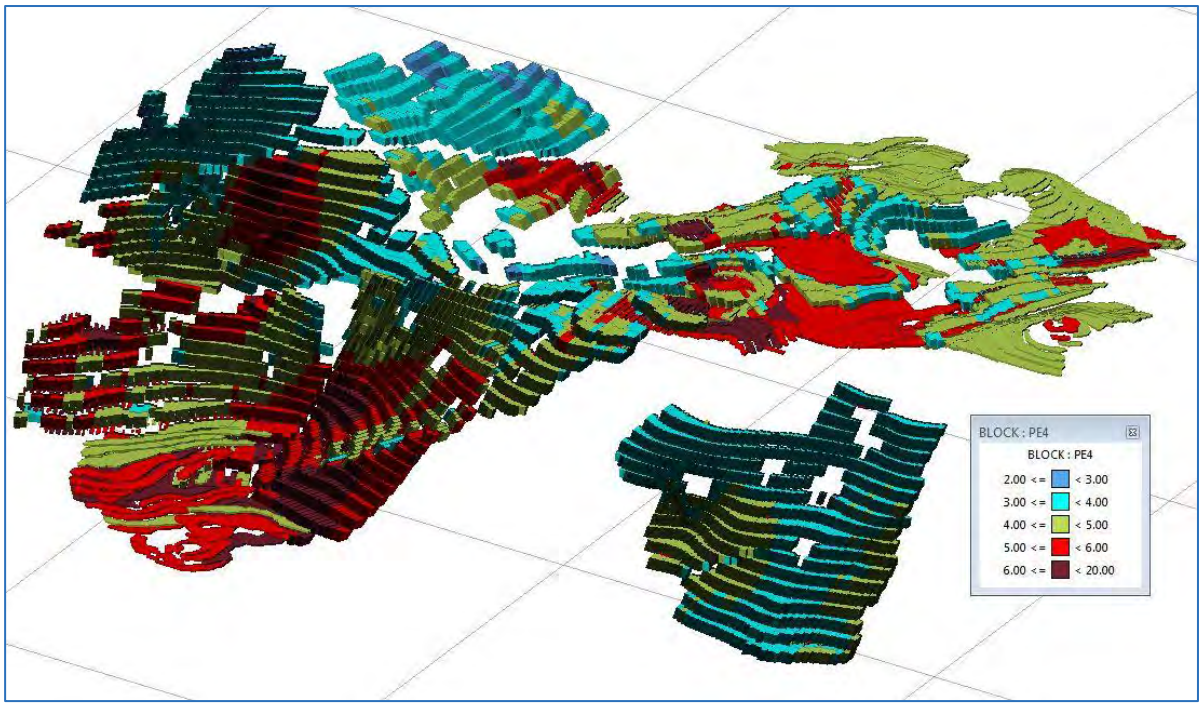


Figure by Stantec, 2017.

The scheduling process used Studio5 D Planner along with inputs from the Primavera P6 schedule developed for the infrastructure. Monthly detail of the schedule for both production and development was completed and analysed for over 50 periods up to September 2023. After this point, the schedule was detailed quarterly for 13 quarters until the end of Q4'26. Then the remaining 27 years of the schedule were analysed on an annual basis.

This analysis involved levelling of resources, ensuring constraints were followed, and looking for opportunities for improvement. The overall result provided for a high-level of detail in the early and critical ramp-up period.

Production Summary

Production is summarised in the following table and six figures. Table 16.40 shows the overall LOM production summary. Figure 16.30 and Figure 16.31 show the annual production rate along with the associated NSR based on BDT20 and 3PE+Au, respectively.

Figure 16.32 and Figure 16.33 show the annual production rate by mining method along with the associated NSR based on BDT20 and 3PE+Au, respectively. Figure 16.34 and Figure 16.35 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 life of mine), which is not included in the block model. Therefore, this material is referenced as "Paste and Other Tonnes" in Figure 16.34 and Figure 16.35.

Table 16.40 Life-Of-Mine-Production Summary

Description	Units	Total
Drift-and-Fill	kt	20,441
Longhole Stopes	kt	93,133
Ore Development	kt	11,107
Total Ore	kt	124,681
Diluted Grades	–	–
NSR	$/t	$157.66
Cu	%	0.166
Ni	%	0.339
Pt	g/t	1.954
Pd	g/t	2.007
Au	g/t	0.302
Rh	g/t	0.135
S	%	0.825
3PE+Au	g/t	4.398
Max Daily Production Rate	t/d	12,116
Low-Grade Stockpile	kt	9,563
Low Grade Stockpile NSR	$/t	$73.95
Waste (excl Low Grade)	kt	14,479

Figure 16.30 Annual Production with NSR

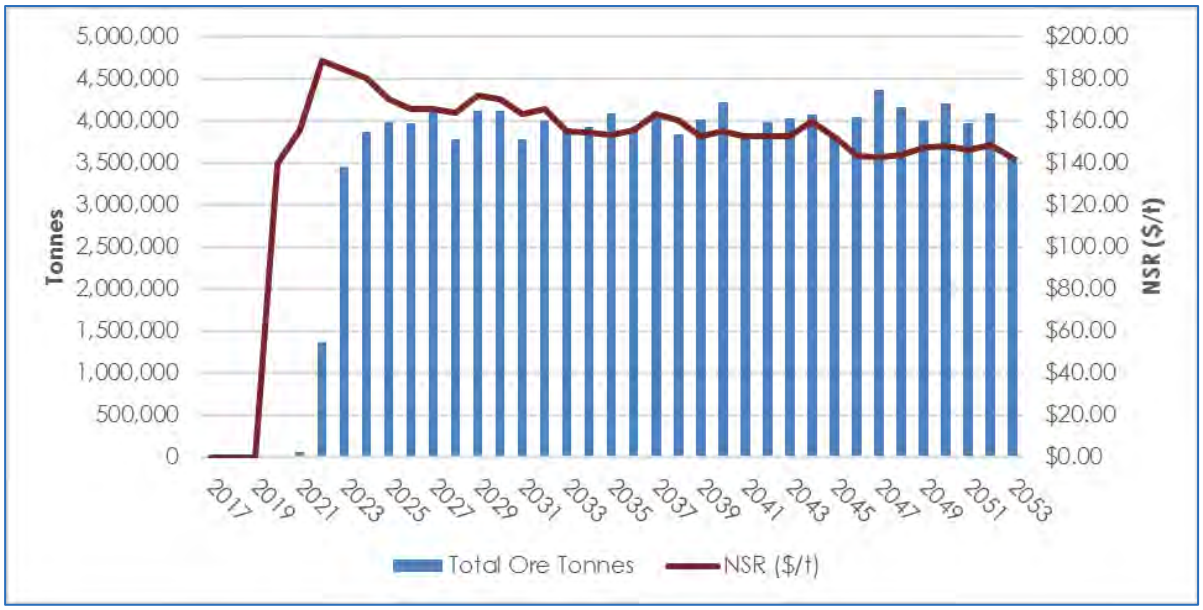


Figure by Stantec, 2017.

Figure 16.31 Annual Production with 3PE+Au

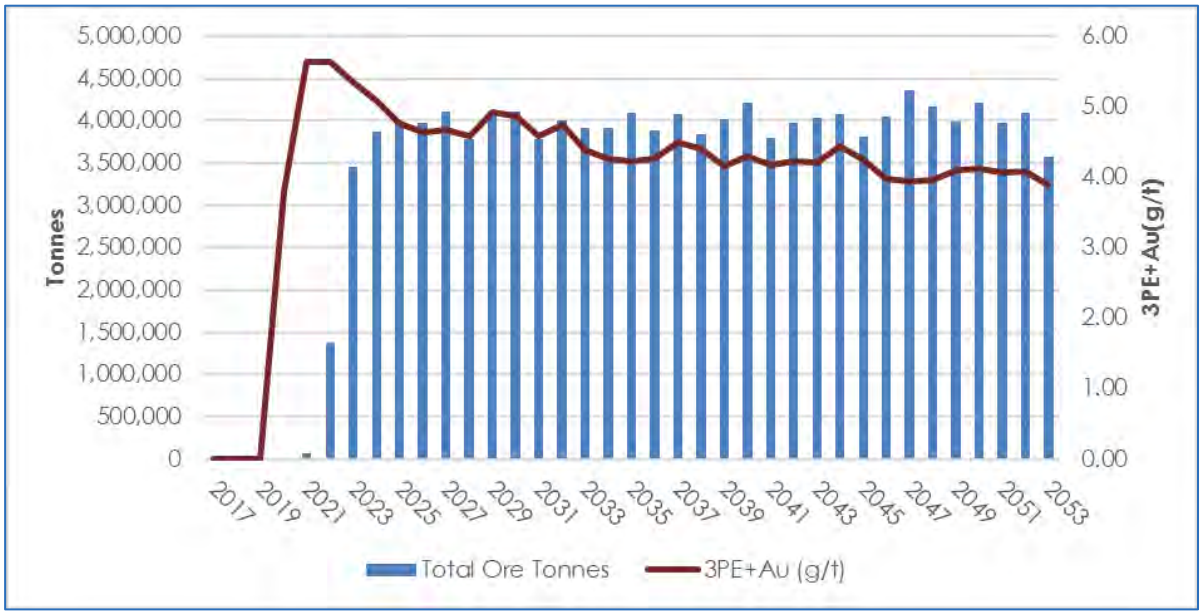


Figure by Stantec, 2017.

Figure 16.32 Annual Production by Mining Method with NSR

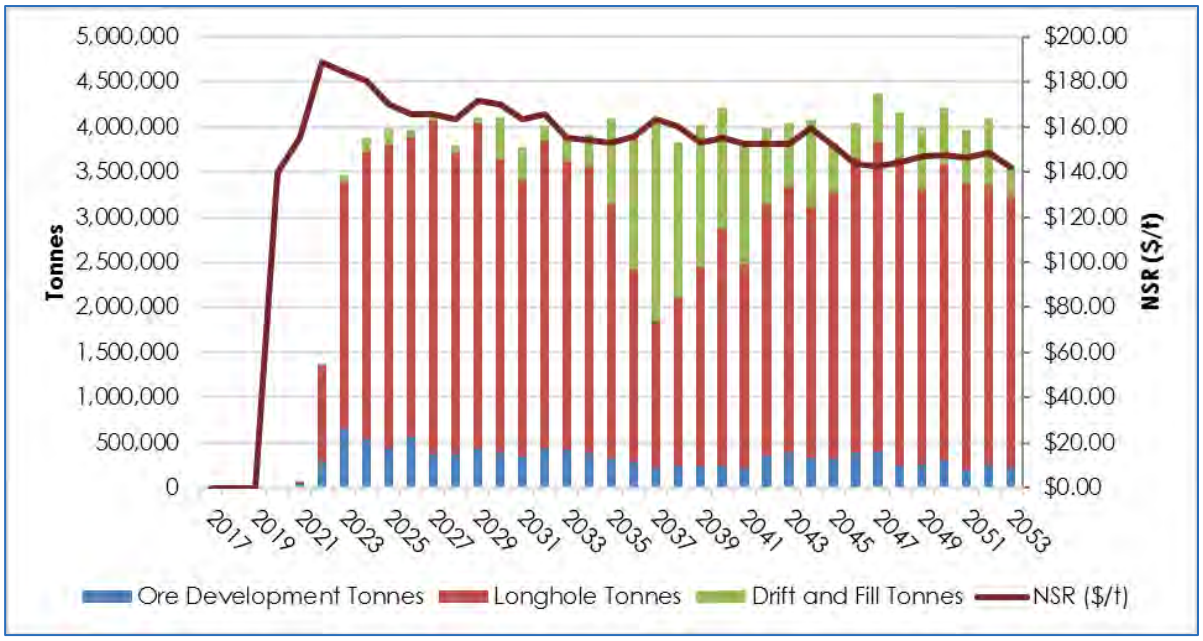


Figure by Stantec, 2017.

Figure 16.33 Annual Production by Mining Method with 3PE+Au

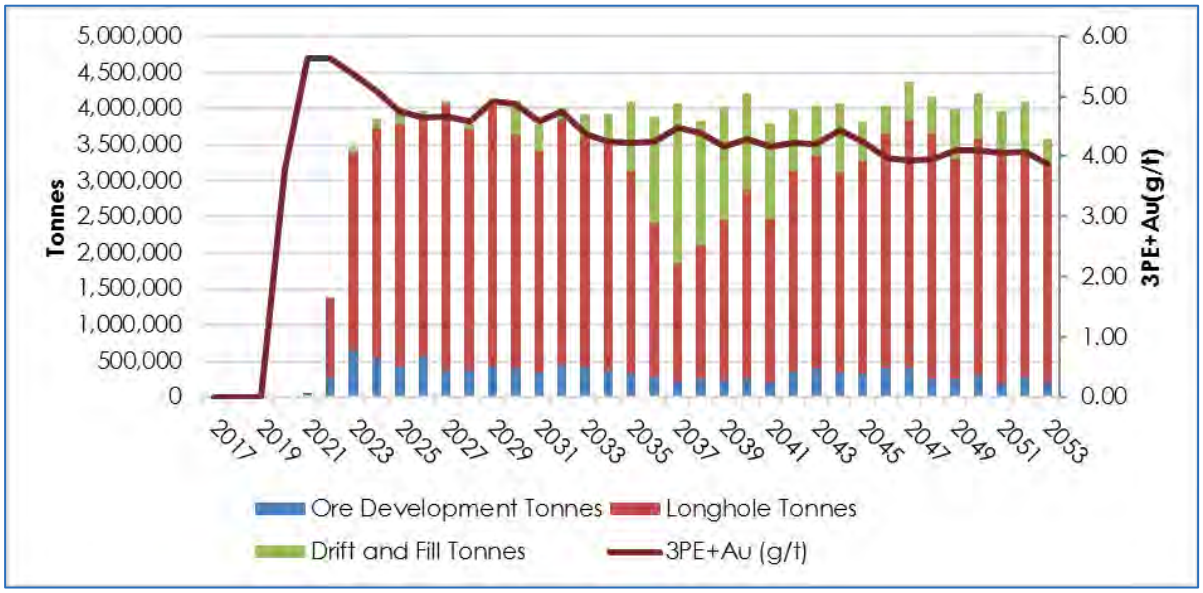


Figure by Stantec, 2017.

Figure 16.34 Annual Production by Rock Type with NSR

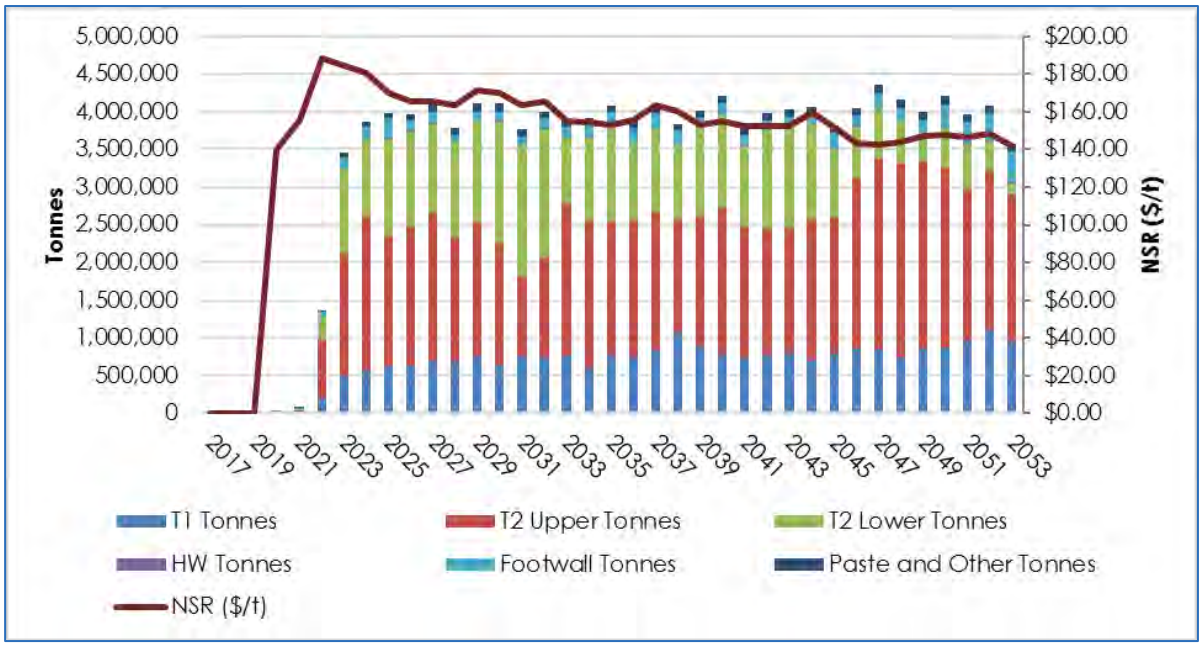


Figure by Stantec, 2017.

Figure 16.35 Annual Production by Rock Type with 3PE+Au

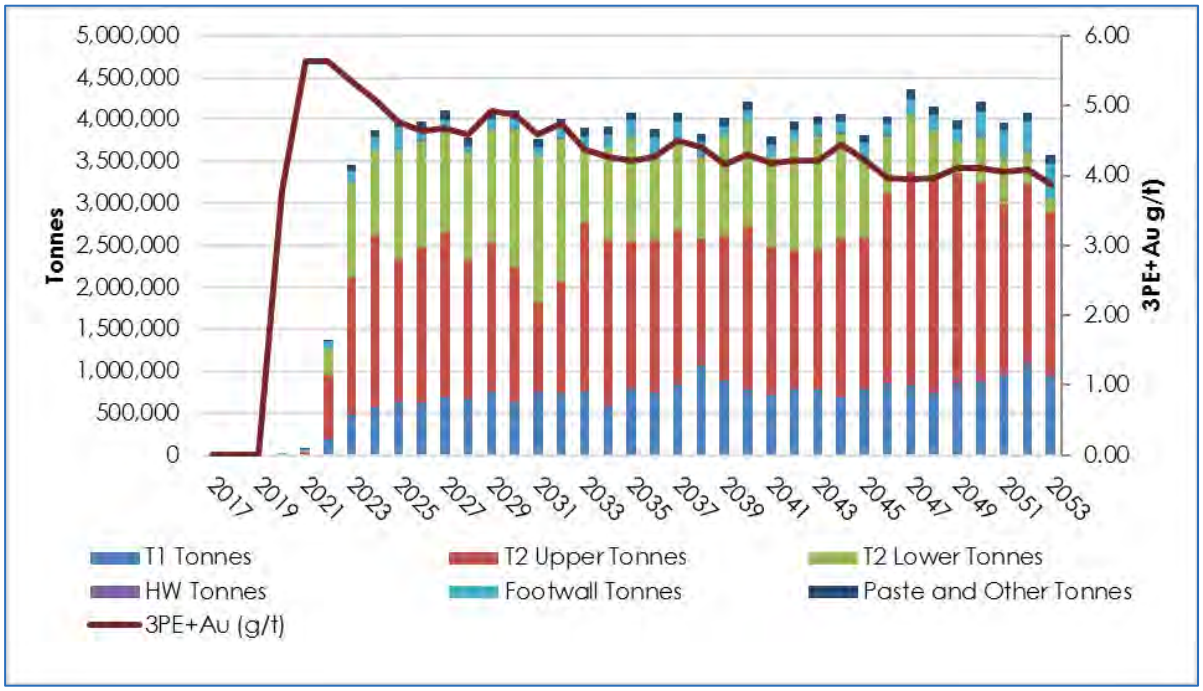


Figure by Stantec, 2017.

Table 16.41 Mine Production by Method

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Ore Development	kt	11,107	–	–	–	15	44	275	653	538	429	564	1,899	2,992	3,698
NSR	$/t	160	–	–	–	140	148	200	184	170	174	172	169	151	151
Platinum	g/t	1.96	–	–	–	1.68	1.74	2.58	2.25	2.02	2.12	2.18	2.12	1.82	1.85
Palladium	g/t	2.05	–	–	–	1.74	1.97	2.77	2.51	2.10	2.21	2.21	2.21	1.92	1.89
Gold	g/t	0.30	–	–	–	0.24	0.27	0.33	0.32	0.32	0.31	0.31	0.30	0.28	0.31
Rhodium	g/t	0.14	–	–	–	0.12	0.12	0.19	0.17	0.14	0.15	0.15	0.15	0.12	0.13
Copper	% Cu	0.17	–	–	–	0.16	0.17	0.17	0.18	0.20	0.18	0.17	0.17	0.17	0.17
Nickel	% Ni	0.35	–	–	–	0.33	0.34	0.35	0.37	0.39	0.38	0.35	0.34	0.35	0.34
Sulphur	% S	0.85	–	–	–	0.84	0.87	0.82	0.92	1.02	0.94	0.87	0.81	0.83	0.83
3PE + Au	g/t	4.45	–	–	–	3.79	4.10	5.87	5.24	4.57	4.79	4.85	4.77	4.14	4.17
Longhole	kt	93,133	–	–	–	–	21	1,086	2,735	3,181	3,356	3,320	16,965	25,308	37,161
NSR	$/t	152	–	–	–	–	170	186	185	182	170	165	165	147	141
Platinum	g/t	1.88	–	–	–	–	2.17	2.48	2.36	2.25	2.08	2.03	2.09	1.80	1.73
Palladium	g/t	1.95	–	–	–	–	2.22	2.61	2.54	2.45	2.22	2.12	2.16	1.89	1.76
Gold	g/t	0.29	–	–	–	–	0.29	0.30	0.31	0.32	0.31	0.30	0.30	0.27	0.29
Rhodium	g/t	0.13	–	–	–	–	0.14	0.18	0.17	0.16	0.15	0.14	0.15	0.12	0.12
Copper	% Cu	0.16	–	–	–	–	0.17	0.15	0.17	0.18	0.18	0.17	0.16	0.16	0.16
Nickel	% Ni	0.33	–	–	–	–	0.34	0.32	0.34	0.36	0.36	0.35	0.34	0.33	0.32
Sulphur	% S	0.80	–	–	–	–	0.85	0.71	0.82	0.90	0.89	0.87	0.81	0.80	0.79
3PE + Au	g/t	4.25	–	–	–	–	4.82	5.58	5.39	5.18	4.75	4.60	4.70	4.08	3.89
Drift and Fill	kt	20,441	–	–	–	–	–	8	70	147	202	88	1,027	11,451	7,449
NSR	$/t	182	–	–	–	–	–	171	177	177	165	163	190	180	186
Platinum	g/t	2.28	–	–	–	–	–	2.06	2.01	2.20	1.96	1.98	2.35	2.24	2.36
Palladium	g/t	2.23	–	–	–	–	–	1.90	2.25	2.13	2.37	2.20	2.26	2.22	2.23
Gold	g/t	0.37	–	–	–	–	–	0.55	0.34	0.33	0.42	0.40	0.43	0.36	0.37
Rhodium	g/t	0.15	–	–	–	–	–	0.09	0.13	0.14	0.09	0.10	0.15	0.15	0.16
Copper	% Cu	0.18	–	–	–	–	–	0.18	0.19	0.18	0.19	0.18	0.20	0.18	0.18
Nickel	% Ni	0.37	–	–	–	–	–	0.38	0.42	0.38	0.31	0.32	0.39	0.37	0.37
Sulfur	% S	0.91	–	–	–	–	–	0.93	1.03	0.94	0.76	0.81	0.94	0.91	0.91
3PE + Au	g/t	5.03	–	–	–	–	–	4.60	4.73	4.80	4.85	4.67	5.20	4.97	5.12

Table 16.42 Total Mine Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Total Ore Mined	kt	124,681	–	–	–	15	65	1,369	3,457	3,867	3,987	3,971	19,890	39,751	48,307
NSR BDT20	$/t	158	–	–	–	140	155	189	185	180	170	166	167	157	148
Platinum	g/t	1.95	–	–	–	1.68	1.88	2.50	2.34	2.22	2.07	2.05	2.10	1.92	1.84
Palladium	g/t	2.01	–	–	–	1.74	2.05	2.64	2.53	2.39	2.23	2.14	2.17	1.99	1.84
Gold	g/t	0.30	–	–	–	0.24	0.28	0.31	0.31	0.32	0.31	0.31	0.30	0.30	0.30
Rhodium	g/t	0.14	–	–	–	0.12	0.13	0.18	0.17	0.16	0.15	0.14	0.15	0.13	0.12
Copper	% Cu	0.17	–	–	–	0.16	0.17	0.16	0.17	0.18	0.18	0.17	0.16	0.17	0.16
Nickel	% Ni	0.34	–	–	–	0.33	0.34	0.32	0.35	0.37	0.36	0.35	0.34	0.34	0.33
Sulfur	% S	0.83	–	–	–	0.84	0.86	0.74	0.84	0.92	0.89	0.87	0.82	0.83	0.81
3PE + Au	g/t	4.40	–	–	–	3.79	4.33	5.63	5.35	5.08	4.76	4.64	4.73	4.34	4.10

Note NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.

16.4 Underground Infrastructure

Underground infrastructure involves several components, such as ore and waste handling systems, dewatering, maintenance shops, fueling, ventilation, etc. Stantec worked with DRA and others on the underground infrastructure to ensure that the functional specifications were aligned with the mine design. This Section includes the three items in Stantec's scope for the Platreef 2017 FS: ore and waste handling down to the crushers, ventilation/ refrigeration, and production return water. The additional facilities were reviewed and agreed upon, and Stantec completed the excavation designs to accommodate them.

16.4.1 Ore and Waste Handling Systems

The ore and waste handling system is designed with a capacity of 6 Mt/a. This is in excess of the initial production requirements of 4 Mt/a of ore and 0.4 Mt/a of waste, but allows for future potential increased ore production. Two ore handling facilities and one waste handling facility are planned. In addition, there are two underground crushing stations, one below each of the coarse ore bins midway between the 950 Level and the 1050 Level loading station.

Figure 16.36 illustrates the ore and waste handling system.

Figure 16.36 Ore Handling System (Isometric View)

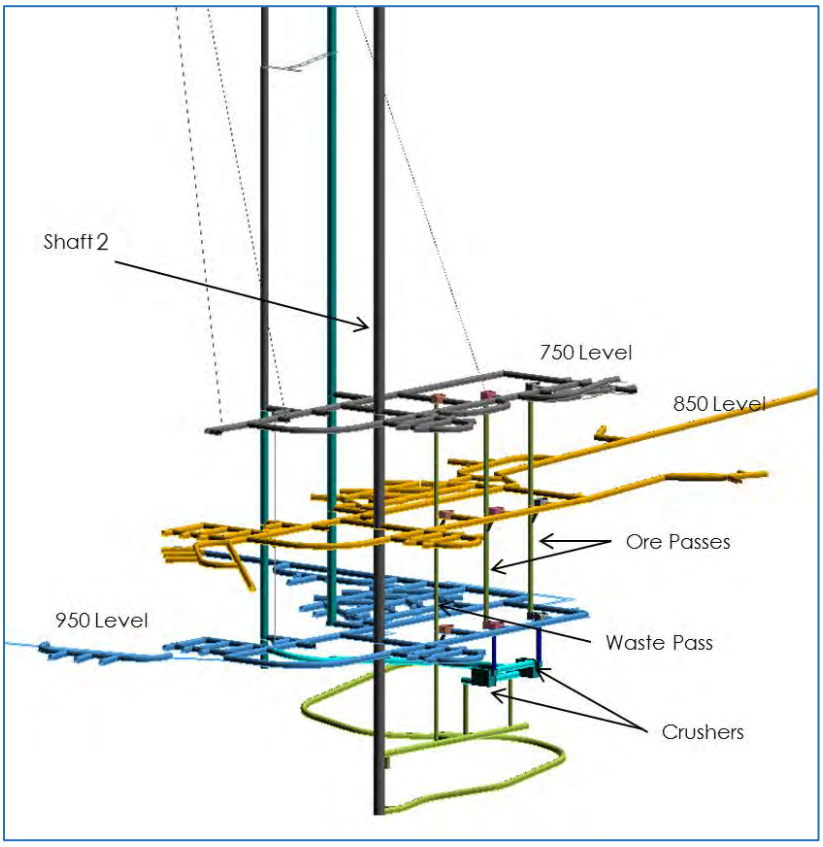


Figure by Stantec, 2017.

A temporary waste pass located approximately 45 m from Shaft 1 will allow early development on the levels. This vertical borehole will have tips on the 750 and 850 Levels, and will collect on the 950 Level. Subsequent rock handling will be via an LHD to the loading arrangement at Shaft 1.

There are multiple sublevel ore pass systems fed by LHDs transporting ore from the mining areas. 50 t haul trucks below the sublevel ore passes on the 750, 850, and 950 Levels transport ore to the main ore passes feeding the crushing systems. Ore below the 950 Level is transported by haul trucks up to the 950 Level main ore pass.

A single waste pass system is provided with finger raises on various levels that report the waste directly to the load-out conveyor, bypassing the crushing plants.

16.4.1.1 Temporary Rock Handling Pass

The temporary rock handling pass is located offset from Shaft 1. Off-shaft lateral development to this pass is driven with the shaft crews during shaft sinking. As soon as Shaft 1 is commissioned, a raisebore machine is set up on 750 Level to pilot the hole down to 950 Level. No development can occur on 750 Level or 850 Level until the piloting, reaming, and commissioning of the temporary waste pass is completed, as this temporary waste pass on the 950 Level is the only method of removing waste rock on these two levels.

On 850 Level, a finger raise will be drilled into the waste pass, which will allow for the removal of waste rock from 850 Level. All waste rock will be loaded with an LHD at the bottom of the temporary waste pass on the 950 Level and dumped into the skipping arrangement at Shaft 1.

16.4.1.2 Sublevel Ore Passes

Rock production above 950 Level will use sublevel ore passes for the movement of material down to the main levels. Rock from the development headings, Longhole stopes, and drift and fill will be mucked from point of origin directly into the sublevel ore passes. The sublevel ore pass design incorporates the following.

- 4 m diameter raise bore holes will be used for the ore passes.
- Ore passes will go from above 750 Level to 750 Level, from 750 Level to 850 Level, and from 850 Level to 950 Level.
- Finger raises will be provided at each sub level into the ore pass.
- The top of the pass and its fingers raises will have LHD–style grizzlies sized at 600 mm × 600 mm.
- Mobile rock breakers will be available in the mining zones or at the grizzly stations to handle oversized rock.
- Grizzlies will be constructed at sill level to allow oversized rock to be swept off by the LHD and later resized by the mobile rock breakers.
- Truck chutes at the bottom of the ore pass system will be Spillminator-style chutes per South African standards depicted by Platreef.

- Sublevel ore pass locations are designed to optimize LHD tramming versus development costs. The ore passes are spaced 400 m apart on the sublevels, which makes the average haul distance for all stopes 100 m on the footwall drift and the longest haul distance 200 m, excluding top cut distances. To accommodate for the top-cut distances, an average of 175 m was used as the one-way haul distance in the productivity calculations.

Refer to Figure 16.36 for the sublevel loading and transfer arrangement.

Figure 16.37 Sublevel Loading and Transfer Arrangement

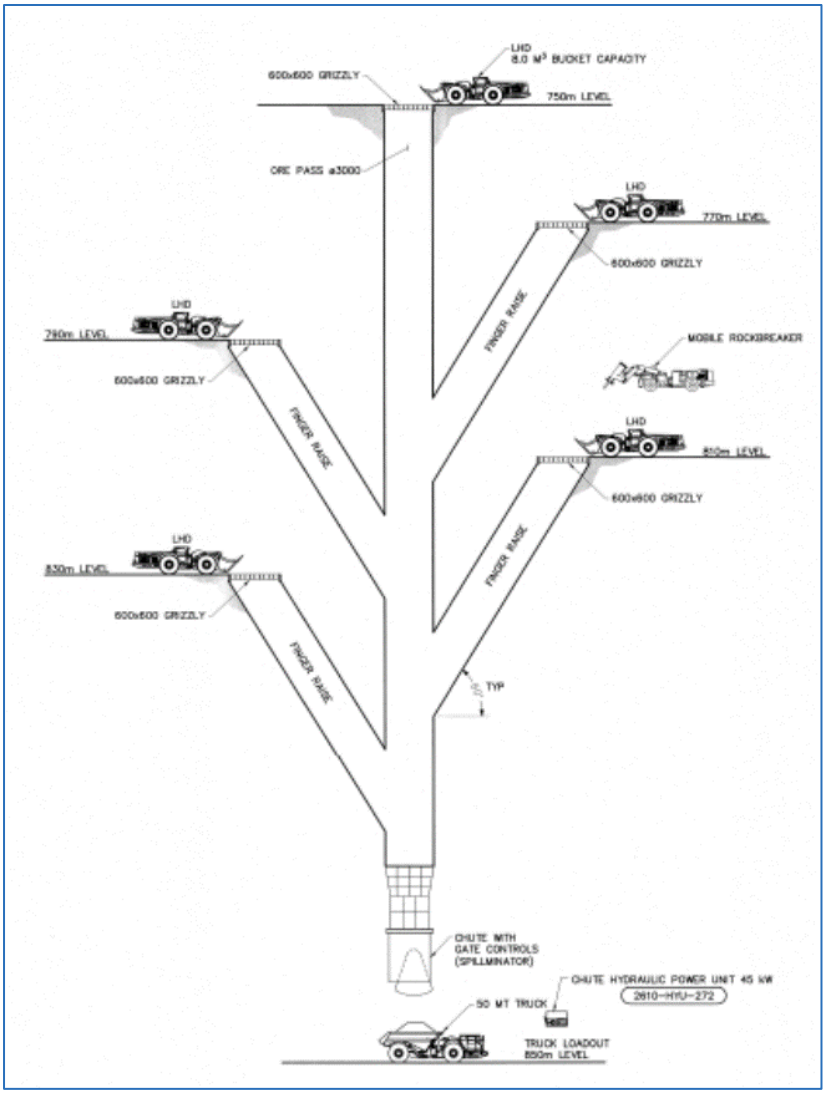


Figure by Stantec, 2017.

The 50 t haulage trucks are loaded by Spillminator-style chutes at the bottom of the ore passes and will haul across the main levels (750, 850, and 950) to the main ore pass. At the main ore pass, the material will be sized to 450 mm × 450 mm. Refer to Figure 16.38 for a typical truck loading arrangement.

Figure 16.38 Typical Truck Loading Arrangement

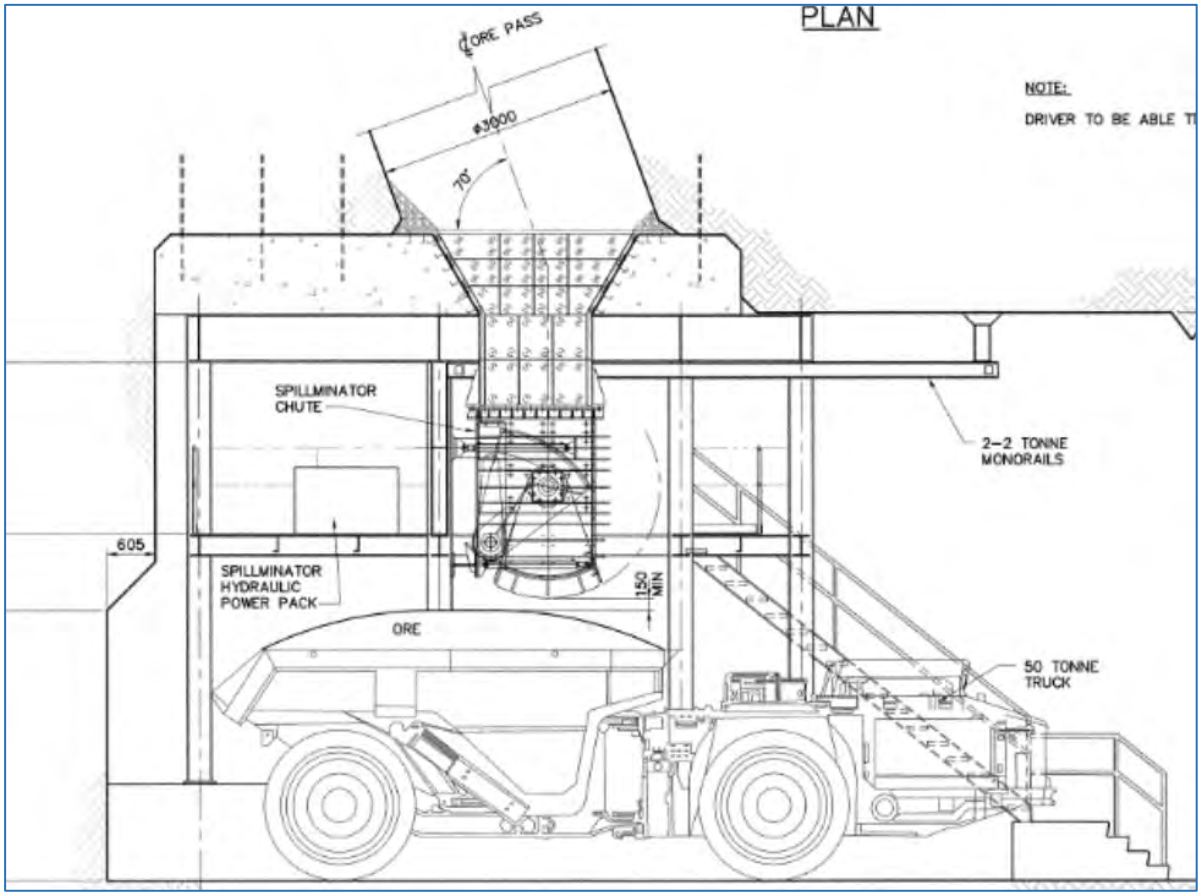


Figure by Stantec, 2017.

16.4.1.3 Rock Handling Below 950 Level

Rock production below 950 Level will use haulage trucks to truck the material to the 950 m Level ore or waste passes. Rock is removed from the headings or stopes and moved to a remuck area on the level. From the remuck areas a dedicated LHD will load the haulage trucks and the material will be trammed up to 950 m Level.

The current design accounts for 33% of the fill quantity in secondary Longhole Stopes and 25% of the Drift-and-Fill stopes to be direct waste rock. During the CRF period of filling, the secondary stopes are filled with 50% waste rock. Ideally, the waste rock will be moved from the closest generation point to the stopes being filled.

16.4.1.4 Main Ore Passes

There will be two 4 m diameter vertical raise bore ore passes above the 950 Level. Refer to Figure 16.39 for a schematic of one of the pass arrangements. The design of the ore passes will incorporate the following.

- Truck dump ore pass grizzlies sized 450 mm × 450 mm on the 750, 850 and 950 Levels are provided complete with stationary rock breakers with remote operational capability.
- At the 950 Level truck dump, a transfer chute is provided to control feed to the coarse ore bin. This feed bypasses the truck dump grizzly. The transfer chute is a Kiruna-style chute with low level detection. See Figure 16.40 for a comparison of Spillminator-style and Kiruna-style chutes.
- Capability to control chutes automatically from a remote location will be provided.

Figure 16.39 Main Ore Pass Arrangement and Truck Chutes

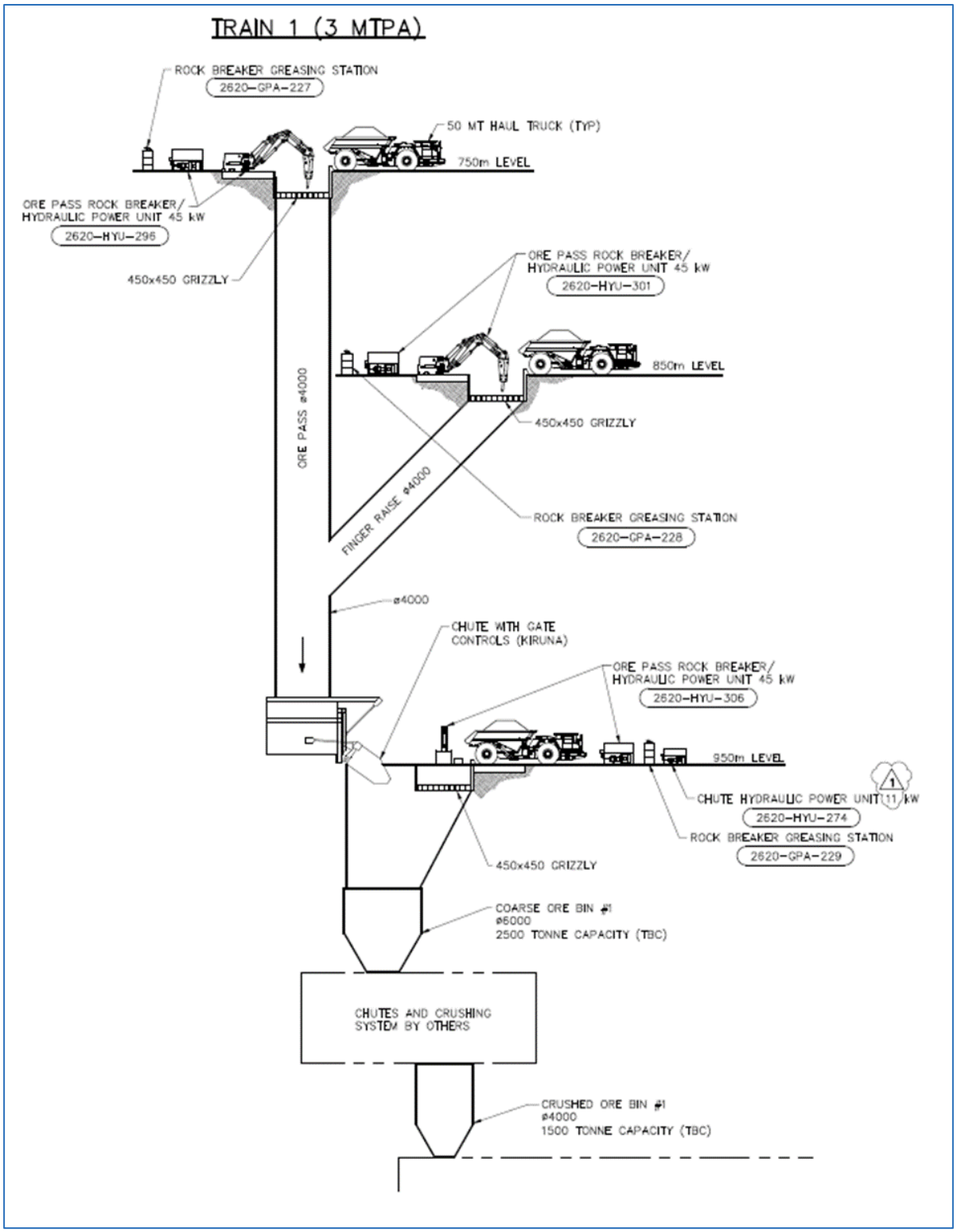


Figure by Stantec, 2017.

Figure 16.40 Chute Comparison

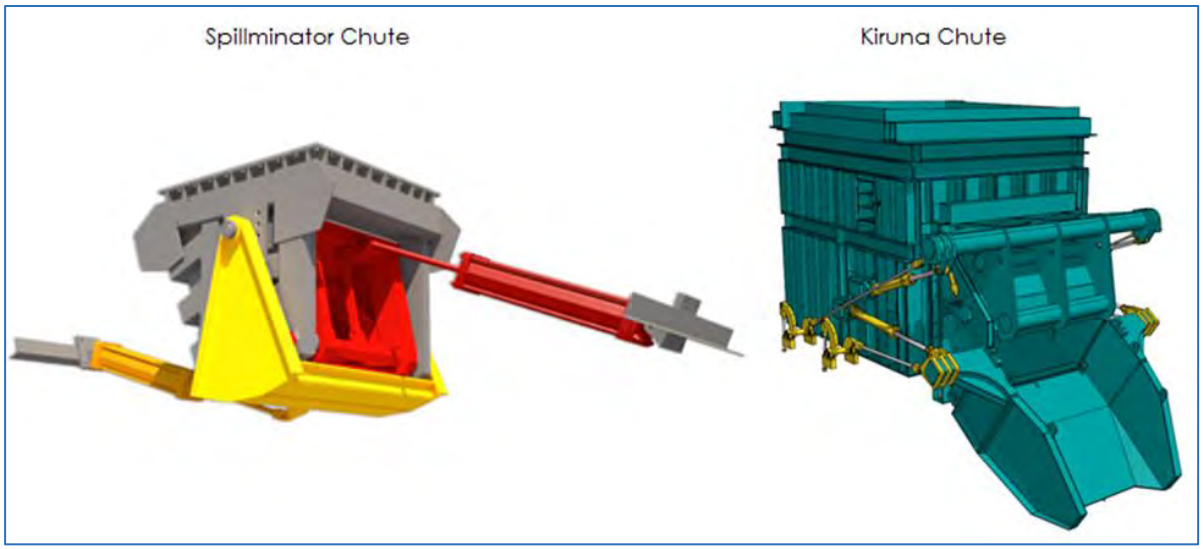


Figure by Stantec, 2017.

The Spillminator-style chute was selected for ore pass arrangements feeding haul trucks due to its unique safety capability of controlling a run of wet muck. The Kiruna style chute was selected for the transfer of ore from one ore pass to another due to its side-loading feature and capability to control ore flow well. It also facilitates the safe blasting of hang-ups better than the Spillminator-style design.

Below the 950 station is the coarse ore bin and crushing plants that feed into the crushed ore bin and the high-speed load-out conveying system. These systems were designed by DRA and Murray & Roberts.

16.4.1.5 Waste Pass System

The waste pass system from the 750 Level to the 1050 Level will be a pass 4 m diameter raise bore hole developed in two legs: one leg from 750 Level to 950 Level, and the other from 950 Level to 1050 Level. The waste pass design will incorporate the following.

- Truck dumps will be located on the 750, 850, and 950 Levels, with finger raise access into the main waste pass.
- A 300 mm × 300 mm grizzly will be located at each dump point.
- A stationary rock breaker with remote capabilities will be located at each grizzly.
- Ventilation control will be designed to prevent recirculation of air or excessive dust at grizzlies.

Figure 16.41 illustrates the waste pass system.

Figure 16.41 Waste Pass System

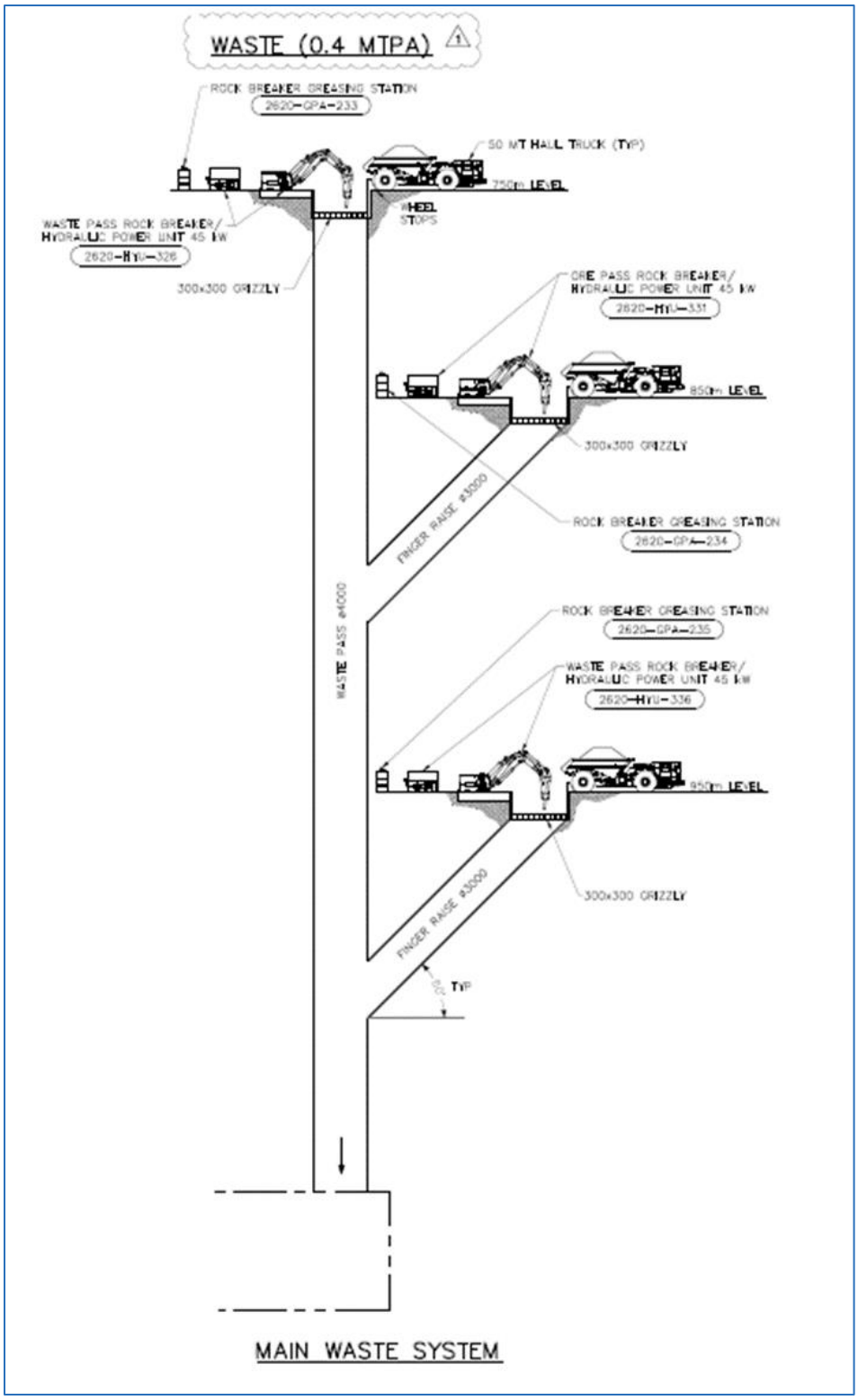


Figure by Stantec, 2017.

16.4.2 Mine Ventilation and Refrigeration Design

16.4.2.1 Ventilation

The purpose of the mine ventilation system is to provide sufficient air in quantity and quality to support the climate conditions of underground workings and, most importantly, to maintain working conditions at an acceptable level and in accordance to regulations. This Section describes the ventilation required for the Project.

The design principles that have been incorporated are proven and internationally recognised. The prime objective is to provide a safe and healthy environment in all underground workings and to comply with the regulations of the Mine Health and Safety Act of South Africa. Where little or no legislation is available, best practice and international guidelines are used.

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

Due to the geographically widespread mineralisation, the mine was divided into smaller mining zones and subzones. For the Project's ventilation study, these zones were further grouped into independent ventilation districts.

16.4.2.2 Method and Design Criteria

The ventilation system is designed as an exhausting pull system. The two main intake shafts (Shaft 1 and 2), located at the centre of the mining district, will provide fresh air, while Ventilation Raise 1 (near the intake shafts), Ventilation Raise 2 (in the north), and Ventilation Raise 3 (in the south) will be exhausts. Fresh air will be further distributed through a series of main stations, ramps, and haulages at the 750, 850, and 950 Levels. Regulators will be used to direct air to the active mining areas. Exhausted raises have been incorporated into both ends of the mining sublevels.

Compared to the Platreef 2015 PFS design, converting the Platreef 2015 PFS shop exhaust Shaft (3 m diameter) to a 6 m diameter raisebore (Ventilation Raise 1) and relocating it relative to Shaft 1 so it can be used as one the main exhausts columns significantly improves ventilation during development. In this new arrangement, Ventilation Raise 1 is used for most of the preproduction and ramp-up period until the full production, when Ventilation Raise 3 will be needed.

Another change from the Platreef 2015 PFS is that Shaft 3 has been replaced with a 6 m diameter raisebore (Ventilation Raise 2). This resulted in cost savings associated with conventional shaft sinking of the 7.25 m diameter shaft proposed in the Platreef 2015 PFS.

As for the design criteria, air quantities are based on a diesel emission dilution of 0.06 $m^3/s/kW$ of diesel engine rating and the required flow for mine cooling and refrigeration purposes. The diesel engine ratings are based on technical specifications for major equipment.

To address the health and safety of personnel, and as a general guideline in design, velocities are kept between 0.25 m/s as a minimum and 7.6 m/s as a maximum. Air velocity of 1 m/s in stoping areas and 2 m/s in the sublevels are used as the criteria, in which the former provides a minimum of 25 m^3/s of airflow.

The "critical velocity" range of 7–12 m/s was avoided, as this is the point at which water droplets will remain suspended in the airstream, increasing shaft resistance and adding extra stress on the fan. In horizontal intake airways, a maximum velocity of 6.5 m/s is specified, which is the practical limit before airborne dust becomes a problem.

16.4.2.3 Auxiliary Ventilation Arrangements

To achieve a production rate of 11,000 t/d, an exhaust-overlap ventilation system is proposed. The intake of the exhaust column must be carried to a point not exceeding 30 m from the face. The distance between the discharge of the force column and the advancing face should be designed in such a way to ensure that the ventilation duct will not be more than 20 m from the face after the blast. The minimum overlap distance between the exhaust column intake and the force column intake points should at least be 10 m and not exceeding 25 m. Fans in the exhaust column must be positioned in such manner that the exhaust column will remain under negative pressure, thus ensuring that no exhaust fumes leak back into the intake air flowing to the face.

In exhaust-overlap systems, it is recommended to interlock the exhaust fans, which are the primary source of ventilation, with all other electrical appliances and equipment. This will ensure that all other electrical appliances and equipment will also shut down in the event of the exhaust fans stopping. It is also recommended that the force fan only be positioned in the overlap section of the ventilation system and that no butterfly valves be used in any exhaust column in development ends.

At least one LHD will be required in each active production drift. By using remotely adjustable regulator controls, inactive production drifts will be ventilated with a fraction of the design airflow. The ventilation rates of 25 m^3/s for active production drifts and 10 m^3/s for inactive production drifts are considered as the basis of the design. Figure 16.42 shows the general arrangement of the equipment and ventilation layout.

Figure 16.42 General Ventilation Arrangement and Layout

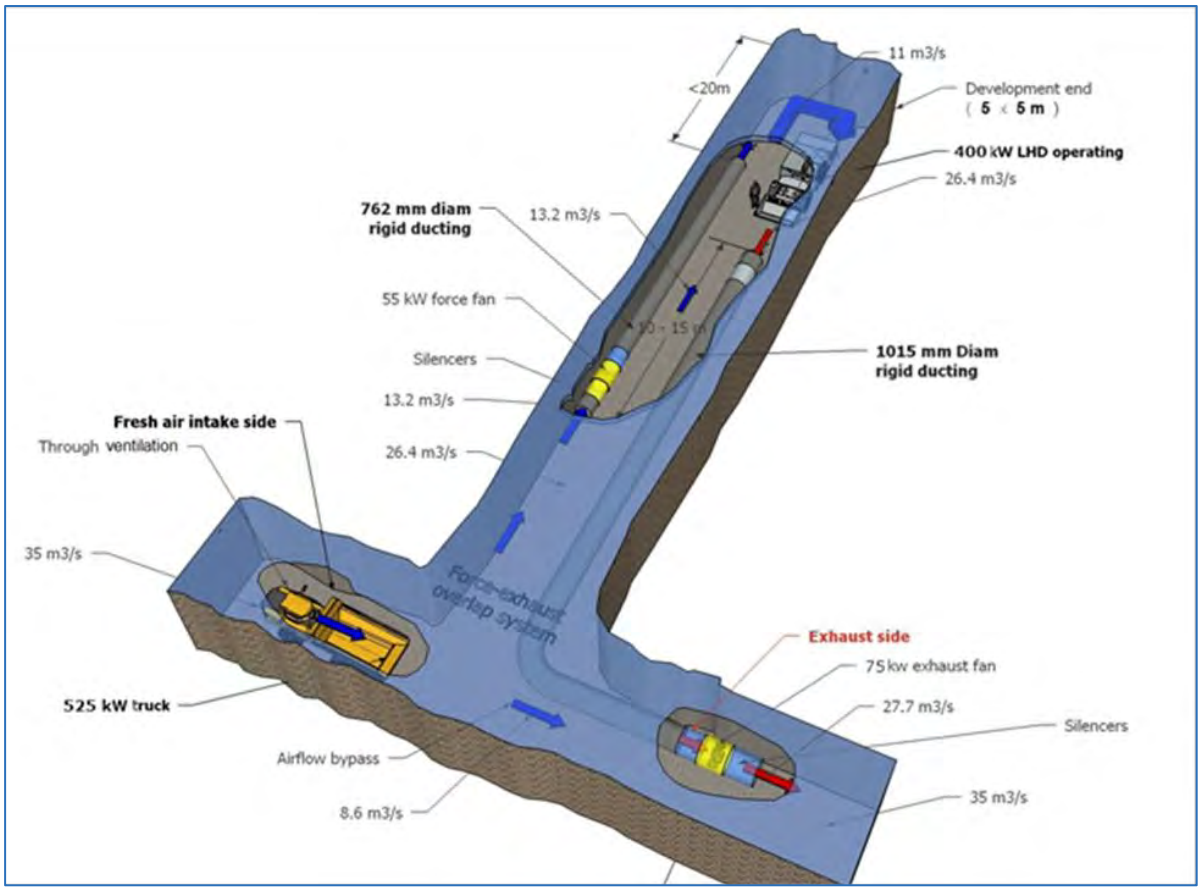


Figure by Stantec, 2017.

16.4.2.4 Lateral Airways and Ramp Systems

Lateral airways, for the most part, will be designed to provide access, haulage, and fresh air to the active mining areas. Dust control considerations and equipment cross-sectional areas will limit the air velocity. Provided that the air velocity is less than 6.5 m/s (where possible less than 6.0 m/s), there should not be any problems with dust control, but the velocities around the equipment with bigger cross-sectional areas will remain as a concern in shock loss calculations and safety of the personnel getting in and out of the vehicles.

16.4.2.5 Ventilation Modelling

Throughout the Platreef 2017 FS, various ventilation models were developed; among them, four major stages of the mine design, including preproduction, early stage, mid-life, and late-LOM, were selected for further study. The mid-life scenario includes all the deep and far-to-reach areas from the main shaft for cooling and refrigeration analysis.

Final mine design from Vulcan software was imported into the Ventsim simulation program for modeling purposes in DXF format. A cross-sectional area of 25 m² was assigned to all haulages and sublevels. Main ventilation intake and exhaust shaft properties were assigned as specified in the mine design criteria, and surface intake fresh air temperatures of 20°C and 30°C (WB and DB, respectively) were assumed.

16.4.2.6 Mine Air Cooling and Refrigeration

- The cooling design is based on achieving a reject WB temperature of 28.5°C. The designed surface intake air temperature of 20°C and 30°C (WB and DB, respectively) is the average temperature of the warmest month of the year. This section relates to the in mine heat load requirements and the associated cooling requirements. Primary heat load components are as follows.
- VRT and related thermal gradient.
- Artificial heat loads (e.g., trackless equipment, fissure water, broken rock, pumps, auxiliary fans, people, lighting).
- Ambient summer and winter surface fresh air intake temperatures.
- Mining depth and auto-compression.

Each system or combination thereof contribute either positively or negatively towards a deep underground mine's heat load and related cooling and refrigeration needs. The air mass flow applied for the project is 1,200 kg/s. The return air is split into 400 kg/s for the three exhausting ventilation raises positioned north and south of the ore body (with one located centrally close to the two main intake shafts).

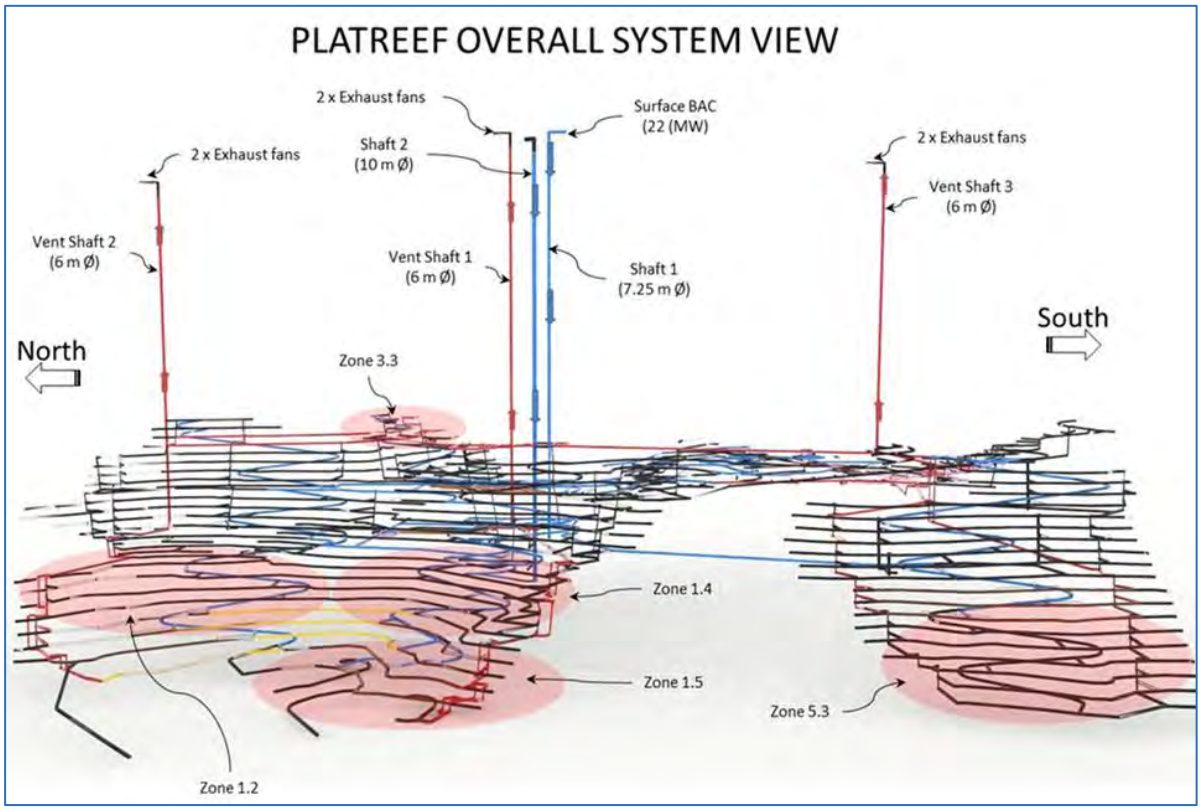
Taking into consideration the geographic location of the project and the trackless mining system, primary heat load contributors are as follows.

- VRT: Gradient = 18.3°C/km with a surface starting temperature of 25.5°C and the deepest mining level rock temperature range between 47.0°C and 48.0°C (1,200 mbs). Heat load contribution is around 6.6 MW.
- Trackless Equipment: Overall vehicle duties were calculated to be approximately 22.6 MW, and the related heat load in conjunction with the utilisation of all the equipment was calculated to be approximately 23.0 MW.
- Auto-compression Air Heat Load: Surface to shaft depth related to approximately 13.5 MW.
- Other In-Mine Heat Loads: Approximately 4.5 MW from auxiliary fans and pumping systems.
- Service and Fissure Water Heat Load Contribution through the Mine: It was estimated that about 120 L/s service water would be applied. An estimated 5.9 MW of heat could be applied to the underground mine environment and, due to the return water piping system, a conservative estimate of 20% is applied to heat load ingress, thus about 1.2 MW.
- Backfill and Other Heat Loads (people, lighting, etc.): Approximately 2.5 MW.

The calculated overall heat load is around 51.5 MW, including auto-compression. The fresh intake air cooling capacity calculated is 30.0 MW; thus, the mechanical cooling required to maintain the reject temperature of 28.5 °C is 22.6 MW. Including a 10% spare capacity for the BAC and provisions for 15% energy loss, the refrigeration plant design capacity would be 28.0 MW, which includes a 22.0 MW BAC system (horizontal, direct–contact, spray-chamber type). It is furthermore estimated to apply a precooling tower (PCT) for the service water distribution system, and initial calculations indicate a 5.0 MW PCT. This is to ensure service water recirculation in the mine does not constitute an additional heat load on the underground environment.

Figure 16.43 illustrates the overall project intake and return systems including some of the production areas allocated.

Figure 16.43 General Underground Mine Ventilation Arrangement



Note: Blue indicates the distribution of the fresh air intake system and red indicates return air from sections to surface. Figure by Stantec, 2017.

16.4.2.7 General Criteria and Design

To evaluate the heat load of the underground mining environment, it is important to determine the refrigeration and cooling design criteria, which are as follows.

- Reject WB temperature of 28.5 °C
- Ambient surface mean fresh air intake temperature of 20 °C and 30 °C (WB and DB, respectively)

Table 16.43 shows the design parameters for the cooling and refrigeration system application.

The overall ventilation system for the project uses two fresh air intake shafts (10 m ø and 7.5 m ø) and three primary ventilation raises (6 m ø each). One ventilation raise is located near the fresh air intake shafts (Ventilation Raise 1), and the other two ventilation raises are located at the extremity of each side of the north and south ore body (Ventilation Raise 2 and 3). Fresh air is reticulated through the underground mine via a pull (exhaust surface fans) system. Each ventilation raise will have two similar exhaust fans located on surface. The smaller intake shaft (7.5 m ø) will provide chilled fresh air (~50% of the total intake air cooled) from the surface-located BAC. Airflow quantity for the mine at full production is calculated as 1,200 kg/s when the air is distributed to the underground operation via the two intake shafts. Further details on the ventilation system are provided below.

- The ventilation system, and therefore the cooling system, supplied from surface are both designed on a "once-through" basis with series ventilation avoided.
- The reject WB temperature is set at 28.5°C. For the amount of cooling supplied, this temperature will not be exceeded in production headings where LHDs and trucks operate. LHDs, trucks, and drill rigs will be equipped with air-conditioned cabs.
- Production levels will be supplied with no less than 42 m^3/s of air (based on one truck and one LHD per level).
- For main level development, a quantity of 45 m^3/s (20% leakage included) has been allowed for during the primary development operation on the initial three levels, with two 1,500 mm ø galvanized steel exhaust ventilation ducts and one 1,015 mm Ø force duct in the headings.
- For steel ducting, there is 1.5 m Ø in primary and ramp development and 1.22 m Ø in production crosscuts.
- For main shaft sinking, the fresh air intake shafts (10 m Ø and 7.25 m Ø) are blind sunk and the ventilation raises and exhaust ventilation raises will be raise-bored (6 m Ø) to surface. It must be ensured that the bottom of the shaft is not closed by raise bore fragments during the shaft development to allow airflow for underground operations.
- The maximum air velocity in Shaft 1 will range between 11 m/s and 15 m/s.

The surface temperatures used in simulations were 20°C and 30°C (WB and DB, respectively), which is the average daily maximum temperature calculated over five years of records during mid-summer conditions.

Table 16.43 Summary of Ventilation Criteria

Basis of Design	Item	Criteria Range	Units
Best Practice	Ventilation Philosophy	Pull – Exhaust System	–
Best Practice (initially surface plant and BAC as required)	Cooling Philosophy	Heat-Load Reliant (mechanical cooling needed for this project due to mining depth, geology, and ambient intake air for the region)	
Rock Properties (Country Rock)			
Database Information	Density	2,860	Kg/m^3
	Thermal Conductivity	2.12	W/m°C
	Thermal Capacity	830	J/kg°C
Calculation	Thermal Diffusivity	0.89	m^2/s × 10^{-6}
Rock Properties (Ore Body)			
Database Information	Density	3,140	Kg/m^3
	Thermal Conductivity	2.94	W/m°C
	Thermal Capacity	870	J/kg°C
Calculation	Thermal Diffusivity	0.84	m^2/s × 10^{-6}
Current information	Surface Virgin Rock Temperature	25.6	°C
Current Measured Results (to be updated)	General Virgin Rock Temperature Gradient	0.0183	°C/m (vertical)
Heat and Cold Stress			
Legislation	Heat Stress	27.5 WB – Significant Hazard 32 DB – Significant Hazard 32 Radiant Temperature	°C
	Heat Stress – Max. Stop Work	≥32.5 WB ≥37 DB ≥37 Radiant Temperature	
	Cold Stress	≤9 WB – Significant Hazard ≤5 WB – Cold Stress Working Conditions Implemented	
Equipment Related	Heat Stress – Air Conditioned Cabs	≤30 WB	°C
Project Related	BAC Force Fans (×3)	~500 Pa – Axial-Type	–

16.4.2.8 Heat Loads, Ambient Surface Temperatures, and Underground Air Temperatures

Heat loads in underground mines come from various sources. As part of the design criteria, the following heat load aspects are identified.

- VRT and related thermal gradient.
- Backfill application and distribution system.
- Artificial heat loads (e.g., trackless equipment, fissure water, broken rock, pumps, auxiliary fans, people, lighting).
- Ambient summer and winter surface fresh air intake temperatures.
- Mining depth and auto-compression.
- Energy exchange unit efficiencies.
- Energy recovery system efficiencies.
- Chilled water reticulation methodology.
- Surrounding rock heat flow (VRT related).

Each system or combination thereof contribute either positively or negatively towards a deep underground mine's heat load and related cooling and refrigeration needs.

The heat load on the mine is directly related to the production profile. The current LOM plan has the mine operating until 2053. Initial ore production will be around 11,000 t/d and initial waste production around 4,000 t/d. The total mine air quantity increases as required to meet the production profile. Figure 16.44 shows the development, preproduction, and full production phases. The air quantity follows the production schedule, although the dotted line indicating the fan operation profile will be the quantity incurred from the operational perspective. The fan motors will be variable frequency drives, thus the change in quantity and pressure will be attained as needed. The maximum quantity (surface quantity) is estimated to be 1,200 m^3/s at peak production.

Figure 16.44 Surface Quantity with Production and Waste

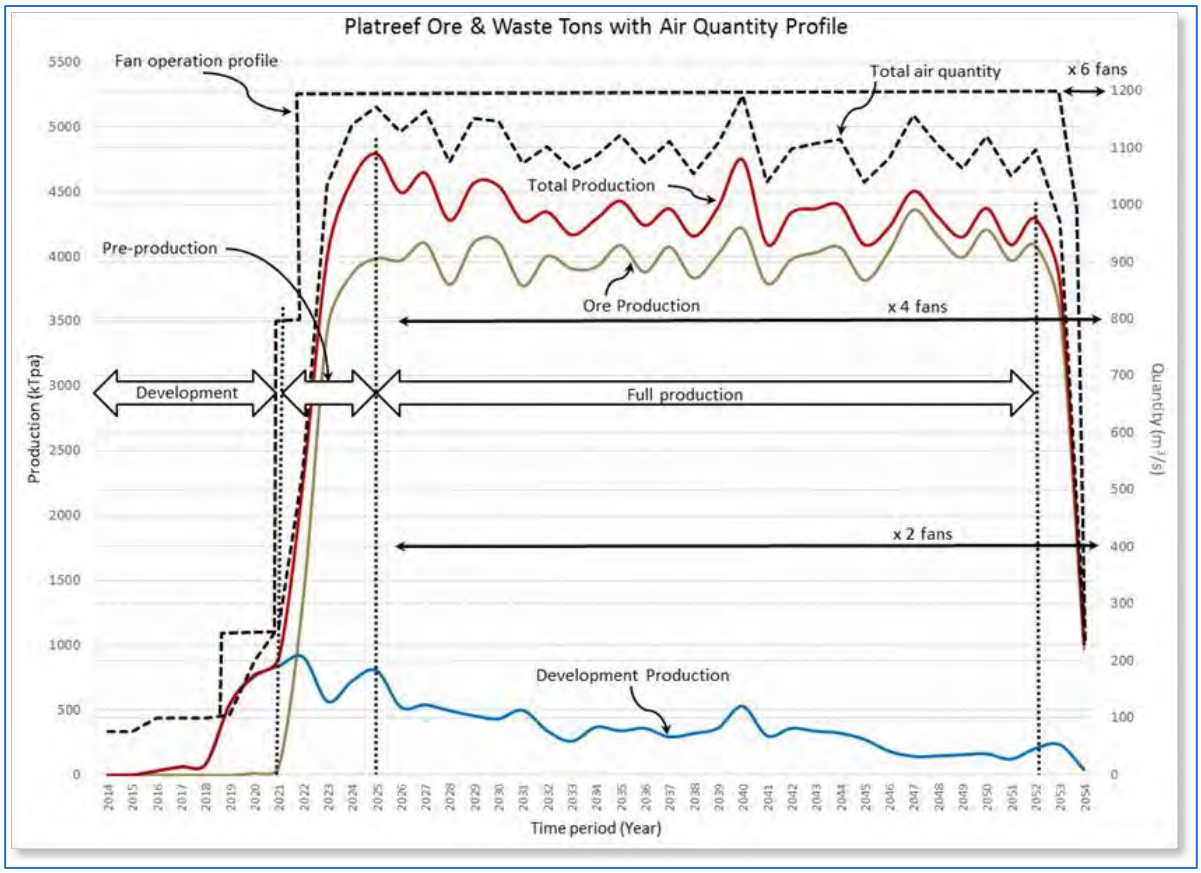


Figure by Stantec, 2017.

Figure 16.45 indicates the quantity and production profiles over the period 2014 to 2027. This is done to show the facilities (shafts) that become available for air and cooling.

Please note that the air quantities indicated are surface air quantities. Initially, the air quantity required for the underground operation was calculated. Using the air density of 1.16 kg/m³ in the underground environment and the surface air density of 1.02 kg/m, the required surface air mass flow was then calculated using the estimated underground quantity.

Figure 16.45 Quantity and Production Profiles (2014–2027)

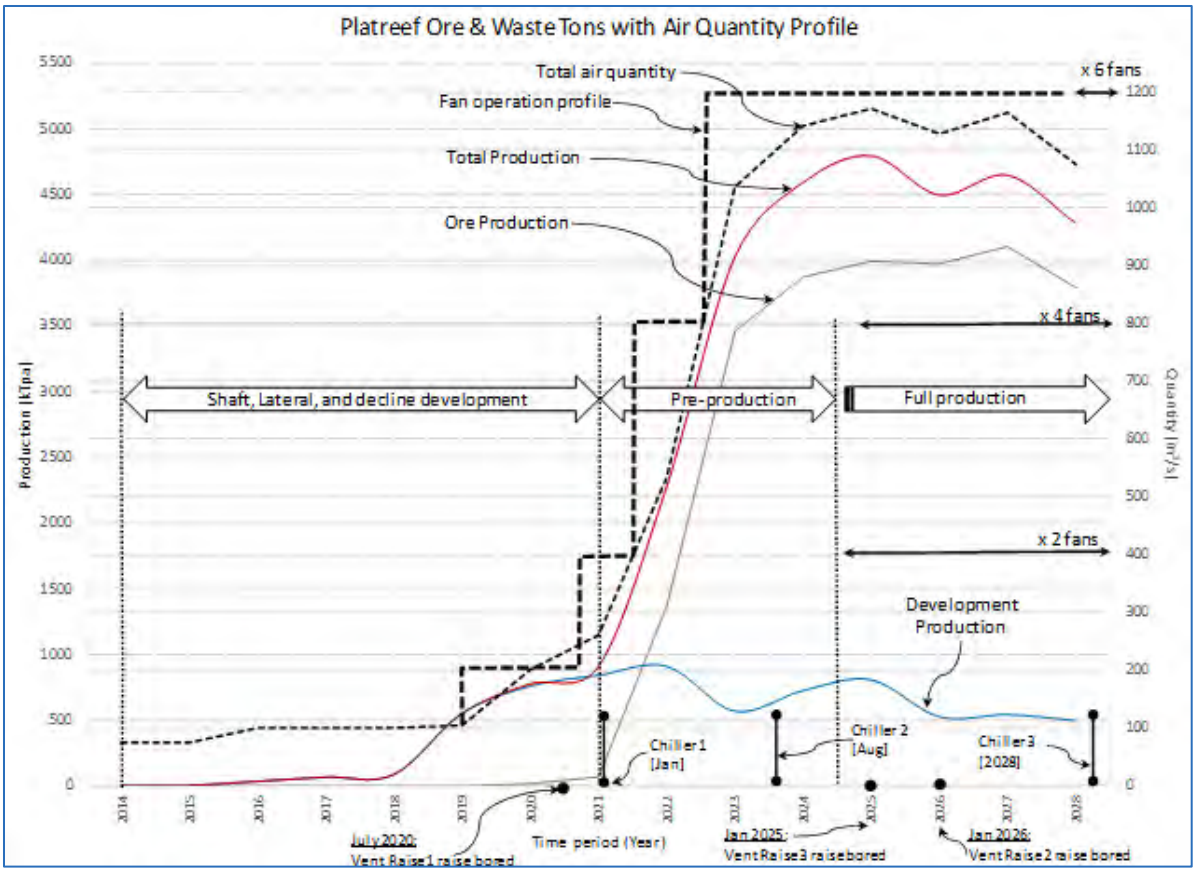


Figure by Stantec, 2017.

Some development will be done towards the Shaft 2 position with auxiliary fans and ducting. No cooling is required at this stage (initial level development around the shafts).

Early mine development prior to establishing cooling where conditions exceed the reject temperature of 28.5 °C will be mitigated with the use of closed-cab air conditioned mining equipment. Operator's assistants will follow a work-rest regime and will be assigned a schedule to limit impacting productivity levels.

Taking into consideration the location of the project and the trackless mining system, primary heat load contributors are as follows.

- VRT: Gradient = 18.3°C/km with a surface rock temperature of 25.5 °C and the deepest mining level rock temperature range between 47.0°C and 48.0°C (1,200 mbs).

For the simulation purpose, the rock age was taken as five years overall except for the vertical fresh air intake shafts, which were aged 10 years or more for the mean heat load estimation purpose over the project life. From various ventilation models developed and provided by the Stantec's engineering team, the most distant and deepest scenario (as the worst-case scenario) was used for the refrigeration and cooling study.

Rock properties are used in the simulation models to determine the heat load from the rock as thermal conductivity and rock-specific heat is important in these calculations. Due to the relative low rock thermal conductivity for this region, the rock tends to release less heat than in rock with a higher thermal conductivity such as in the Witwatersrand geological area. This is one of the contributing factors why cooling is required to be introduced at shallower depths below surface than for mines in other regions.

Measurements of thermal properties have been made by the Bernard Price Institute at the University of the Witwatersrand (Dr. Mike Jones). Rock mass surrounding shafts, main intakes, and service excavations can be represented by the properties of the surrounding rock, and stoping operations can be represented by the Platreef orebody, as indicated in Figure 16.42.

Table 16.44 Rock Properties

Description	Surrounding Rock	Platreef Ore Body	Units
Thermal Conductivity	2.12	2.94	W/mK
Specific Heat Capacity	830	870	J/kgK
Density	2,860	3,150	kg/m³
Diffusivity	0.89	0.84	X 10-6 m²/s

Table 16.45 indicates the effect of cooling and not cooling on surface. The system where no cooling is applied on surface is where the ambient surface intake fresh air WB temperature is 20°C (worst-case scenario taken), and when cooled on surface, the shaft intake air temperature is 12°C.

Table 16.45 Data for Air Cooling Capacity and Heat Load Calculated

Description	No Surface Cooling	Surface Cooling Applied
Fresh Air Surface Intake Temperature	20 °C WB	12 °C WB
Sigma Heat	2.5 kJ/kg	39 kJ/kg
Cooling Capacity	3 MW	50 MW

As indicated, the shaft auto-compression heat addition was discounted in this scenario shown. As shafts become deeper, the arriving station temperatures increase and cooling capacity of the air is reduced; thus, mechanical cooling such as surface refrigeration plants are required to maintain the reject temperature required underground.

Diesel equipment affects underground ventilation in the generation of both heat and exhaust gas. A diesel dilution factor of 0.063 $m^3/s/kW$ (100 cfm/HP – North American generic legal requirement) has been used for the Platreef 2017 FS. The general air dilution criteria is taken as 0.063 m3/s per kW rated diesel power at point of use, assuming EURO 3 engine, low sulphur diesel (50 ppm or less), catalytic converters, particulate filters, and good maintenance. A heat load value of about 2.0/kW is also applied as a good estimate for diesel vehicle heat load determination. Fuel usage is the optimum heat load value to be calculated and given as follows.

The diesel dilution factor of between 0.05 $m^3/s/kW$ and 0.06 $m^3/s/kW$ is only valid if the engines use low-to-medium sulphur emission diesel (50 ppm readily available), catalytic purifiers, and outlet pipe diesel particulate matter filters (where very old equipment is used).

For trackless equipment, overall vehicle duties were calculated to be 22.6 MW, and the related heat load in conjunction with equipment utilisation was calculated to be approximately 23.0 MW.

Table 16.46 shows the trackless equipment details, including power and heat load estimates. The underground machinery used in certain mining operations (e.g., hoisting machinery, trackless equipment, booster and auxiliary fans, and pumping equipment) all add to the in-mine heat load. Diesel vehicle heat load calculations are shown to concur with the general allowable minimum of about 1.5 to 3.5 times the rated output power of trackless machinery as heat ingresses into the underground workings. Naturally, as technology improves, machinery becomes more efficient in energy utilisation, consequently reflecting a reduction in heat load and exhaust fumes contaminating the surrounding air. An accurate means of calculating trackless diesel equipment heat load is from a fuel consumption and fuel property point of view. Diesel consumption for the LHD (LH517) is taken as 0.18 L/h/kW.

Table 16.46 Equipment Data and Heat Load Estimates

Mobile Equipment	Engine Power (kW)	Total Units	Utilisation Factor (%)	Unit Power (kW/unit)	Heat Load (kW)	Total Quantity (m^3/s)
Haul Truck – 50T	515	10	75	386	7,729	243.5
Haul Truck – 40T	375	4		281	2,251	70.9
LHD – 17T (Production and Development)	289	11		217	4,774	150.4
LHD – 14T (Development)	191	6	70	134	1,604	50.5
Cable Bolter	82	2	15	12	49	1.6
Jumbo – Rock Bolter	82	14		12	345	10.9
Jumbo – Twin Boom	111	8		17	267	8.4
Production Drill	111	5		17	167	5.2
Cable Bolter	103	0		15	0	0.0
Grader	100	2	40	40	160	5.0
Telehandler	81	0	20	16	0	0.0
Truck – Explosive	129	7	50	65	903	28.4
Truck – Crane	110	0	5	5	0	0.0
Truck – Lube	110	0	35	38	0	0.0
Truck – Personnel Carrier	110	7	10	11	153	4.8
Truck – Scissor Lift	129	4	40	52	413	13.0
Truck – Shotcrete Mixer	170	7	45	77	1,071	33.7
Truck – Shotcrete Sprayer	110	2	30	33	132	4.1
Truck – Utility	129	14	40	52	1,445	45.5
Truck – Water	160	2	30	48	192	6.1
Light Duty	81	8	35	28	451	14.2
Mobile Rock Breaker	191	5	15	29	286	9.0
Scaler	129	5	30	39	387	12.2
Multi-purpose Vehicle (Manitou)	60	3	35	21	126	4.0
Forklift	86	3	5	4	26	0.8
Roadway Compactor	118	2	30	35	142	4.5
Subtotal (underground heat load and quantity – kW and m^3/s)					23,071	727
Subtotal including leakage (15%) (m^3/s)						855
Workshops (×2) (m^3/s)						120
Fuel Bays (m^3/s)						40
Explosives Bay (m^3/s)						20
Total Quantity Underground (m^3/s)						1,035
Total Underground Mass Flow (kg/s)						1,190
Total Quantity (surface – m^3/s)						1,190

The air quantity indicated in Table 16.45 is only allowable when ultra-low sulphur fuel (<0.05%) is used and the vehicle is operated at sea density. The fuel type (sulphur content) to be used in the project is not known at this time, and assumed to be the minimum South African standard of 50 ppm, or 0.005% by volume. The standard of applying 50 ppm sulphur diesel used in the fuel for the project is applied and 0.063 m3/s/kW used. Diesel equipment will operate at increased levels above sea level (between 500 masl and at sea level) and it is assumed the project will reduce fuel flow of equipment to compensate for air density changes. This will mean that diesel engine output power will reduce slightly due to these changes.

The two diesel-type machines detailed above were used as they are the primary equipment in the fleet.

Other heat load components in an underground mine would generally be electrical machinery, people, production and development blasting activities, fissure and service water, lighting, etc. To indicate the heat incurred from auxiliary fans, Figure 16.45 shows the number of fans and their related heat load imposed on the environment.

Table 16.47 Auxiliary Fans and Heat Load Imposed

Fans Description	Number	Heat Load
Stoping/Development Fans (75 kW)	110	75 × 110 = 8,250 kW
Pumping Station Fans (45 kW)	8	8 × 45 = 360 kW
Stores (22 kW)	6	6 × 22 =132 kW
Total (LOM)	124	Total = 8,742 kW

It is anticipated that 20% fans listed is spare capacity and about 65% of full fan power is utilised at any time. Thus, the overall heat load for auxiliary fans is about 4,550 kW.

Backfill Heat Load

Heat flow from backfill applications (paste fill in this case) are mainly contributed by two systems: the pipe distribution system and the placement system (heat of hydration from cement contribution and, therefore, by backfill water flow into the environment). Cement will be added on surface in the batching plant before sending the backfill underground. Heat flow contribution is furthermore dependent on the backfill rock properties (e.g., thermal conductivity and specific heat). Typically, backfill temperature increase will be about 4.8°C/km of vertical transport (about double of water which is ≃2.34°C/km).

The calculation for water temperature change is similar except for the thermal capacity, which is about 4,187 J/kg °C. For typical backfill, the density range would be between 1.7 t/m³ to about 2.5 t/m³, and the thermal conductivity ranges between 1.3 W/mK and 1.4 W/mK. A value of about 5.0 kJ/kg of air was assumed at this stage of the study (tested and reference values from Euler de Souza – Kingston University in Ontario) in the heat load calculation; thus, Q = 21 kg/s × 5 = 105 kW per stope. Assuming four stopes in backfill per production zone and five active production zones throughout the mine at a given time 105 kW × 4 × 5 = 2,100 kW heat load is estimated.

Table 16.48 indicates the heat load data.

Table 16.48 Heat Load Summary

Description	Heat load (MW)
Rock	6.6
Trackless Equipment	23.1
Auto-compression	13.5
Auxiliary Equipment	4.6
Water Distribution	1.2
Backfill Heat Load	2.1
Other Heat Loads	0.4
Total Heat Load	51.5

Other heat loads such as broken rock (explosive heat load), people heat load (400 W/person), lighting, and other minor contributors were also estimated at about 230 kW in total (4,000 light bulbs at 60 W each, 400 people at 400 W each, and 0.5 W/m for broken rock at 61,700 m traveling length). The overall heat load determined is around 51.5 MW, including auto-compression.

16.4.2.9 Cooling Requirements

The air cooling capacity calculated is 30.0 MW; thus, the mechanical cooling required to maintain the reject temperature of 28.5°C is approximately 22.0 MW. The refrigeration plant capacity needed to supply the cooling estimated is approximately 28.0 MW (15% energy losses included). The current estimate is to apply a 28.0 MW surface refrigeration plant and a 22.0 MW BAC system (horizontal, direct-contact, spray-chamber type).

When the surface WB temperature is increased to allow for 21°C and 22°C fresh air intake scenarios, the cooling requirement changes as indicated in Table 16.49.

Table 16.49 Cooling/Refrigeration Requirements for Surface Wet Bulb Temperatures

Description	Bulk Air Cooling Requirement	Refrigeration Plant Requirement
20°C Intake Temperature	21.5 MW	28.2 MW
21°C Intake Temperature	25.7 MW	33.6 MW
22°C Intake Temperature	30.1 MW	39.3 MW

In selecting ventilation/cooling systems and component sizes, it is important to evaluate the alternatives available. From start of production and the ultimate depth scenarios, thermal issues dominate the design criteria. The analysis included evaluates the heat load for different depths and intake distances and is based on overall energy balances to determine the required primary cooling needs.

The mining operation will make extensive use of diesel equipment. At shallow depths of mining, diesel gas dilution is usually the dominant design criterion, but at the Platreef Project, thermal aspects become a concern in addition to the shallow mining levels.

In determining the mine heat load, Ventsim simulation software was used to study the cooling and refrigeration requirements using existing models provided by Stantec. Detailed simulations were carried out for a number of critical scenarios to determine the heat loads, air temperatures, and airflow requirements by Stantec's engineering team. The detail of the simulation work done for the heat load determination is provided as follows.

The ventilation system design and simulation work was done by Stantec, as well as the model provided for the heat load input and evaluation requirements. Important parameters such as the rock properties and the diesel fleet, including the geothermal gradient, was allowed for in the models. Figure 16.46 illustrates the model simulation for the worst-case scenario.

Figure 16.46 Worst-Case Scenario Simulation

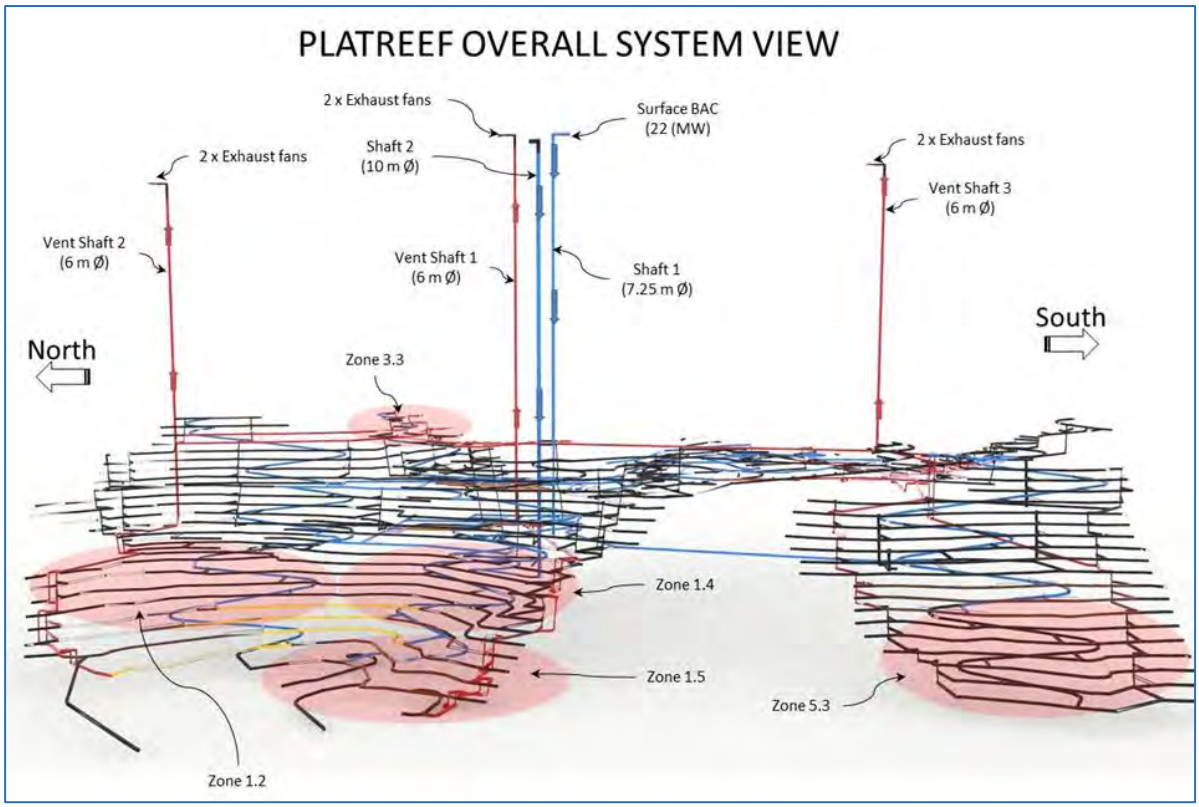


Figure by Stantec, 2017.

The reject temperature was not exceeded in the simulation, as 22 MW of surface BAC was applied to counter the heat ingress from the various components indicated previously. It is further noted that each active level in the production zones was supplied with 50 m^3/s of fresh air. This scenario then included the 1,200 kg/s of fresh air downcast to the underground workings with 600 m^3/s of air that was cooled on surface and distributed via Shaft 1.

For the development simulation, it was required to apply at least 7 MW of surface plant cooling for the development end cooling requirements. It is recommended to apply 7 MW of plant capacity in stages for cooling the underground environment as development and production expands to full production. This indicates that the full BAC and refrigeration plant system will need to be installed at that time (although not fully operated at the development initial stages).

Refrigeration is required to provide sufficient cold air to ensure that the development and stope reject temperatures remain within design parameters (average development and stoping reject WB temperatures of 28.5°C). Making use of small heat exchangers underground closer to the point of need is most efficient, although with piping and pumping compromise some of the energy savings attained. Additional energy conservation schemes could include ramping down of the cooling plant during winter months. The cost of power in South Africa is significant (≃R$0.85 per kWh; for continuous consumption, a single kW will cost about R$7,446 per annum) and should be monitored to ensure power consumption on the projects is maintained at acceptable levels. Eskom tariffs could increase significantly in the future due to electrical power demand and distribution in the country.

Figure 16.47 illustrates the cooling requirements over time, including initial development up to full production.

Figure 16.47 Plant Cooling and Air Quantity Distribution Schedule

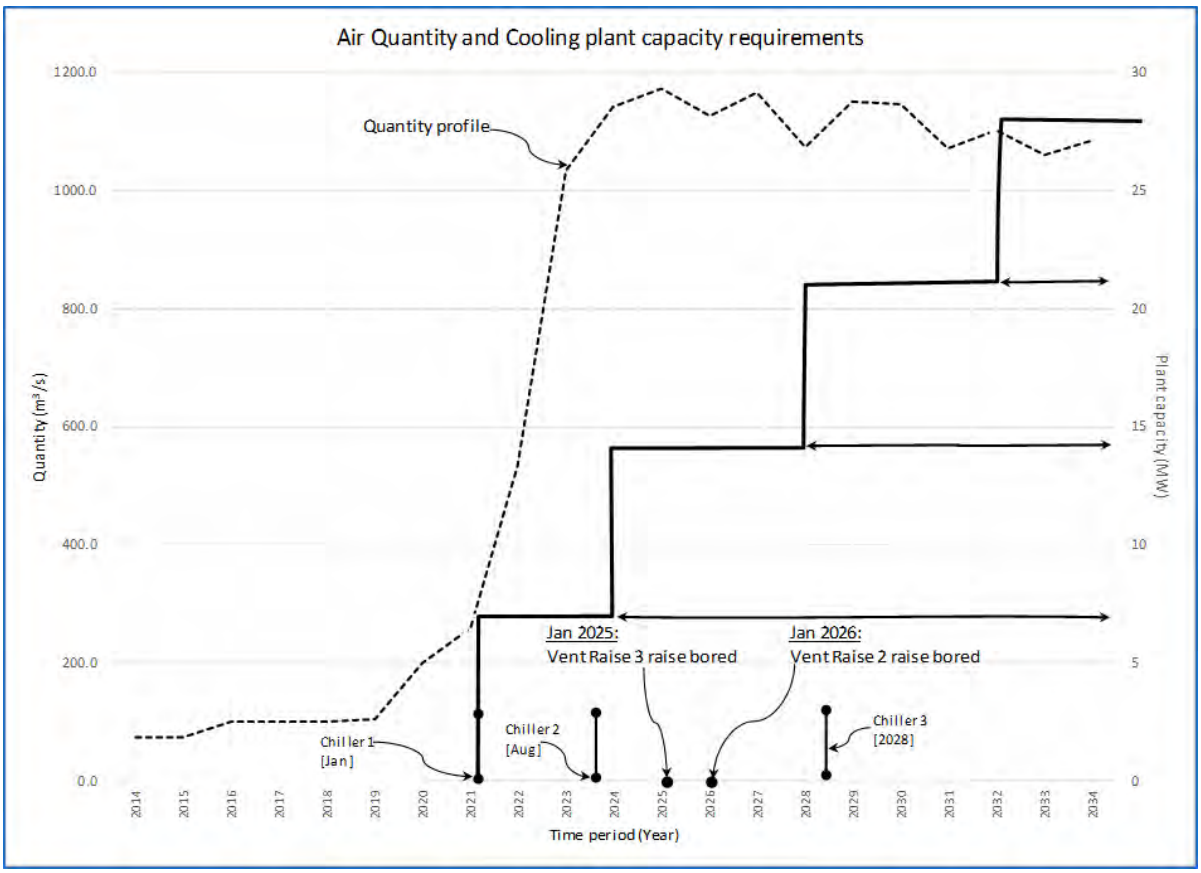


Figure by Stantec, 2017.

Figure 16.47 illustrates the cooling distribution as required over the production and development period leading to full production. Introduction of cooling could vary slightly and is dependent on the specific positions of development and production areas in the underground mine. The specific cooling required could be verified during construction by applying the actual development and production tonnages. The first 7 MW of plant cooling capacity was determined from the initial development on the first four levels leading from the main shaft facility. The remainder of the cooling schedule was based on a similar time frame (slight adjustment with a reduction of 3 months per time frame or stage) to ultimately arrive at the 28 MW plant duty (22 MW BAC cooling) cooling applied to the full production underground operation.

Shaft 1 is the fresh air downcast shaft and will be used exclusively for distribution of the chilled air from surface to underground. An underbank chilled air distribution system will be constructed to provide a path for the chilled air transition from the BAC to the vertical shaft system. This underbank system connects the BAC to the vertical shaft via an airway. Chilled air from Shaft 1 is dedicated specifically for deeper and hotter areas of the mine. The system is designed such that fresh air from Shaft 1 and Shaft 2 will not be mixed to achieve the maximum cooling for the deeper and hotter zones, such as Zones 1 and 5.

Due to the back pressure from the BAC, nominal leakage of chilled air through the Shaft 1 collar is expected. This will ensure that the fresh air from surface will not be mixed with the cold air from the BAC.

Duties for the overall cooling system are presented in Table 16.50.

Table 16.50 Overall Cooling System Duties

System	Duty
Bulk Air Cooler	22.0 MW (maximum of 24.0 MW)
	3 × 200 m^3/s Force Fans (total = 600 m^3/s)
Refrigeration Plant	Evaporator: 4 × 8.0 MW (32.0 MW)
	Water Flow Rate: 1,200 L/s
	Condenser: 4 × 6.6 MW (26.4 MW)
	Water Flow Rate: 600 L/s
Condenser Cooling Tower	33.9 MW
	Water Flow Rate: 1,200 L/s
	4 × 350 m^3/s, 44 kW Exhaust Fans (total = 1400 m^3/s)

Due to the locational contributing factors of the project area (VRT and mean mid-summer ambient temperature), the widespread geological system, the mining depth (about 1,200 mbs), and using trackless equipment, the heat load and associated cooling contribution needed to be determined.

The heat load components were evaluated and their respective heat load contributions were determined. It is Important to note that the project heat load was determined for mid-summer conditions (i.e., 20°C WB and 30°C DB). The design reject WB temperature applied in the underground mine is 28.5°C. An indication of the cooling requirements for a WB temperature increase using 21°C and 22°C was done and the related cooling needs determined. The cooling requirement for surface WB intake temperatures are as follows.

- 20°C WB – 22.0 MW (plant capacity = 28.0 MW)
- 21°C WB – 26.0 MW (plant capacity = 34.0 MW)
- 22°C WB – 30.5 MW (plant capacity = 39.5 MW)

Energy losses of 15% were allowed for in the plant capacity determination. Taking that the BAC and condenser cooling tower will be located in close proximity and good pipe insulation is applied and maintained, the losses could be reduced to 10% or less, and this will ultimately be evaluated for construction purposes.

During winter conditions, the cooling requirement will reduce to about 16.0 MW; thus, a reduction of 25% in mid-winter conditions (months between June and July) is anticipated.

The primary heat load component in the mine will be the diesel vehicles, followed by auto-compression, strata heat load, and lesser heat load contributions from auxiliary equipment such as fans, backfill, people, lighting, rock transport, and fissure water. There were utilisation factors assigned to the different diesel vehicles used underground, and any changes to the values will influence the overall heat load (and air quantity needed) of the mine. However, the utilisation values applied throughout the study are practical and typical for this type of mining operation.

To conclude, the total estimated heat load is approximately 51.5 MW. The in-mine air cooling capacity is 30.0 MW; thus, the additional cooling needed to maintain the reject design temperature of 28.5°C is 51.5 MW – 30.0 MW, or approximately 21.5 MW. In order to allow for maintenance and emergency requirements, it is suggested that a 10% additional capacity be allowed for design; thus, approximately 24.0 MW BAC is required. Cooling storage has been allowed for peak power demand.

The cost associated with the cooling system was determined to be US$15.8 million, including spare equipment allowances. The cooling system will consist of four parts: the BAC horizontal spray chamber, the refrigeration plant, the condenser cooling tower unit, and a cooling storage dam facility. This system is estimated to have a base footprint of about 67 m × 65 m. Each system will have the following footprint.

- BAC: 32 m L × 15 m W × 6.8 m H (including the three force fans)
- Refrigeration Plant: 36 m L × 17.3 m W × 7.3 m H
- Condenser Tower: 48.3 m L × 12m W × 15.6 m H

It is suggested that a surface PCT be installed to reduce the underground service water temperatures, which could induce a greater heat load to the system than calculated. The PCT should have a cooling capacity of 5 MW to reduce the service water temperature from 28 °C to 22 °C. The footprint of such a PCT would be about 8.5 m W × 8.5 m L × 11 m H.

The surface horizontal BAC will be connected to a plenum (underbank fresh air intake system) that is connected to the 7.25 m Ø vertical shaft. Chilled air (10°C saturated) will flow into the shaft from the BAC and mix with a small amount of fresh air intake through the surface of the downcast shaft system.

The cooled service water (120 L/s design) is reticulated back from the collection dam at the bottom of the PCT to the underground operation. The exhaust fan handling 317 kg/s at 150 Pa is located at the top portion of the PCT. Two 55 kW water pumps are allowed for (one standby).

16.4.2.10 Production Return Water System

Production Dewatering

During mine production, water will be collected in sumps located throughout the mine. The upper level sumps will transfer water through boreholes to lower level sumps where mine duty pumps will then transfer the discharge water through a pipeline and borehole system to the main pump and clarification system located at Shaft 2.

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

The mine water inflow is estimated to be 35 L/s during maximum production. The pumping system is designed for 150 L/s to account for spikes from initial groundwater inflows and paste backfill flushing. Each sump will not operate on a continuous basis but will have an overall utilisation dependent upon actual mine water inflows and mine water usage. This system will allow for improved settling in the main sumps and less fines reaching the main pumps at Shaft 2.

The following Sections discuss the various types of sumps on each level that comprise the overall production return water system.

750 Level Sumps

The two sumps on the 750 Level are gravity transfer sumps and are provided to collect water from the development and mining areas above the 750 Level. These sumps are designed to transfer mine water and solids to boreholes which report to the collection sumps on the 850 Level. Depending on ground conditions, casing is advisable to keep the boreholes open.

850 Level Sumps

There are three gravity transfer sumps on the 850 Level identical to the 750 Level. These sumps are intended to collect water from the development and mining areas above the 850 Level and from the gravity transfer sumps on 750 Level. Boreholes will transfer mine production water from the 850 Level sumps to the 950 Level sumps or in some case directly to the 950 Level main pump station clarification system.

There also three slurry transfer sumps on this level, which are designed to collect water and solids from the level and pump it to the gravity transfer sumps. The Slurry transfer sumps are a small cavity-style sump proven to work extremely effectively in the transfer of mine water, while alleviating the cleanup requirements of solids that is common with most transfer sumps.

One single-bay, settler-style sump is provided at 850 Level for shotcrete borehole flushing. It also reports to the 950 Level main pump clarification system.

950 Level Sumps

The 950 Level will be the main level for the collection and transfer of mine production water to the mine discharge pumping system located at Shaft 2. The sumps on the 950 Level will collect water from the development and mining areas above the 950 Level, sumps on the 850 Level, and the sumps below the 950 Level. They collect mine water that has not already been directed to the main pumps from the 850 Level.

There are two dual-bay, settler-style sumps on the 950 Level. Both settle out the solids prior to pumping the mine water to the main pumps also on 950 Level. For settling capacity of all the sumps of this nature, refer Table 16.51.

Table 16.51 Sump Settling Capacity

Settler	Capacity per Side (L)	Inflow (L/s)	Percent Solids (w/w)	Solids (L/s)	Sump Retention Time (days)
North Sump –950 Level	154,688	24	1.0	0.080	22.4
North Sump A – 1050 Level	103,000	24	0.5	0.040	29.8
North Sump –1170 Level	103,000	16	1.0	0.053	22.5
North Sump –1050 Level	103,000	16	1.0	0.053	22.5
South Sump –950 Level	103,000	24	0.5	0.040	29.8

1050 Level Sumps

There are three sumps on the 1050 Level. Two are dual-bay, settler-style sumps, while the other is a slurry transfer sump. All these sumps either directly or indirectly report the mine water to the 950 Level main pump clarification system.

1200 Level Bottom of Ramp Sumps

There are two bottom of ramp sumps on the 1200 Level. One is a dual-bay, settler-style sump, while the other is a slurry transfer sump. Both of these sumps report to their respective settler sumps at the 1050 Level.

16.5 Mining Opportunities

The following is a list of potential mining opportunities for Platreef.

- Continued optimisation of the stope design and sequencing to improve the grade profile during the early years of production.
- Remote operation of the fixed rock breakers at ore and waste truck dumps.
- Placing a small percentage of waste rock into primary stopes.
- Evaluating batch design to optimise cement and admixture usage minimising fill curing rates.
- Storing and hoisting development ore during the Shaft 1 hoisting phase.
- Reducing the angle of the footwall and hanging wall in production stopes as a benefit of the definition drilling programme.
- Reviewing the use of a small raisebore or a drill similar as an alternative to drilling and blasting the initial slot raise.
- Reducing the number of individuals supporting the development crews (e.g., service crews, materials handlers, operator assistants).
- Replacing the drop raise in Longhole Stopes with an Atlas Copco Easer to increase overall production of the stope.

16.5.1 Risks

The following is a list of potential mining risks for Platreef.

- Lack of available workforce with sufficient skills to meet the specified performance rates.
- Production and schedule constraints due to shaft pillar designs.
- Possible requirement of replacement ore and waste passes; the design provides for locations but they are not currently in the design.
- Uncertainty of raiseboring to 750 m in length and deeper in this location.
- The amount of lining required in the ventilation raises once they are reamed.
- Mining through the Tshukudu Fault.
- Timeliness of the definition drilling during the preproduction and early production stages.
- Cooling capacity during the summer peaks. This can be managed with a work-rest regime and supplemental underground air conditioning units.
- Mining underneath cemented paste fill or rock fill in the sill pillars (although less than 1% of the tonnage is mined under a sill pillar).

- The amount of additional S3 support that may be required in the later years of the mine life if ground conditions deteriorate and mining impacts are more severe.
- Difficulty handling the low-grade material during development.
- The stability of the accesses in the secondary stopes during and after the mining of the primary stopes.
- The impact on the Platreef site of any social uncertainty within South Africa during the project's life.

16.5.2 Recommendations

Stantec makes the following mining recommendations.

- Optimise the definition drilling programme required for the initial mining areas.
- Determine stope sizes and footwall and hanging wall angles once the block sizes in the block model are reduced due to more detailed definition drilling.
- Monitor fragmentation during the development stage to eliminate the need for secondary breaking on development rock.
- Maximise the flow of ventilation from Shaft 1 (cooled air) to the deepest and warmest mine workings.
- Monitor and optimise the first development through the Tshukudu Fault.
- Develop an operating procedure to allow waste rock to go into primary Longhole Stopes and Drift-and-Fill areas.
- Set up a programme for ore pass monitoring to ensure the longevity of the passes.
- Set up a ground control observation programme to proactively recondition the ground support as needed during the mine life.

17 RECOVERY METHODS

17.1 Summary

The Platreef process plant is designed to treat 4 Mtpa Platreef ore and includes all processing requirements from the run-of-mine (ROM) storage silos through to final concentrate plant load out and tailings disposal. The design is based on two 2 Mtpa milling and flotation circuits which were selected based on the concentrator production ramp-up profile. The modular approach with shared crushing, tailings disposal and concentrate handling circuit, allows for increased processing flexibility and introduces plant redundancy while allowing for phasing of capital.

The process design has been developed based on the test work findings and assessments, various desktop level trade-off studies and relevant DRA design information. The Platreef concentrator flowsheet is based on a conventional three-stage crushing and ball milling circuit followed by flotation. This single-stage milling and flotation process flowsheet is well known in industry, and has historically been proven as a suitable processing route for various platinum ores. The process description is presented in Figure 17.1.

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

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

The final concentrator layouts were decided taking into consideration the environmental and social impacts and aimed to minimise transfer and wear points in the ROM section, reduce building height and improve constructability and maintainability in the main concentrator area.

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

Figure 17.1 Concentrator Flowsheet

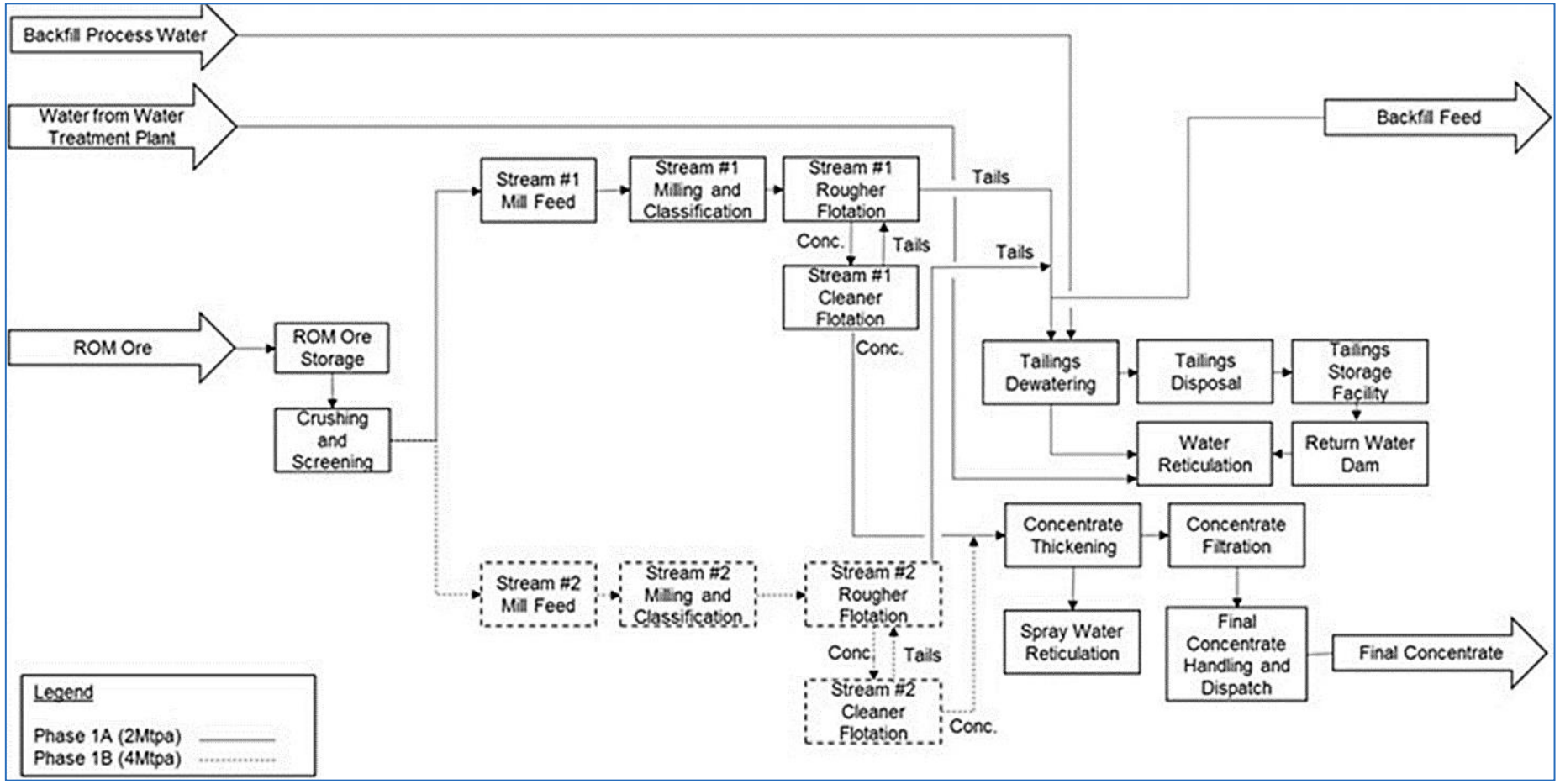


Figure by DRA, 2017.

17.2 Introduction

The process plant is designed to process 4 Mtpa Platreef ore and includes all ore processing requirements from the ROM storage silos through to final concentrate load out and tailings disposal. The concentrator consists of the following:

- 1 x 4 Mtpa combined crushing and screening circuit
- 2 x 2 Mtpa modules for the milling and flotation circuits
- 1 x 4 Mtpa combined concentrate handling circuit
- 1 x 4 Mtpa combined tailings disposal circuit
- 1 x 4 Mtpa combined reagent building
- 1 x 4 Mtpa combined utilities

Two separate 2 Mtpa milling and flotation circuits were selected based on the concentrator production ramp-up profile. This modular approach allows for increased processing flexibility and plant redundancy while also allowing for phasing of capital expenditure.

This Section outlines the concentrator process design basis as well as the engineering design.

17.3 Concentrator Process Design

The Platreef concentrator design is based on a conventional three-stage crushing and ball milling circuit followed by flotation. This single-stage milling and flotation (MF1) process flowsheet is well known in industry, and has historically been proven as a suitable processing route for various platinum ore types containing significant concentrations of base metal sulphides. The key aspects of the concentrator plant design are discussed below.

Table 17.1 Summary of Concentrator Design Criteria

Criteria	Units	Nominal	Design
Production Summary			
Mining			
Annual Plant Feed Rate Year 1	Mtpa	–	1.37
Annual Plant Feed Rate Year 2	Mtpa	–	3.53
Annual Plant Feed Rate Year 3	Mtpa	–	3.89
Annual Plant Feed Rate Year 4+	Mtpa	–	3.99
Life of Mine	Years	–	31
Plant Throughput			
Design Throughput Phase 1A	Mtpa (dry)	2.00	2.22
Design Throughput Phase 1B	Mtpa (dry)	4.00	4.44
Mass Pull	%	4.40	5.10
Head Grades			
Platinum	g/t	1.95 - 2.19	2.30
Palladium	g/t	2.01 - 2.34	2.48
Rhodium	g/t	0.14 - 0.16	0.17
Gold	g/t	0.30 - 0.31	0.32
3PE+Au	g/t	4.40 – 5.00	5.27
Copper	%	0.17	0.17
Nickel	%	0.34 – 0.35	0.35
Overall Recovery			
3PE+Au recovery	%	–	84 - 90
Nickel recovery	%	–	69 - 75
Copper recovery	%	–	86 - 90
Concentrate Grades			
3PE+Au grade target	g/t	–	85
Nickel grade target	%	–	4.1 – 6.2
Copper grade target	%	–	2.5 – 4.0
Crushing Operating Schedule			
Operating days per annum	days	365	365
Operating shifts per day	number	2	2
Hours per shift	h	12	12
Availability	%	67	74
Utilisation	%	94	94

Criteria	Units	Nominal	Design
Crushing Circuit Running Time	%	63	70
Overall Running Time	h/annum	5517	6115
Design Circuit Feed Rate	t/h (dry)	725	725
Milling and Flotation Operating Schedule			
Operating Days per Annum	days	–	365
Operating Shifts per Day	number	–	2
Hours per Shift	h	–	12
Availability	%	–	94
Utilisation	%	–	98
Milling Circuit Running Time	%	–	92
Overall Running Time	h/annum	–	8061
Circuit Feed Rate	t/h/module (dry)	248	275
Milling & flotation modules phase 1A	number	1 x 2 Mtpa	
Milling & flotation modules phase 1B	number	Phase 1A + 2 Mtpa	

17.3.1 ROM Handling

The concentrator plant crushing circuit receives ROM from underground mining at a top size of 270 mm based on the underground crusher expected particle size.

A 4 Mtpa, conventional three-stage crushing and screening circuit was selected, operating in closed circuit to produce a mill feed size of approximately 13 mm. The first stage of crushing (primary crushing) is included within the underground mining scope, while the concentrator design includes an additional two stages of crushing (secondary and tertiary crushing) on surface.

17.3.2 Milling

Two separate 2 Mtpa milling circuits were selected based on the updated plant ramp-up and comminution trade-off and concentrator production ramp-up profile. Each milling circuit is comprised of a single 2 Mtpa ball mill and cyclone cluster operated in closed circuit. Each milling circuit is fed with 13 mm tertiary crushed material and will produce a 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 ball mill size for each stream is 22½'ftØ x 34½'ft Effective Grinding Length (EGL) with a grate discharge liner arrangement and 12 MW (2 x 6 MW) variable speed geared pinion drives. The design has been based on a maximum ball charge of 35%. The ball charge, together with the variable speed motor capability, allows feed material variability, required grind size and mill throughput. A summary of the expected ball mill performance when treating ore of the 85th percentile hardness, is presented in Table 17.2.

Table 17.2 Summary of Ball Mill Performance Criteria

Throughput Parameter	Treatment Rate per Mill (dry tph)	Ball Charge	Expected Power Draw (kW)
Nominal	248	33%	10 445
Maximum	275	35%	10 713

Test work conducted as part of the Platreef 2017 FS indicated that the flotation circuit response was sensitive to the grinding media type used. It was noted that the flotation performance improved when using high chrome media (Cr>16%) as compared to carbon steel media. Based on the test work 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

Two separate 2 Mtpa flotation circuits were selected based on the production ramp-up and comminution trade-off studies conducted.

The optimised flotation circuit flowsheet as developed during testing was used as the basis of design. This optimised flowsheet 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 Flowsheet as Derived from Test Work

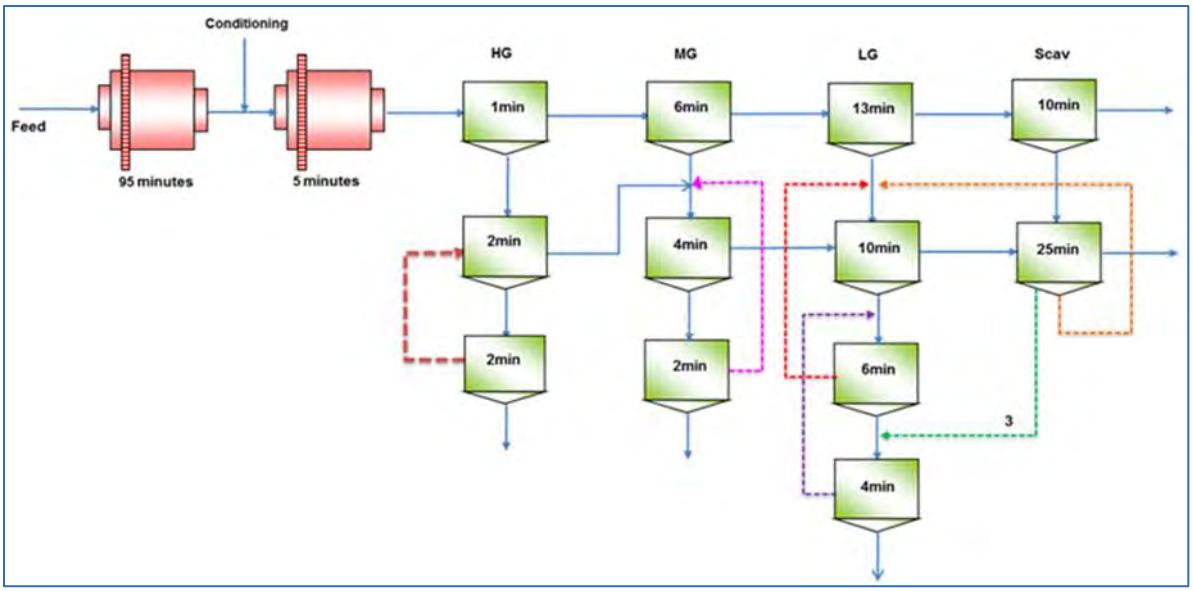


Figure by DRA, 2017

The flotation testing conducted at three independent laboratories during the Platreef 2015 PFS campaign highlighted that the flotation feed Eh is negative when the sample is milled using mild steel grinding media. A positive Eh prior to flotation was achieved by using high chrome grinding media. The milling circuit design is based on high chrome grinding media and mill liners. All mild steel piping within the slurry plant is specified to be epoxy lined to minimise any slurry contact.

17.3.4 Concentrate Thickening and Filtration

The concentrate thickening and filtration circuit design is based on the installation of a single concentrate thickener and horizontal plate filter to treat the combined concentrate from both the 2 Mtpa flotation modules.

The concentrate thickener and the concentrate filter is sized for a maximum tonnage of 25.3 dry tph. At the time of conducting the Platreef 2017 FS, concentrate thickening test work and concentrate filtration test work had not been conducted, due to the amount of concentrate sample required for definitive test work. For the design industry benchmark thickening and filtration fluxes have been used to size the thickener and the concentrate filter. These rates are based on operational data for similar operations treating Platreef and Great Dyke ore types at similar grind sizes. In the absence of concentrate thickening test work, it has been assumed that an underflow solids concentration of 55%(w/w) can be achieved.

Concentrate thickening and filtration test work will be conducted during the mini pilot plant test work campaign, which will take place prior to detailed design.

17.3.5 Tailings Dewatering and Transfer

The tailings dewatering and disposal circuit design is based on the installation of a single tailings thickener and clarifier, multiple stage tailings disposal pump trains and booster station to treat the combined tailings from both the 2 Mtpa flotation modules. The tailings dewatering and disposal circuit includes conventional thickening and clarification of thickener overflow. There is also allowance for brine disposal to the TSF via the tailings disposal tank. The target pumped tailings slurry density is 45% (w/w) solids when operating a single 2 Mtpa module and 50-55% (w/w) solids when operating both modules at 4 Mtpa.

A backfill plant is included in the mine design. The backfill plant will operate intermittently depending on mining backfill requirements. The design of the backfill plant is based on treating 57-72% of the tailings from both 2 Mtpa milling and flotation modules over LOM. The backfill plant will receive un-thickened concentrator plant flotation tailings. During operation of the backfill plant, the concentrator plant will not pump tailings to the tailings facility, however, when the backfill plant is offline, the concentrator plant will need to process tailings at 100% of mill capacity. Thus, the tailings thickening and disposal system has a maximum capacity of 275 dry tph. The intermittent use of the tailings disposal system incorporates the use of a flushing system to the TSF when the switchover to backfill occurs. This is to minimise settling of solids in the pipeline due to intermittent interruptions.

The process water from the backfill plant will be returned to the process plant tailings thickener to be used as process water within the concentrator plant.

The tailings from each 2 Mtpa module reports to the tailings thickener via a guard cyclone. A 35 mØ tailings thickener has been selected to treat guard cyclone overflow, with a target underflow solids concentration of 57-60% (w/w). The design also includes a 35 mØ clarifier to remove any solids from the thickener overflow. The achievable thickener underflow density requires confirmation based on thickening test work for a classified feed. This confirmatory thickening test work will be conducted during the mini pilot plant test work campaign, planned ahead of detailed design.

Three tailings lines will be installed to provide operational flexibility, one line to cater for the 2 Mtpa throughput and two lines (duty/standby) to cater for the 4 Mtpa throughput. The tailings line is approximately 8 km long, and there is an elevation difference of 126 m between the plant and the TSF. The tailings pumping system design requires a booster station pumping system located at a distance of approximately 5 km from the concentrator plant.

17.4 Concentrator Plant Production Schedule

The concentrator plant production schedules showing feed tonnes and grade, recoveries and metal production is presented in Table 17.3 and Table 17.4.

Table 17.3 Process Production Plant Feed

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrator Feed	kt	124,681	–	–	–	–	–	1,367	3,540	3,867	3,987	3,971	19,890	39,751	48,307
NSR BDT20	$/t	158	–	–	–	–	–	186	185	180	170	166	167	157	148
Platinum	g/t	1.95	–	–	–	–	–	2.46	2.34	2.22	2.07	2.05	2.10	1.92	1.84
Palladium	g/t	2.01	–	–	–	–	–	2.60	2.53	2.39	2.23	2.14	2.17	1.99	1.84
Gold	g/t	0.30	–	–	–	–	–	0.31	0.31	0.32	0.31	0.31	0.30	0.30	0.30
Rhodium	g/t	0.14	–	–	–	–	–	0.18	0.17	0.16	0.15	0.14	0.15	0.13	0.12
Copper	% Cu	0.17	–	–	–	–	–	0.16	0.17	0.18	0.18	0.17	0.16	0.17	0.16
Nickel	% Ni	0.34	–	–	–	–	–	0.32	0.35	0.37	0.36	0.35	0.34	0.34	0.33
Sulphur	% S	0.83	–	–	–	–	–	0.74	0.84	0.92	0.89	0.87	0.82	0.83	0.81
3PE + Au	g/t	4.40	–	–	–	–	–	5.55	5.35	5.08	4.76	4.64	4.73	4.34	4.10

Note NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.

Table 17.4 Concentrate and Metal Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrator Recoveries	–														
Platinum	%	87.37	–	–	–	–	–	89.46	90.69	89.85	88.67	88.21	88.56	87.11	86.20
Palladium	%	86.93	–	–	–	–	–	89.24	90.47	89.64	88.42	87.92	88.29	86.66	85.55
Gold	%	78.63	–	–	–	–	–	79.49	80.70	80.23	79.53	79.25	79.46	78.53	77.93
Rhodium	%	80.47	–	–	–	–	–	83.46	84.67	83.68	82.20	81.60	82.04	80.14	78.84
Copper	%	87.88	–	–	–	–	–	85.68	88.38	89.63	89.05	88.24	87.50	88.23	87.46
Nickel	%	71.88	–	–	–	–	–	69.09	72.67	74.69	74.07	72.85	72.01	72.46	70.80
Sulphur	%	70.99	–	–	–	–	–	67.38	71.38	73.52	72.65	72.18	70.75	71.20	70.49
3PE + Au	%	86.36	–	–	–	–	–	88.61	89.81	88.95	87.75	87.28	87.65	86.10	85.07
Mass Pull	%	–	–	–	–	–	–	5.49	5.64	5.32	4.92	4.76	4.88	4.40	4.11
Concentrate Produced	kt	5,568	–	–	–	–	–	75	200	206	196	189	970	1,748	1,984
Concentrator Grade	–														
Platinum	g/t	38.24	–	–	–	–	–	40.10	37.62	37.41	37.40	37.99	38.21	38.13	38.54
Palladium	g/t	39.07	–	–	–	–	–	42.30	40.56	40.23	40.07	39.45	39.35	39.16	38.34
Gold	g/t	5.32	–	–	–	–	–	4.43	4.48	4.85	5.07	5.11	4.93	5.32	5.73
Rhodium	g/t	2.43	–	–	–	–	–	2.70	2.57	2.52	2.46	2.45	2.51	2.40	2.39
Copper	% Cu	3.27	–	–	–	–	–	2.44	2.68	3.07	3.20	3.14	2.93	3.40	3.47
Nickel	% Ni	5.46	–	–	–	–	–	4.07	4.46	5.13	5.42	5.33	5.03	5.68	5.68
Sulphur	% S	13.12	–	–	–	–	–	9.05	10.62	12.71	13.11	13.17	11.85	13.48	13.87
3PE + Au	g/t	85.07	–	–	–	–	–	89.53	85.23	85.00	85.00	85.00	85.00	85.00	85.00
Recovered Metal	–														
Platinum	koz	6,846	–	–	–	–	–	97	242	247	236	231	1,192	2,143	2,458
Palladium	koz	6,994	–	–	–	–	–	102	260	266	253	240	1,227	2,201	2,445
Gold	koz	952	–	–	–	–	–	11	29	32	32	31	154	299	365
Rhodium	koz	436	–	–	–	–	–	7	17	17	16	15	78	135	153
Copper	klb	401,728	–	–	–	–	–	4,045	11,779	13,923	13,841	13,095	62,585	130,899	151,561
Nickel	klb	669,993	–	–	–	–	–	6,733	19,656	23,273	23,425	22,212	107,512	218,807	248,375
Sulphur	klb	1,610,633	–	–	–	–	–	14,976	46,770	57,630	56,650	54,933	253,490	519,564	606,621
3PE + Au	koz	15,228	–	–	–	–	–	216	547	562	536	517	2,651	4,777	5,421

17.5 Process Description

The process descriptions are based on the detailed process flow diagrams (PFDs) that include mass balances. A summary block flow diagram for the concentrator plant is presented in Figure 17.1.

17.5.1 ROM Handling

ROM is crushed underground to a top size (F100) of 270 mm. The pre-production ROM material is conveyed by a ROM handling system and can be routed to either one of the two 5,200 t ROM silos or the ROM stockpile.

Upon commencement of the concentrator plant operations, this stockpiled material will be reclaimed via a ROM reclaim system consisting of a static grizzly, surge bin, vibrating screen and reclaim conveyor fitted with a magnet for tramp metal removal. The reclaim conveyor feeds the secondary screens.

Once the concentrator plant is in production, ROM ore is conveyed from Shaft 2 headgear, at a peak flow rate of 1,350 dry tph and a top size of 270 mm, into either one of the two 5,200 t ROM silos. Each ROM silo is equipped with apron feeders for extraction onto the ROM Screen Feed Conveyor that feeds the secondary screens in the crushing circuit. Tramp metal is removed prior to crushing by means of a tramp metal magnet situated on the conveyor.

17.5.2 Crushing and Screening

The concentrator crushing and screening circuit consists of secondary and tertiary crushing stages in closed circuit with secondary and tertiary screening stages.

Ore from the ROM Screen Feed Conveyor, ROM Reclaim Conveyor, and Secondary Crusher Product Conveyor are transferred to the Secondary Screen Feed Bin. Ore from the Secondary Screen Feed Bin is transferred via vibratory feeders to the secondary screening circuit which is comprised of two double-deck screens. The top deck (+30 mm) of both the secondary screens reports to the Secondary Cone Crusher Feed Conveyor which is equipped with a magnet for the removal of tramp metal. The bottom screen deck (+12 mm-30 mm) material of both the screens reports to the Tertiary Cone Crusher Feed Conveyor, which is also equipped with a magnet for the removal of tramp metal. The secondary screen undersize (-12 mm) from both secondary screens reports to the Screen Product Conveyor.

The Secondary Cone Crusher circuit is comprised of two cone crushers operated in parallel. Each cone crusher is fed via a feed bin and vibrating feeder. The secondary crusher product (-35 mm) is removed by the Secondary Cone Crusher Product Conveyor and fed back to the Secondary Screen Feed Bin for re-screening.

The Tertiary Cone Crusher circuit is comprised of two cone crushers operated in parallel. Each cone crusher is fed via a feed bin and belt feeder.

The tertiary crushed material (-10 mm) is removed by the Tertiary Cone Crusher Product Conveyor and fed to the Tertiary Screening Circuit, which is comprised of two single-deck screens (+10 mm). Oversize reports to the tertiary cone crushing area while undersize reports to the Screen Product Conveyor.

The Screen Product Conveyor transfers the final product material (P100 =13 mm) to either of the two 8,000 t mill feed silos. A diverter chute on top of the first silo diverts the material to Mill Silo No.1 or Mill Silo 2 via an additional feed conveyor.

17.5.3 Milling

The crushing and screening circuit product (P100 =13 mm) is stored in two off 8,000 t silos. Each silo is dedicated to a 2 Mtpa milling module. The crushed ore is transferred via belt feeders onto the associated Mill Feed Conveyor. Belt feeders are specified to minimise spillage and dust of the relatively fine material.

Each milling circuit consists of a single 22½'ftØ x 34½'ft EGL ball mill, with a grate discharge liner arrangement and 12 MW (2 x 6 MW) variable speed geared pinion drives. Each ball mill operates in closed circuit with a classification cyclone cluster. The mill feed material (F100 of 13 mm) is fed to the mill feed hopper where process water is added for in-mill density control. Copper collector is also added into the mill feed hopper to assist in maximising copper recovery in the high-grade flotation circuit.

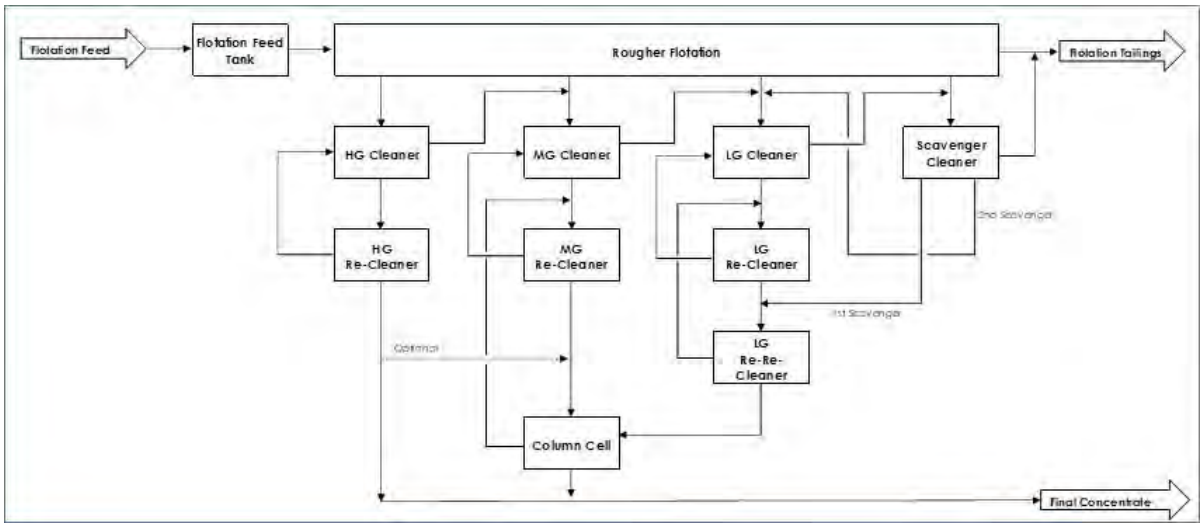
The milled material discharges onto the vibrating mill discharge screen for scats removal. Mill scats are deposited onto a scats stockpile via a scats removal conveyor. Scats from the stockpile are removed via front end loader and taken to the waste handling area, or alternatively scats can be re-loaded onto the mill feed conveyor.

The screened material is collected in the mill discharge sump and pumped to the mill classification cyclone cluster, which produces an overflow product of P80 = 75 µm. The cyclone underflow is recycled to the mill feed hopper for further regrinding. The cyclone overflow gravitates to the Rougher Flotation Feed Tank via a linear trash screen and two-stage sampling system. The oversize from the linear screen is removed as trash.

17.5.4 Flotation

The flotation circuit consists of two identical modules, each capable of treating 2 Mtpa. A single 2 Mtpa flotation module is described below and a high level summary flowsheet is presented in Figure 17.3.

Figure 17.3 Flotation Circuit High Level Flowsheet



The rougher flotation feed, at approximately 28% solids (w/w), is pumped from the Rougher Flotation Feed Tank to the Rougher Flotation Circuit. The rougher flotation bank consists of 1x 30 m³ and 8 x 130 m³, forced air, tank cells with total a residence time of 75 minutes.

Four rougher concentrates will be produced namely high grade (HG), medium grade (MG), low grade (LG) and scavenger concentrate that report to the HG, MG, LG and Scavenger cleaners respectively.

The rougher flotation tailings gravitates to a tailings sump via a two-stage sampling system. The rougher tailings, together with the scavenger cleaner tailings, is pumped to the final tailings thickening and disposal circuit.

Collector, depressant and promotor are added into the MG and LG Rougher feed box, and coagulant is added to the rougher tailings tank to assist with coagulation in the tailings thickener.

HG rougher concentrate is pumped to the HG Cleaner flotation circuit. Depressant and frother are added to the HG Cleaner feed box. The high-grade cleaners consists of one bank of 2 x 3m³, forced air, trough cells. The HG Cleaner concentrate is collected and pumped to the HG Re-Cleaner flotation circuit. The tailings of the HG Cleaner gravitates to the HG Cleaner Tailings Sump, from where it is pumped to the MG Cleaner feed box.

Pumped HG Cleaner concentrate feeds the HG Re-Cleaners consisting of one bank of 4 x 0.5 m³, forced air, trough cells. Depressant and frother are added to the HG Re-Cleaner feed box. Concentrate from these cells is pumped to a vezin sampler at the concentrate handling section. An option also exists to pump the HG Re-Cleaner concentrate to the column flotation cell. The tailings of the HG Re-Cleaner gravitates to the HG Cleaner feed.

MG Rougher Concentrate and HG Cleaner tailings are pumped to the MG Cleaner feed box where depressant, collector and frother are added. The medium-grade cleaners consists of 3 x 10 m^3, forced air, tank cells. Concentrate from these cells is collected and pumped to the MG Re-Cleaner flotation circuit. The tailings of the MG Cleaner is pumped to the LG Cleaner circuit.

The pumped concentrate from the MG Cleaners, along with added depressant and frother, is fed to the MG Re-Cleaners. The MG Re-Cleaners consists of one bank of 4 x 3 m^3, forced air, trough cells. Concentrate from these cells is collected and pumped to the Cleaner Column flotation circuit. The combined MG Re-Cleaner concentrate, LG Re-Re-Cleaner concentrate and HG Re-Cleaner (optional) concentrate combine as feed into the column flotation cell. Concentrate produced from the cleaner column flotation cell is pumped to a dedicated sampler at the concentrate handling area. The tailings of the cleaner column flotation cell is pumped to either the MG Cleaner feed box or alternately the MG Re-Cleaner feed box.

LG Rougher Concentrate, MG Cleaner Tailings and Scavenger Cleaner Concentrate are pumped to the LG Cleaner feed box, where depressant, collector and frother are added. The LG Cleaners consists of one bank of 5 x 50 m^3, forced air, tank cells. Concentrate from these cells is pumped to the LG Re-Cleaner flotation circuit. The LG Cleaner tailings is pumped to the Scavenger Cleaner feed box.

LG cleaner concentrate, with added depressant and frother, is pumped to the LG Re-Cleaner feed box. The LG Re-Cleaners consists of one bank of 6 x 10 m^3, forced air, tank cells. Concentrate from these cells is pumped to the LG Re-Re-Cleaner feed box. The LG Re-Cleaner tailings gravitates to the LG Cleaner circuit. The LG Re-Cleaner tailings is pumped to the LG Cleaner feed box.

LG Re-Cleaner concentrate and the first Scavenger Cleaner concentrate is pumped to the LG Re-Re-Cleaner feed box, where frother is added. The LG Re-Re-Cleaners consist of one bank of 6 x 3 m^3, forced air, trough cells. Concentrate from these cells is pumped to the cleaner column flotation cell and the LG Re-Re-Cleaner tailings gravitates to the LG Re-Cleaner feed.

Scavenger Rougher concentrate and LG Cleaner tailings are pumped to the Scavenger Cleaners feed box. Depressant, collector and frother are added. The Scavenger cleaners consists of 6 x 70 m^3, forced air, tank cells. The first concentrate from the Scavenger Cleaner cells feed the LG Re-Re-Cleaner circuit, and the rest of the concentrate is fed to the LG Cleaner flotation circuit. The tailings of the Scavenger Cleaner feeds the Scavenger Cleaner Tailings Sampler, located at the rougher tailings sump area.

The flotation circuit design includes a FloatStar software system to ensure optimal level and mass pull control in the flotation circuit. In addition, online analysers on the final concentrate and flotation column concentrate streams will allow for real-time analysis of the concentrate grade.

17.5.5 Concentrate Handling and Filtration

The HG Re-Cleaner concentrate and Column cell concentrates from each 2 Mtpa flotation module feed dedicated vezin samplers ahead of a vibrating trash screen. The screened concentrate gravitates to a 25 m diameter concentrate thickener. Flocculant is added to the thickener feed. The thickened concentrate at 55% 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 thickener is utilised as spray water in the flotation circuit.

Slurry from the concentrate storage tanks is fed to the horizontal plate pressure filter. The filter cake, with a moisture content of 12-15%, discharges onto a transfer conveyor which feeds a reversible shuttle conveyor. The concentrate cake is conveyed into concrete storage bunkers. The filter filtrate reports to the concentrate thickener. The concentrate cake product is loaded onto trucks and sampled by an auger sampler before dispatch.

17.5.6 Tailings Dewatering and Transfer

The combined tailings from each 2 Mtpa flotation module is pumped to a guard cyclone. Guard cyclone overflow gravitates to the 35 m diameter tailings thickener, where flocculant is added in the feedwell. The thickened underflow and guard cyclone underflow is combined in the tailings disposal tank. Tailings is pumped to the TSF by means of a duty/standby pumping installation. The overflow product from the tailings thickener is utilised as process water for the concentrator.

Three tailings lines will be installed; a single 2Mtpa line and two lines (duty/standby) for the 4 Mtpa capacity. The tailings pumped to the TSF will be at 45% solids (w/w) for the 2 Mtpa scenario and approximately 50-55% solids (w/w) for the 4 Mtpa scenario. The tailings pumping system design includes a booster station with duty/ standby pump train installation at a distance of 5 km from the concentrator plant. The tailings disposal system design is based on pumping100% of the concentrator plant tailings to cater for the maximum expected slurry volumes when backfill is not in operation.

A backfill plant is included in the mine design. The design of the backfill plant is based on treating 57-72% of the combined flotation tailings produced from the 4 Mtpa concentrator over LOM. The backfill plant will operate intermittently, dependent on mining backfill requirements. The backfill plant will receive un-thickened flotation tailings. During operation of the backfill plant the concentrator plant will not pump to the TSF. The process water from the backfill plant will return to the concentrator plant tailings thickener to be used as process water.

17.5.7 Water Circuit

Process water is stored in a 1,000 m^3 process water tank. The process water tank is fed with tailings thickener overflow water, excess flotation spray water and TSF return water. Each milling-flotation module is equipped with a dedicated process water pump installation. Clean water from the 5,000 m^3 clean-water tank provides for the process water requirements. Excess process water will be pumped to the mine return water dam.

Clean water from the water treatment plant is distributed throughout the mine site. The 5,000 m^3 clean water tank, located in the concentrator plant area, supplies the dust suppression, gland service water, process water make-up, reagent water, spray water make-up and backfill clean water make-up systems. A dedicated 1,000 m^3 fire water tank, also located in the concentrator plant area, supplies fire water for firefighting purposes. For rapid concentrator plant start up and shutdown, clean water can be gravitated through the clean water tank overflow to the process water tank.

The water treatment plant produces potable water which is distributed throughout the mine site.

17.5.8 Air Services

Low pressure blower air for the flotation circuits is supplied by positive displacement blowers. The rougher flotation circuit and scavenger cleaner flotation circuit in each module is serviced by a dedicated blower bank, each consisting of four blowers. The cleaner flotation circuit of each 2 Mtpa module is serviced by a dedicated blower bank, each consisting of three blowers.

Plant and instrument air is supplied by compressors, delivering compressed air at 1,300 kPa(g) which is then pressure reduced to 800 kPa(g). Instrument air passes through an air filtration and drying system. The remainder of the air is used as compressed air. Dedicated air receivers, prior to each flotation section, provide instrument air buffering for valve operations. Further air receivers are placed throughout the plant close to large consumers such as the flotation column cell.

The drying air for the concentrate filter press is drawn from dedicated compressors and air receivers. Filter pressing air is supplied by two high pressure compressors delivering compressed air at approximately 1,600 kPa(g). The compressors and air receivers are situated adjacent to the filtration building.

17.5.9 Reagents

Two flocculant systems allow for anionic and cationic flocculant make-up. The processes for both plants are as follows:

- Flocculant granules are delivered in 1,000 kg bags and are manually loaded into a bulk-bag bin receiver. The flocculant granules, together with reagent water, are transferred to a wetting and mixing system. The flocculant is diluted to 0.5% w/v strength in the flocculant dosing tanks. Further in-line dilution occurs while being pumped to the respective dosing points.
- Depressant granules are delivered in 1,000 kg bags and loaded into a depressant storage hopper. The depressant granules together with reagent water are transferred to a wetting and mixing system. The depressant is diluted to 1.0 % w/v strength in depressant dosing tanks prior to being pumped to the respective dosing points.

- Collector (SIPX) pellets will be received in bulk bags and added manually into the collector mixing tank. Reagent water will be added into the agitated collector mixing tank. After mixing, the collector solution is transferred to the collector dosing tank, after which it is pumped to the respective dosing points.
- Copper collector will be received in 1 m^3 intermediate bulk containers (IBC), added manually into the copper collector dosing tank, and pumped to the respective dosing points.
- Promotor will be received in 1 m^3 IBCs, added manually into the Promotor dosing tank and pumped to the respective dosing points.
- Frother will be received in 1 m^3 IBCs, added manually into the frother dosing tank and pumped to the respective dosing points.
- Coagulant will be received in 1 m^3 IBCs, added manually into the Coagulant dosing tank and pumped to the respective dosing points.

17.5.10 Control Philosophy

High-level concentrator plant control philosophy and piping and instrumentation diagrams were prepared.

17.6 Concentrator Engineering

This Section describes the engineering process and factors that influence and shape the final concentrator plant layout and physical designs. It further provides relevant technical information and descriptions of main equipment associated with the concentrator plant.

17.6.1 Basis of Engineering

The basis of engineering provides the legal and technical framework on which the concentrator design is founded. It consists of three main pillars:

- Legislation and Standards,
- Environmental Management Plan, and
- 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.

Major noise-generating sources were identified and collectively modelled to arrive at mine boundary noise levels. The rock handling area, consisting of conveyors, crushing and screening areas were identified as major noise and dust contributors. Surface vent fan noise was further identified. A specialist noise consultant was contracted to simulate the crushing and screening building noise generators and to provide an effective noise-reducing cladding design for the buildings. The noise consultant further modelled vent fan noise propagation with and without the perimeter berm wall.

Dust extraction and suppression has been considered across the stockpiles, rock handling and processing systems. Two well-established South African dust control equipment suppliers were approached to review the layout designs and to provide best practice dust suppression and extraction designs. The crushing building, screening building and all silos are designed with independent dust extraction systems. Conveyor discharge chutes, transfer towers and tipping areas all include dry fog dust suppression systems.

The temporary day stockpiles will be sheeted with wind protection netting.

Haulage roads will be treated with dust suppression chemicals, 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 test work programme has yielded sufficient information to develop a definitive metallurgical flow sheet. Further mini-plant test work will be conducted prior to detailed design to validate the process design and to provide additional design parameters. The Platreef 2017 FS design allows for process and ore variability through conservative design and introduction of various concentrate diverting options.

The engineering design principles are aligned with the process design. The engineering design has considered the required international and national laws and standards. Design criteria and detailed specifications ensure conformity of design and safety across the mine site. The concentrator design is based on previously constructed and proven process modules designed by DRA, taking lessons learnt and experience into account.

The engineering design has taken cognisance of the required environmental and social impact of the concentrator in terms of noise, dust, light and visual pollution requirements stipulated by various national and international bodies. Expert consultants were contracted to incorporate best practice design and equipment into relevant sections of the plant to minimise the overall environmental and social impact.

The concentrator plant has been designed in accordance to the required level of accuracy for a feasibility study whilst adhering to social and environmental responsibilities. The modular approach toward the milling and flotation plants allows for redundancy and phasing of capital spend.

18 PROJECT INFRASTUCTURE

18.1 Summary

The 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 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. The land within the mine area is mainly used for agricultural activities and livestock.

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, a process plant, a tailings facility and various general infrastructure sections, were done with these areas in relation to each other after which these areas were looked at individually.

18.1.1 Power

A bottom up estimating methodology was used to arrive at a predicted electrical consumption and notified maximum demand (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 were used to arrive at a total connected load. The connected loads are reduced to connected running loads by excluding standby circuits and further reduced to absorbed power running loads as determined by the process and mechanical data.

The bulk power supply is to be sourced from Eskom, the South African national power utility. The initial application for 70 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 Impact Assessment (EIA), together with the land and rights package for the works, has been completed and received as part of the Eskom detail design package responsibility.

The Project surface infrastructure is further broken down into three distinct areas:

- Mining surface infrastructure – all supporting infrastructure located within the shaft bank area, i.e. roads, buildings, dams, services, bulk earthworks and electrical reticulation, etc.
- Process plant infrastructure – all supporting infrastructure located within the process plant area, i.e. roads, buildings, dams, services, bulk earthworks and electrical reticulation, etc.

- General site infrastructure – bulk supply infrastructure found within the waste management area, the general office area and all supporting infrastructure which links the three areas together. Additionally, the bulk supply of water and power are included under this area.

To mitigate the noise, dust and the visual impact of the mine, a 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.

The main access road passing the site is the N11. As this is a national road, it falls under the jurisdiction of the South African National Roads Agency (SANRAL). With regards to off-ramps and exits from the N11, SANRAL has approved one intersection.

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.

18.1.2 Water

The mine area is situated in a former riverbed, and it is exposed to large amounts of run-off rain water. 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 rain water flows in a sheet flow fashion thus being of no consequence. However, once the mine infrastructure is constructed, concentrated flow of rain water 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-of Drains
- Stormwater Run-off Drains
- Discharge Drain

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
- Ore
- Waste rock uncrushed and crushed
- Subsoils (soft and hard)
- Temporarily imported G4 material for platform construction

The buildings have been classified into the following categories:

- Architectural Buildings
- Workshops and Stores
- Change House Buildings
- Electrical Buildings

Certain identified architectural buildings will be designed and registered as Green Star Buildings.

The electrical design is based on the equipment specifications and electrical design criteria developed for the project. The electrical equipment is designed or selected to:

- Comply with Eskom energy efficient requirements
- Provide for high plant availability
- Provide an effective, simple solution which is maintainable by the plant operating personnel
- Provide a safe working environment for personnel and equipment
- Ensure that effort has been made to optimise the efficient use of energy and to minimise any adverse effects on the environment.

18.2 Mine Locality

The 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 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 indicates the Platreef Project's location within South Africa, Figure 18.2 the location within the Bushveld Complex and Figure 18.3 the location within the local region.

Figure 18.1 Platreef Project Location Within South Africa

Figure by DRA, 2017.

Figure 18.2 Platreef Project Location Within Bushveld Complex

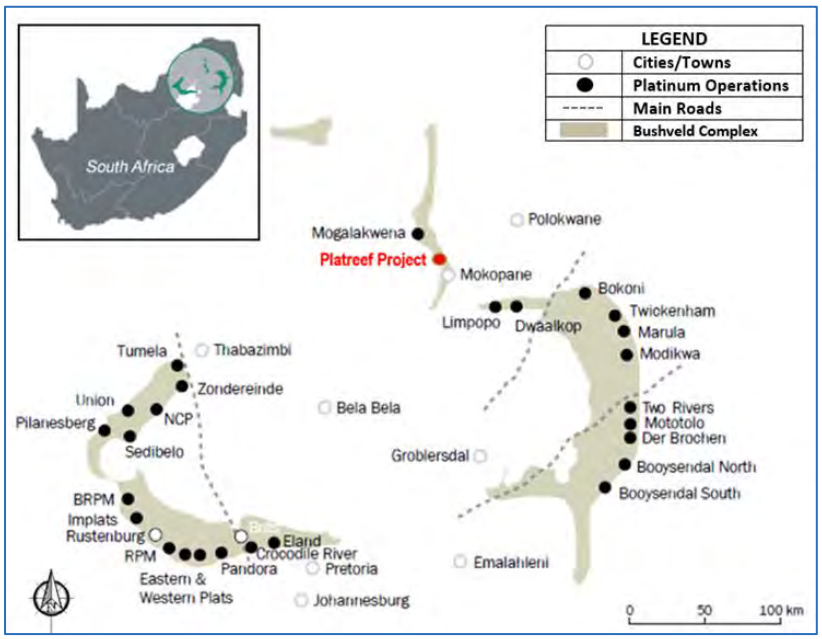


Figure by DRA, 2017.

Figure 18.3 Platreef Project Location Within Local Region

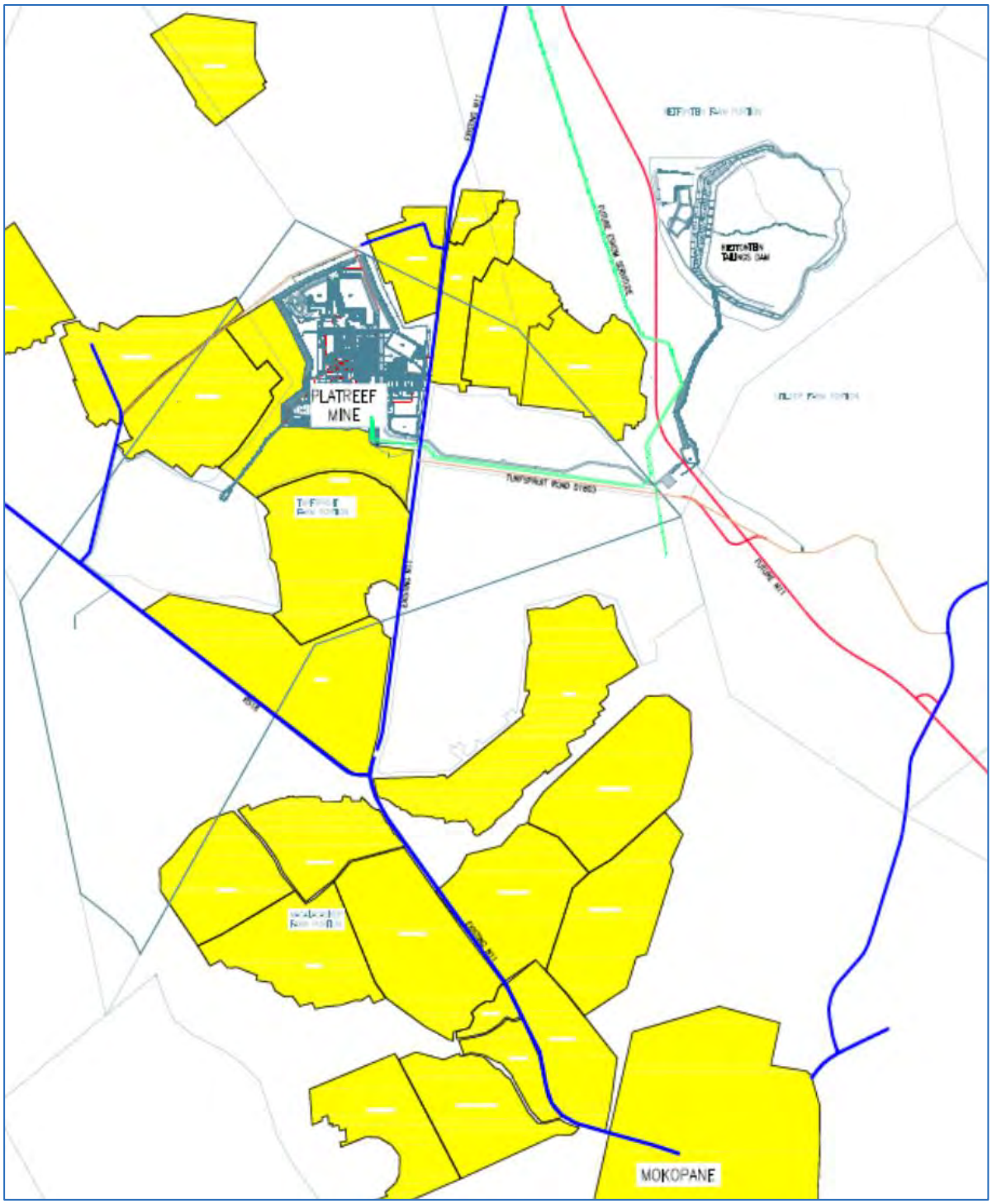


Figure by DRA, 2017

Figure 18.4 Platreef Project Lease Area

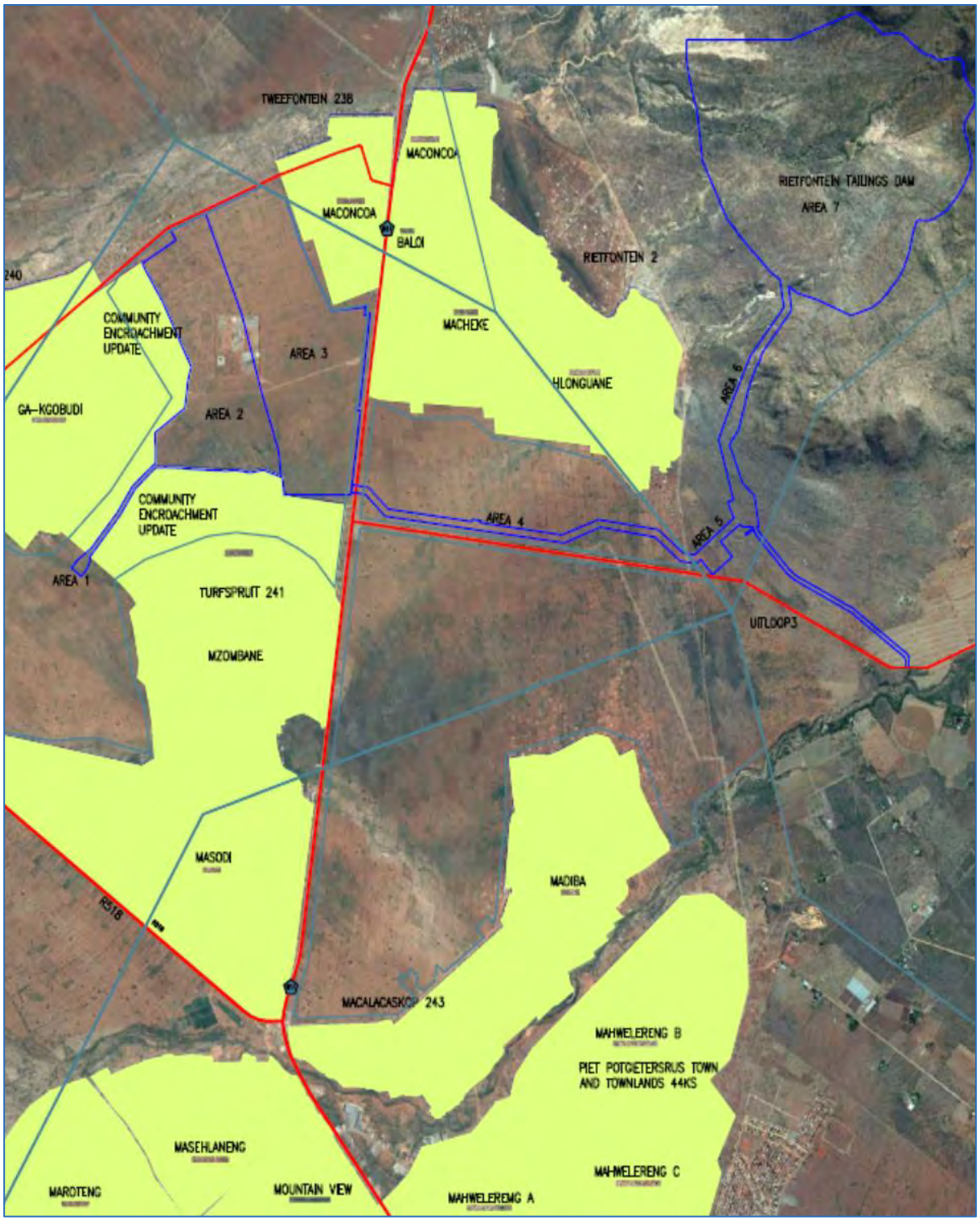


Figure by DRA, 2017.

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 18.4. 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° and 5°. Moderate slopes of between 6° and 15° occur in some areas. Isolated steeper slopes of between 16° and 21° occur along the banks of the Rooisloot and Klein-Sandsloot Rivers. The steepest slopes occur on the ridges and range between 22° and 69°.

The slope aspect/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 on the ridges.

The topographical model indicates that the elevation of the Project area increases from 1,030.5 mamsl in the Mogalakwena River floodplain in the south-western corner to 1,359 mamsl on the ridges in the north-eastern corner of the lease area.

18.4 Plot and Block Plans

18.4.1 Plot and Block Plan Development

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, a process plant, a tailings facility and various general infrastructure sections, were done with these areas in relation to each other, after which these areas were looked at individually. All layout work was done in close co-operation with the Platreef Project owner's team. The following major factors specifically influenced these plot and block plans:

- The requirement to allow footprint for potential expansion to an 8 Mtpa production mine.
- The hydrology requirements prohibiting infrastructure from being placed in the 1 in 100 year flood lines.
- The proximity of neighbouring communities and their potential encroachment onto mine lease areas.
- Various licence application and environmental and social requirements.

The rationale behind the plot and block plan development is as follows:

- The plot plan started off with the positioning of Shaft 1 and Shaft 2 and subsequent positioning of shaft-related infrastructure around the shafts, thus developing the shaft area block plan. Based on the surrounding communities, anticipated encroachment and environmental management programme (EMP) requirements, a suitably sized mining area was identified, and a mining area fence line was developed. This fence line set the perimeter and available area for the complete mining area plot plan development.

- Access to and from the N11 was identified, as well as bulk services to the mining area. Based on the topography, pollution and stormwater systems were developed. Based on the site selection process completed, waste facilities were located on the plot plan. Upon receipt of the mine production schedule, suitably sized areas were identified for waste and ore stockpiles. Subsequent site selection processes for waste rock dumps yielded the optimal position for these on the mine plot plan.
- The 4 Mtpa process plant was then positioned on the plot plan as well as future space for an 8 Mtpa upgrade was identified. With the final position of the tailings storage facility at Rietfontein identified, a suitable tailings pipeline corridor was pinpointed and placed onto the plot plans, and then routed to minimise grave relocation and heritage site impacts.

18.4.2 Plot Plans

The three plot plans for the Platreef Project are shown in Figure 18.5 and Figure 18.6.

Figure 18.5 Lease Area Plot Plan

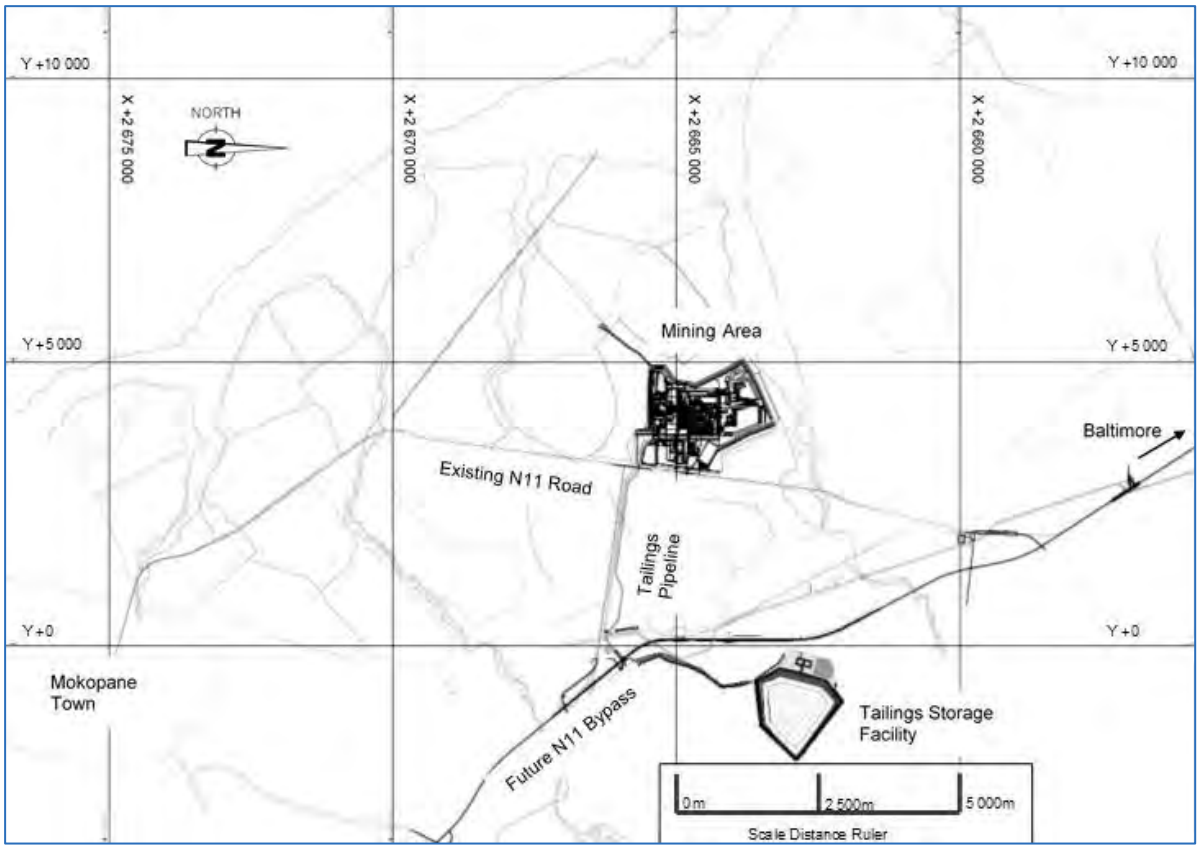


Figure by DRA, 2017.

Figure 18.6 Mine and Reitfontein TSF Area Plot Plan

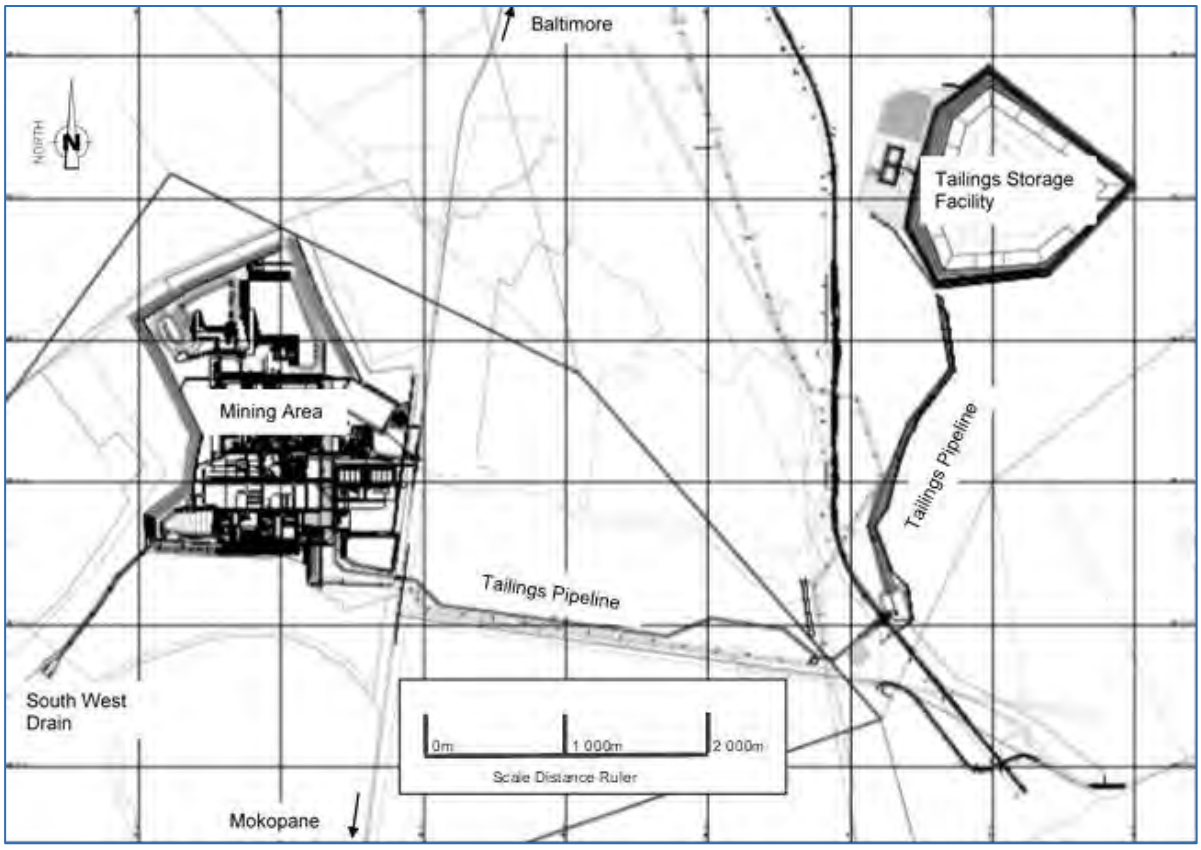


Figure by DRA, 2017.

The mining lease area plot plan outlines the mine surface infrastructure within the registered mine lease area. The Mine and Rietfontein tailings storage facility (TSF) area plot plan outlines the mine area, the TSF pipeline corridor and the TSF dam perimeter in relation with each other. The mine area plot plan outlines all the mining and process plant infrastructure within the fence line west of the N11.

The mine area plot plan is shown in Figure 18.7.

Figure 18.7 Mine Plot Plan

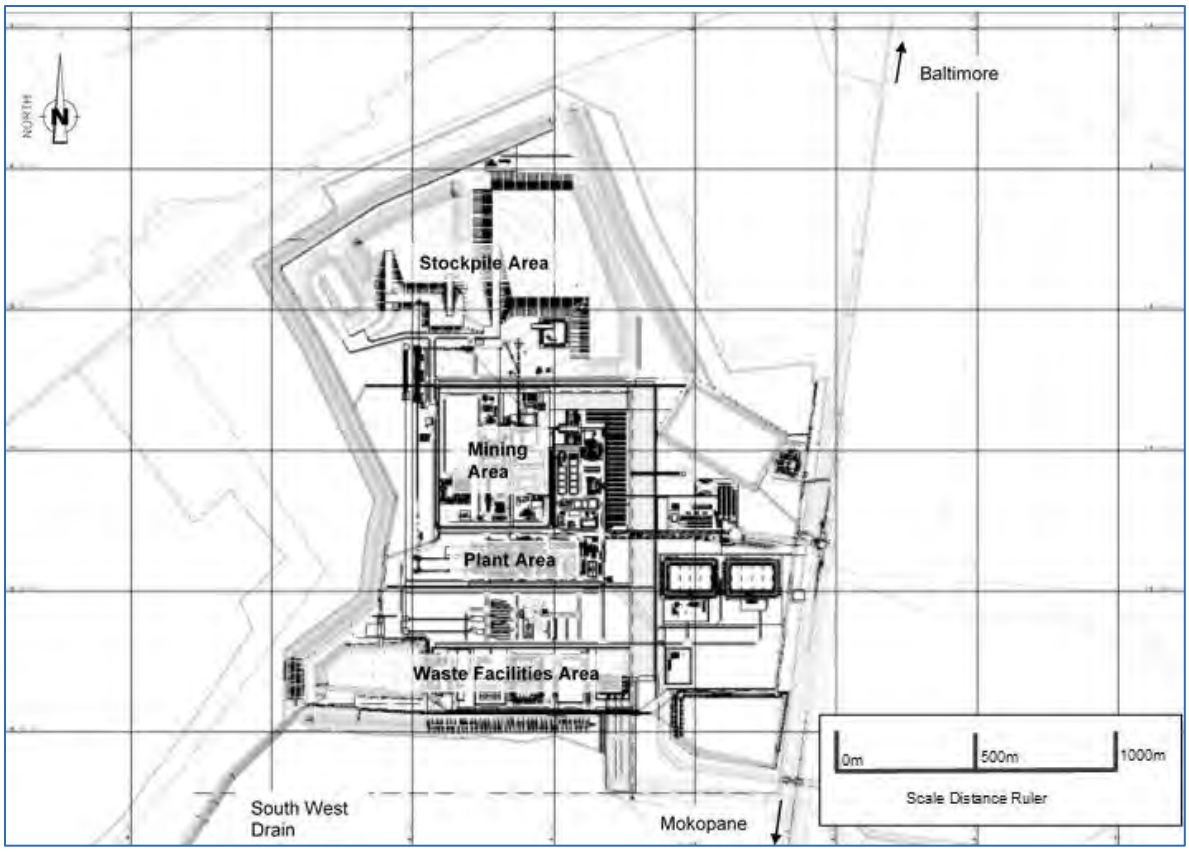


Figure by DRA, 2017.

An irregular-shaped fence line and perimeter berm indicates the available footprint to the west of the N11, which will locate the Platreef 4 Mtpa mine, 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. Treated municipal grey water will enter the mining area from the south-west corner. A 132 kV electrical substation is located near the southern boundary of the mining area between the waste facilities and Attenuation Pond 2. Two raw water dams are located in the eastern portion of the mine area, to hold a bulk water buffer.

A single access point is allowed from the N11, with an emergency second access/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 the accommodating of concentrate transportation on site.

In line with the Integrated Water Use Licence Application (IWULA) and site topography, storm and pollution water systems are suitably located and integrated in the mining area. These systems comprise large drains and channels (mostly running north to south), two stormwater attenuation ponds and a Stormwater Control Pond in the south-western corner of the mining area. Surface run-off water generated from the shaft and process plant areas is channelled to the Stormwater Control Pond.

In line with the Integrated Waste Management Licence Application (IWMLA) and its site selections, dedicated waste facilities are located in the south-western corner of the mining area. These facilities include a landfill site, sewerage plant, tyre storage, etc.

In line with the Environmental Management Programme (EMP), the EMP specialist study requirements for dust, visual impact and noise, the IWULA and IWMLA, a waste rock berm is located on the southern, western and northern perimeter of the mining area. Waste rock and ore stockpiles are also situated in the western sections of the mining area.

The position of Shaft 1 and Shaft 2 has been fixed and is located towards the centre of the mining area. An area approximately 400 m long and 500 m wide houses the shaft-related surface infrastructure and is fenced off by a concrete wall.

The 4 Mtpa (and space allocation for an 8 Mtpa) process plant is situated south of Shaft 1 and Shaft 2.

The engagement centre and clinic is situated next to the main entrance of the N11 on the eastern side of the mine area.

18.5 Tailings Storage Facility

The proposed tailing storage facility (TSF) site is located approximately 5 km from the Platreef mine site on the Rietfontein farm. The proposed TSF site is considered a feasible site considering all applicable engineering standards for tailings storage facilities. The TSF has an operating life of 32 years, during this time approximately 47.2 Mt of tailings will be stored within the TSF, the remainder of the tailings will be used backfill in the underground mine. The design however also caters for an 8 Mtpa ramp up in production. The TSF is compliant in terms of required tonnage profile production split between the Backfill requirement and TSF of 35% average but is conservatively designed for 40% reporting to the TSF.

The hybrid paddock deposition methodology and tailings upstream raising scheme as proposed is feasible, with 100 m wide paddocks and the required drainage systems as proposed within the design. Over and above the standard drainage requirements and implementation of international best practice, a containment berm is proposed downstream of the TSF and newly proposed N11 which serves as an additional mitigation measure to ensure the safety of the downstream communities. The tailings characteristics were confirmed during the Platreef 2017 FS, and the study also confirmed the hybrid paddocking deposition methodology as a feasible option. The proposed geometry of the TSF has an actual rate of rise of 0.9 MPa at final height and an average of 1.3 MPa over the life of the facility. The hybrid paddocking allows for a maximum rate of rise of 3.0 MPa, hence the proposed deposition methodology is well within acceptable operating standards.

The TSF is a HDPE lined TSF which is compliant to the prescribed South African National Environmental Management Waste Act, regulation R636. A staged development (4 pre-deposition stages) is a feasible strategy which will result in capital deferral over 5 year periods for the first 10 years and then 10 year deferral periods during the following 20 years. The stormwater management system and water balance for the TSF is complaint with the South African National Water Act (NWA) Government Notice 704 (GN 704). The dirty and clean water are separated, and the water routing structures are designed for the 1-in-50 year 24-hour storm event, with an acceptable freeboard. The 1-in-100 year 24-hour storm event can be contained within the storm routing structures with a reduced freeboard.

A minimum freeboard of 1.9 m must and can be achieved during the operations, which constitutes the minimum 0.8 m statutory freeboard including 1-in-100 year 24-hour storm event storage volume. The proposed decant system consisting of a floating barge, penstock tower, and outfall pipeline is functional to meet the capacity requirement. Outfall, will be routed through a silt trap for desilting and into the return water dam (RWD) to be returned to the plant.

The TSF as proposed in the Platreef 2017 FS indicates that the facility satisfies all considerations of stability investigated, namely static and pseudo static analysis considering both circular and translational failure modes. The TSF 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.

A site was conducted on 30 August 2017 to look at the current site conditions relating to wetlands and changes since 2014. There is a drainage line that runs through the site with tributaries branching from it. The dry drainage lines do not show all the typical wetland features normally used to delineate and characterise wetlands.

The wetland delineation procedure considers four attributes to determine the limitations of the wetland, in accordance with DWAF guidelines (now Department of Water and Sanitation (DWS) (2005)). The four attributes are:

- Terrain Unit Indicator – helps to identify those parts of the landscape where wetlands are more likely to occur;
- Soil Form Indicator – identifies the soil forms, which are associated with prolonged and frequent saturation;
- Soil Wetness Indicator – identifies the morphological "signatures" developed in the soil profile as a result of prolonged and frequent saturation; and
- Vegetation Indicator – identifies hydrophilic vegetation associated with frequently saturated soils.

The above indicators were used to assess the current situation on site.

For definition purposes the definition of the National Water Act, 1998 (Act No. 36 of 1998) (NWA) will be used. The NWA defines wetlands as:

"Land which is transitional between terrestrial and aquatic systems where the water table is usually at or near the surface or the land is periodically covered with shallow water, and which land in normal circumstances supports or would support vegetation typically adapted to life in saturated soil."

Of the different indicators the Terrain Unit Indicator or Hydro-Geomorphic Unit is the only consistent indicator present in the drainage lines on site. The other indicators were only found in a small area of 0,767ha in the form of an artificial wetland created by an old unregistered farm dam that was historically constructed in the drainage line.

The drainage line system was assessed as a whole and a PES calculation undertaken according to Kleynhans 1999. The value was found to be 1,9 that places it in a Category D (Largely modified. A large loss of natural habitats and basic ecosystem functions has occurred).

The modification is mostly due to sand mining, access roads for sand mining, erosion linked to the aforementioned, overgrazing, terrestrial vegetation encroachment by predominantly Dichrostachys cinerea and impoundment.

The cumulative impact on the broader catchment is seen to be limited. Mining and exploration activities by various large mining companies fall in an area that has been altered and modified significantly by residential developments and associated subsistence agriculture in the form of crop production in small plots and livestock husbandry.

Given the status of wetlands and the small area of wetland potentially impacted, the cumulative impact is seen as localised and limited.

18.6 Water Supply

18.6.1 Bulk Water Source

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

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

Ivanplats requires an average bulk water supply of 7,500 m^3/d for the operational phase of the mine. Ivanplats has been pursuing the following sources of bulk water:

- A supply from the Olifants River Water Resources Development Project (ORWRDP). This supply will be from the Phase 2B pipeline from the Flag Boshielo Dam. An outline of the ORWRDP is shown in Figure 18.8;
- A local source of treated sewage effluent (grey water);
- A supply from Magalies Water from their Klipvoor Scheme in the Crocodile West Catchment; and
- A supply of treated sewage effluent (grey water) from the Polokwane Municipality.

A bulk water supply from the one or more of the above sources together with the local groundwater sources will be used to supply the mine. Ivanplats is an active member of the Joint Water Forum (JWF) which represents the collective interests of the mines in the Eastern and Northern Limbs. Ivanplats, through the JWF, is looking to secure water from the ORWRDP and/or the Polokwane Municipality.

Figure 18.8 ORWRDP Outline

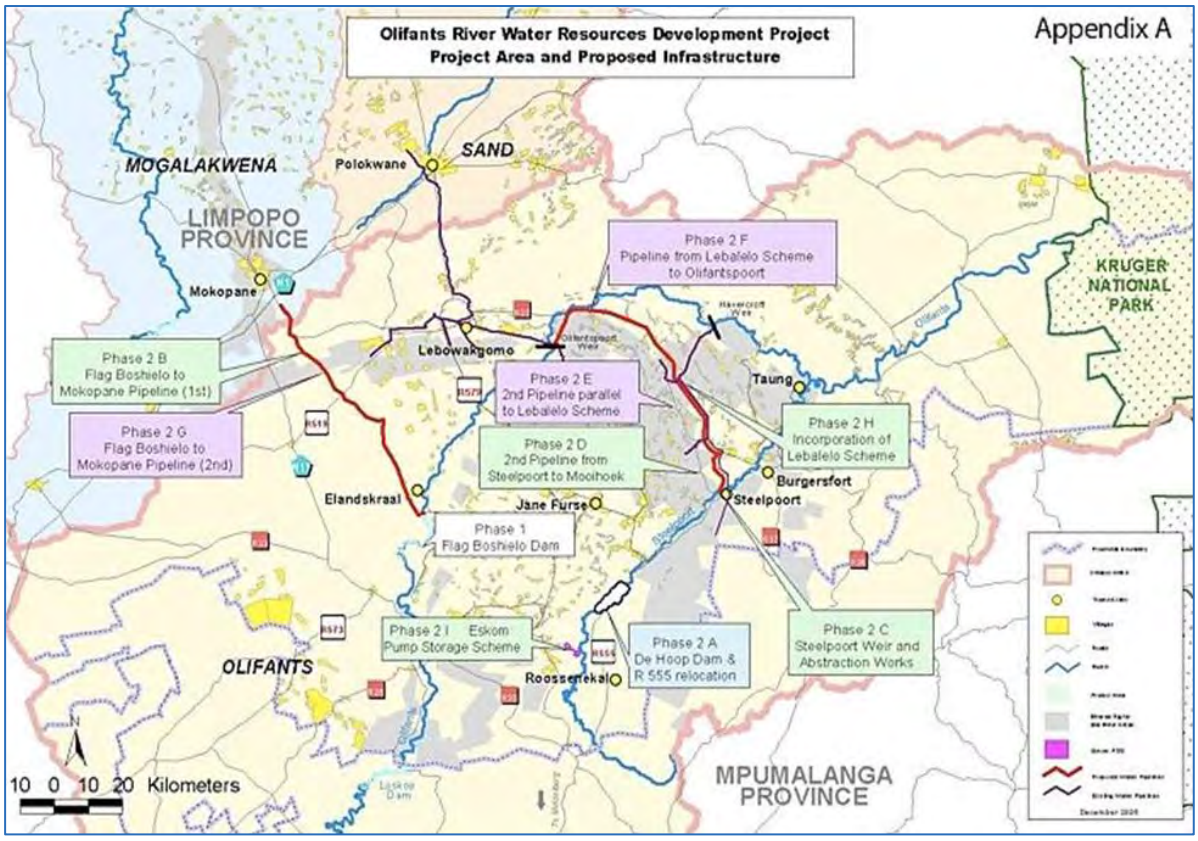


Figure by Ivnaplats 2017

18.6.2 Bulk Water Requirement

The bulk water requirement for the mine is divided into the water required for construction and water required for operations. The water requirements for the construction period (1 January 2017 to 31 December 2021) are shown in Figure 18.9 together with the sources of bulk water. The sources of bulk water during the construction period are local groundwater abstracted from boreholes on Ivanplats' Uitloop and Turfspruit properties, as well as run-off water collected on site in the Shaft 1 area Stormwater Dam, Attenuation Dams and in Stormwater Pond 3. The boreholes on Uitloop and Turfspruit can provide about 1,600 m^3/d of water. The plot in Figure 18.9 shows that the water collected on site and the local groundwater can supply the construction water requirements.

The net operational water requirements are shown in Figure 18.10 for the most likely groundwater inflow scenario. The net operational water requirements are the water requirements after supply from on-site sources such as the re-use of sewage effluent, as well as run-off collected in Attenuation Dams and in Stormwater Pond 3. The net water requirements therefore need to be supplied from a bulk water source. The variation in the net water requirements are due to the variations in run-off caused by the seasonal and annual variations in rain. The plot shows the rapid growth in the water requirement from 5,575 m^3/d to 8,689 m^3/d during the ramp up of the plant throughput from March 2022 to July 2023. In accordance to the project plan, the source of 7,500 m^3/d of bulk water needs to be available by March 2022. The bulk water supply source and local groundwater supply volumes are plotted on Figure 18.10.

The plot shows that the bulk water supply will have to be supplemented by the local groundwater source on occassions. The two bulk sources are sufficient to meet the projected net water requirements.

Figure 18.9 Mine Water Requirements Over the Construction Period and Sources of Bulk Water Supply

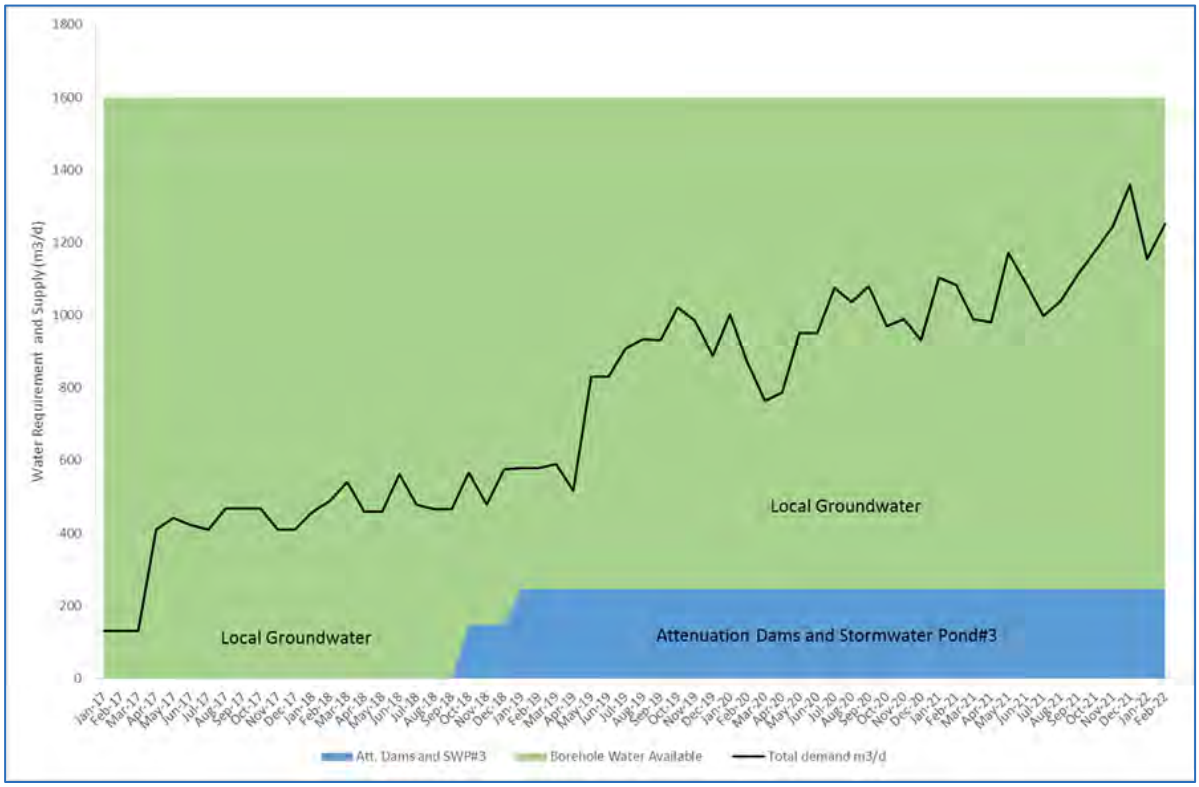


Figure by DRA, 2017.

Figure 18.10 Mine Water Requirements and Sources of Bulk Water Supply Over the Operational Period

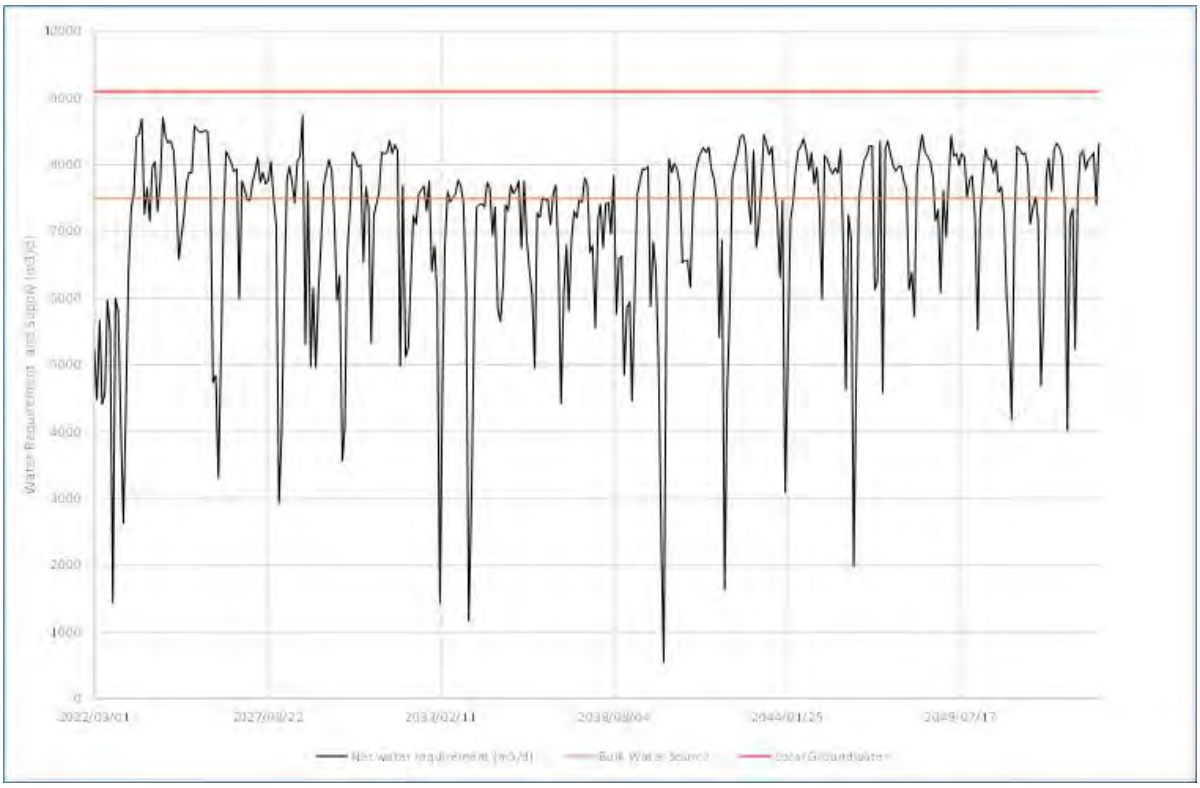


Figure by DRA, 2017.

18.7 Power Supply

18.7.1 Predicted Electrical Consumption and Notified Maximum Demand

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 are summed to arrive at a total connected load. The connected loads are reduced to connect running loads by excluding standby circuits and further reduced to absorbed power running loads as given by the process and mechanical data.

The results of these calculations are presented in Table 18.1.

Table 18.1 Notified Maximum Demand

Area	Connected kW	Running kW	Running kVA	Running kVA with PFC
6000 (Infra)	3,200	1,200	1,400	1,200
3000 (Plant)	66,700	45,500	57,000	51,000
2500 (Surf)	49,000	35,200	35,800	32,500
2600 (UG)	24,000	14,500	16,800	14,700
Total Site	143,000	97,300	111,000	99,300

The load electrical load build-up is presented in Figure 18.11. The electrical consumption over the life of mine is presented in the operational expenditure.

Figure 18.11 Electrical Load Build Up

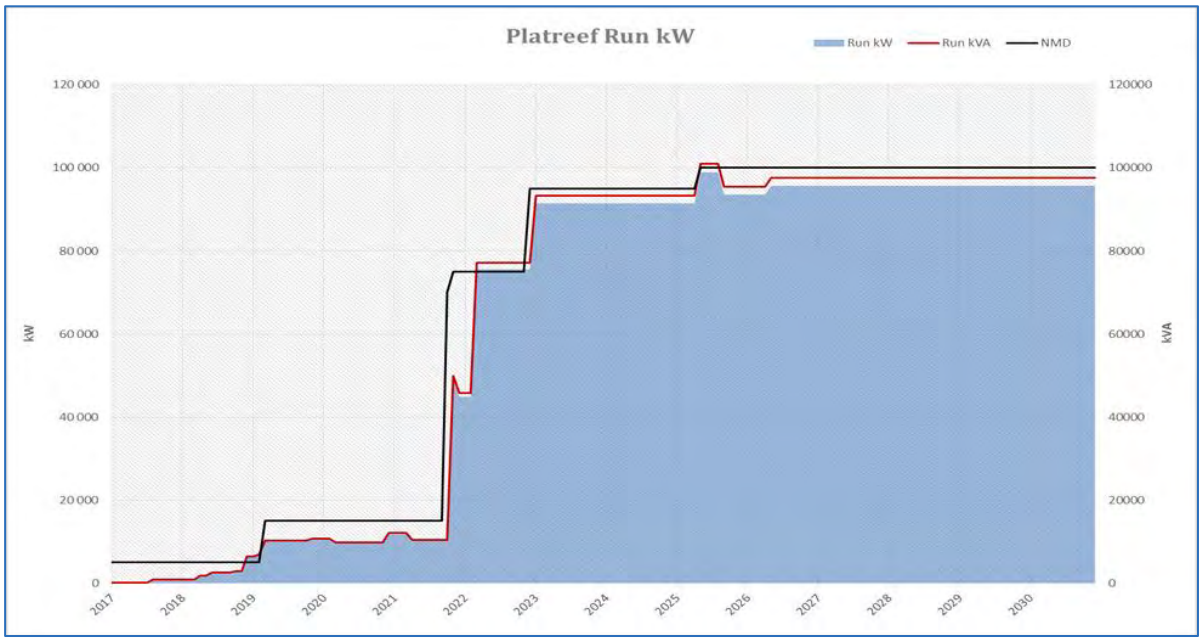


Figure by DRA, 2017.

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 70 MVA of power has been submitted to Eskom, and the budget quote received. The budget quote was accepted and the appropriate deposits provided.

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

Upon completion of the works, these will be handed over to Eskom to form part of the utilities supply network. As represented in Figure 18.12 and Figure 18.13, 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 18.12 Eskom Supply Network

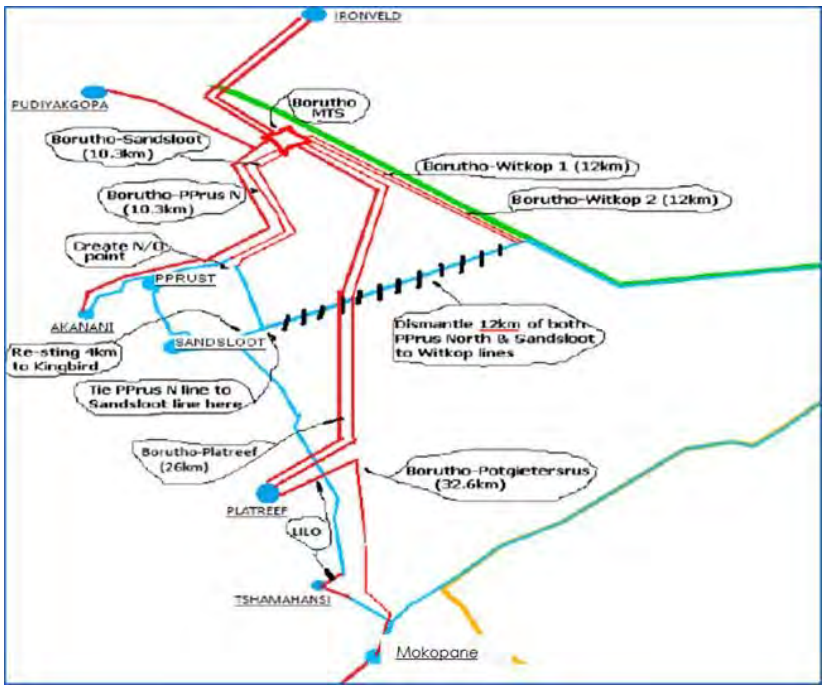


Figure by ESKOM, 2017.

Figure 18.13 Bulk Power Scope of Work

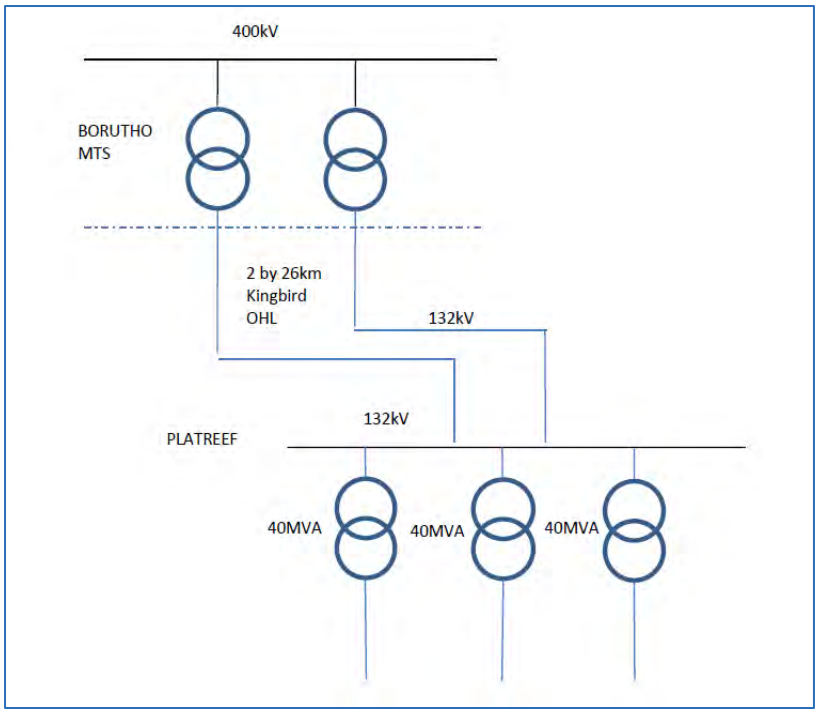


Figure by DRA, 2017.

From the Borutho MTS, 2 x 26 km Kingbird 132 kV OHLs are to be constructed to feed the Platreef 132/11 kV substation. At the Platreef 132/11 kV substation, 3 x 132/11 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 80 MVA with a future capacity of 120 MVA.

The construction of the OHL (Overhead Line) and substation is forecast to commence in the second half of 2017 with completion and handover in the first half of 2019. The final budget quote, self-build and electricity supply agreements have been drafted.

The electricity supply agreement caters for a ramp up period. From an initial supply of 15 MVA in 2019, the supply NMD will be increased as required for the load build up indicated in Figure 18.11. Eskom further indicated that a supply increase to 100 MVA is not seen as a problem as Medupi power station will be fully operational beyond 2023. The application for increased power to 100 MVA will commence once the 70 MVA agreements have been concluded.

In addition to ESKOM bulk power supply agreements and as part of the long-term sustainability plan, Ivanplats have entered into an expression of interest with an independent power producer. Ivanplats have indicated their interested of becoming an off-taker of up to 80 MW of renewable energy. The qualifying criteria would be that at inception, all energy sold to Ivanplats will be at prices below the Eskom Megaflex tariff structure. The annual tariff

increase will be limited to CPI, resulting in a predictable long-term energy pricing forecast as well as mitigation against Eskom's future pricing risk. The independent power producer model is that no capital investment is required from the client. The independent power producer would be constructing a solar PV farm in close proximity to Ivanplats, which will be able to serve multiple customers. Multiple off-takers will ensure the benefit of economies of scale and that the power purchase agreement (PPA) can be treated as an "off-balance-sheet" transaction, thus having no negative impact on Ivanplat's ROCE. Other benefits will include a lower carbon footprint, temporary and permanent job creation and economic stimulation of local secondary industries. With the rapid advancement of energy storage technology, it is envisaged that a second phase of this project will provide energy storage capacity and add significant flexibility in terms of energy usage during the peak tariff periods. This agreement will contribute positively to the long-term sustainability of the Platreef Project.

18.7.3 Construction Power Supply

The construction power supply is to be sourced at 33 kV from Eskom, the South African national power utility. The 5 MVA construction power supply has been concluded and is currently in use. The construction of the 33 kV OHL to supply the site is nearing completion. The 5 MVA construction power supply has been concluded and is currently in use.

The Eskom 33 kV is to feed the 33/11 kV substation located on the Platreef site. The completed 33/11 kV substation is to enter operation as soon as the Eskom line is energised.

Power for the various construction activities on site will be obtained from the substation at 11 kV or the 33 kV OHL as appropriate.

18.7.4 Alternative Power Supply

Various alternative power generation options were considered in the Platreef Project feasibility study.

The initial brief was to consider alternative sources of power for small power and lighting up to full bulk power requirements. This was modified to concentrate further efforts on alternatives equivalent to 20% of the bulk power for curtailment should Eskom request the client to do so and emergency generation requirements. Alternative sustainable LV energy enhancement concepts for site were also investigated to reduce the demand off the Eskom grid.

The primary emergency backup supply is to be supplied from a 20 MVA 11 kV generator plant consisting of multiple containerised prime rated 2.5 MVA generators. The containerised generator solution was preferred, as it is a simpler build up in single unit increments as required. A single feed to the consumer substation has been provided.

For the tailings line booster pump station and the return water pump station, additional individual generators have been included to allow the system to continue operation should the OHL to these outlying facilities be unavailable for any reason.

In the main parking area a 2,600 kWP solar photovoltaic (PV) plant is to be installed as an integral part of the carport installation. It is a grid tied system that does not store energy and thus does not provide back-up power. The energy produced by the plant will be utilised to offset energy consumption during the daytime. Non storage grid tied systems are the most cost effective solar PV systems with significant potential energy savings.

Emergency lighting is required in various areas of the site to provide safe egress from structures and buildings. It is traditionally supplied by battery back-up lighting or lighting fed from uninterruptable power supply (UPS). Here 4 kWP solar PV systems with battery storage provide the back-up power required. The PV panels are to be mounted on the substation roofs with the advantage of re-charging the batteries during daytime with solar energy.

Fuel Cells are a set of technologies that generate electricity from natural gas through electrochemical processes as opposed to combustion. It is a clean-burn technology where the Fuel cells offer heat as a by-product which can be used for heating/cooling (in conjunction with an adsorption chiller). A 100 kW and a 400 kW fuel cell are included for powering the community centre and the main office complex respectively. These fuel cells use platinum group metals as a catalyst which will generate a market for platinum.

18.8 Earthworks

18.8.1 Earthworks Commodity Flow Philosophy and Strategy

On account of the various earthworks commodities that occur and are required during the construction phase, a basic commodity flow model was developed for the project. The commodity flow directly affects the capital estimates and is therefore important to illustrate. The intent of this model is to indicate the net quantities of material/commodities that would need to be stockpiled whether temporarily or permanently. It further indicates commodity requirements, and if importing of commodities is required onto the site.

The different type of earthworks commodities are as follows:

Topsoil

Topsoil is defined as the first 200 mm of material removed (cut to stockpile) from any excavation done on surface. Topsoil may only be used to clad the perimeter. The remainder of the topsoil needs to be stockpiled in a defined area for topsoil storage only. A portion of the perimeter berm will be used for this and will be demarcated as such.

Subsoil (Soft Excavation)

Subsoil is defined as the next type of soft soil material removed (cut to stockpile) from excavation after the topsoil has been removed. The depth of subsoil within the excavation is determined by either bottom of excavation level or when 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 removed or loaded by track-type excavator with a mass of at least 35 t and an engine developing approximately 220 kW. 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.

G4 Material

It is a type of material used in layer works, that complies with certain material properties and classification that could be obtained naturally from a borrow pit or rocky type of material that requires crushing. With no borrow pits available, all G4 material required will be obtained from rock that has to be crushed. G4 material could either be obtained commercially (off site crushing operation) or on site by establishing a crushing plant to crush available waste rock.

Waste Rock

Waste rock is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content below the mine cut-off grade. This material will report to the waste rock stockpile.

Ore

Ore is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content greater than the required minimum concentrator mill feed grade.

Low-Grade

Low-grade is material volumes produced from underground mining activities and hoisted to surface with a 3PE+Au content lower than the required minimum concentrator mill feed grade.

18.8.1.1 Surface Earthworks Material Flow

The flow of surface earthworks material could be divided into two categories:

Cut to stockpile

Cut to stockpile is the loading of material generated from excavations, hauled, dumped and stored in a dedicated spoil or stockpile area.

Cut to fill

Cut to fill is the loading of material obtained from stockpiles or commercial sources, hauled to the required position, processed, levelled and compacted to the required specification.

Table 18.2 indicates the expected surface earthworks material flow cubes per type of commodity and category. The total amount of topsoil available is 359,500 m³ which could be used to clad the perimeter berm and stored for future rehabilitation. The total amount of subsoil available from the excavations is 1,967,700 m³, and hard material is 970,700m³. The total amount of fill requirement is 1,668,600 m³ of subsoil and 800,700 m³ G4 type of material. Of the 800,700 m³ of G4 type of material required, 84,000 m³ is required for base layer works and 716,700 m³ for sub-base layer works (bulk layer works). It is recommended that for the base layer works commercial G4 material is used, and for the sub-base layer works the hard excavated material could be crushed and used as an alternative source.

Table 18.2 Surface Earthworks Material Flow

Area	Cut to Stockpile (m³)			Cut to Fill (m³)	
	Topsoil	Subsoil	Hard	Subsoil	G4 Material
2000	111,600	552,900	73,500	469,000	295,000
3000	29,300	110,800	53,700	122,700	116,900
6000	218,600	1,304,000	843,500	1,076,900	388,800
Total	359,500	1,967,700	970,700	1,668,600	800,700

Table 18.3 indicates the balance flow material that is available on surface and what is required for layer works. A total of 912,600 m³ will need to be stored/stockpiled on surface. It is recommended that this material be stockpiled within Perimeter Berm Stockpile Areas.

Table 18.3 Surface Earthworks Material Flow Available and Required

Surface Material Commodity	Available from Excavations (m³)	Required for Layer Works (m³)	m³ to be Stockpiled
Soft Fill (Subsoil)	299,100	1 668,600	299,100
Hard	254,000	716,700	84,000
Topsoil	359,500		359,500
Total to be stockpiled from surface			912,600

18.8.1.2 Underground Earthworks Material Flow

Table 18.4 indicates the expected underground material flow and volumes will be hoisted to surface.

Table 18.4 Surface Earthworks Material Flow

Total Waste Rock (m³)	Waste to Surface (m³)	Waste to Stopes (m³)	Ore (m³)	Low Grade (m³)	CRF Required (m³)
7,000,000	4,200,000	2,800,000	56,000,000	2,700,000	315,000

18.8.1.3 Surface Storage Capacity

Table 18.5 indicates the available storage capacity per commodity and the required volume that needs to be stockpiled on surface.

Table 18.5 Surface Storage Capacity

Area	Stockpile Capacity (Mm^3)	Required Capacity (Mm^3)	Spare Capacity (Mm^3)
Perimeter Berm 1	0.55		
Perimeter Berm 2	0.31		
Perimeter Berm 3	0.3		
Perimeter Berm 4	0.34		
Perimeter Berm Total	1.5	0.9	0.59
Waste Rock Stockpile	5	4.2	0.8
CRF Stockpile	0.05	0.05	0
Low-Grade Stockpile	2.58	2.7	-0.12
Ore Stockpile	0.71	0.5	0.21

From Table 18.5 the following could be summarised:

- Perimeter berms will be constructed out of 912,600 m^3 surface materials (topsoil, subsoil and hard material), and 587,300 m^3 of underground waste rock will be required to complete these berms.
- The waste rock stockpile has spare capacity of 800,000 m^3.
- It is planned to have 50,000 m^3 crushed waste rock material conforming to the required CRF specification on a surface stockpile.
- The low-grade stockpile has a shortfall capacity of 123,000 m^3. This shortfall could be made up by either reducing the size of the ore stockpile area or by making use of the surplus capacity of the waste rock stockpile area.

18.8.2 Terraces

The founding design provided as part of the geotechnical report suggests blending the excavated material with G4 material in order to achieve a required bearing capacity. Allowance was made to produce some of the G4 material required by crushing and screening the excavated material and the balance from a commercial source.

18.8.3 Perimeter Berm

To mitigate the noise, dust and the visual impact of the mine, the relevant specialist studies were performed, modelling a 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.

This perimeter berm doubles as a suitable stockpile for overburden generated during construction as well as surplus waste rock from underground development. One of the key visual impact study requirements is to clad the berm with topsoils and to hydro-seed the completed sections as the berm is being built.

The perimeter berm is divided in four sections with the sections scheduled to be built at different times.

Figure 18.14 Perimeter Berms

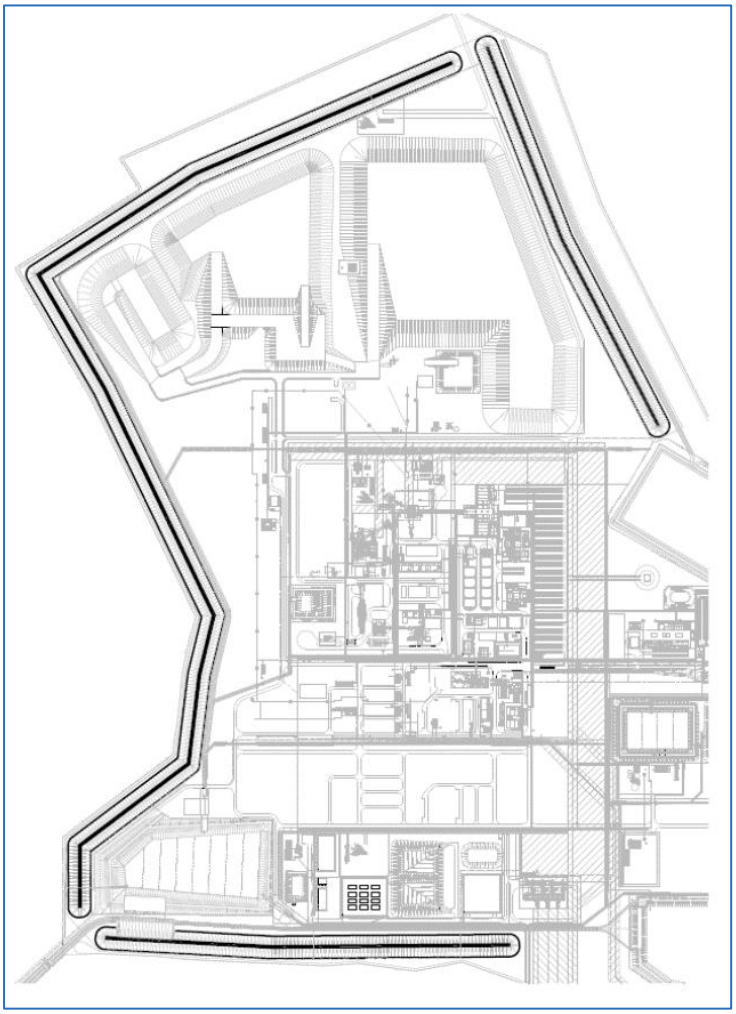


Figure by DRA, 2017.

This design still includes flexibility to change the sequence of building the berm should later developments require the project to do so.

An alternative design of the perimeter berm constructed out of waste rock was completed during the Platreef 2017 FS. The alternative design was presented to the Department of Water and Sanitation (DWS) on 1st July 2016. The alternative design indicating a barrier system without a Class C (single geomembrane) liner system. It was confirmed that a stability assessment was done of the perimeter berm facility, compliant to an acceptable factor of safety. A seepage collection drain will be installed.

DWS confirmed acceptance of the design as proposed.

18.8.4 Roads, Walkways and Parking Areas

The main access road passing the site is the N11. As this is a national road, it falls under the SANRAL jurisdiction. With regards to off-ramps and exits from the N11, SANRAL has approved one intersection. To achieve an exit from the N11 onto the mine property, an access position was chosen, with an existing intersection into a village on the eastern side of the N11.

The N11 intersection was designed and constructed in line with SANRAL requirements to allow for heavy vehicles turning north and south back onto the N11.

The internal road design philosophy is that as far as possible delivery vehicles will remain on the main entrance road to the required delivery points and parking areas, with internal roads reserved for the delivery of equipment from the stores to the work area. The internal roads and parking take into account the traffic flow inside the mine area. Gates separate areas in order to restrict access, but without reducing serviceability and production.

Different types of surface finish and layer works have been designed for different types of application and road uses within the mine area. The following types have been used as part of the design and layout:

- Asphalt road layer works used for the main entrance access road.
- Paving road Type 1 layer works used for internal access roads in and around the shaft, process plant, workshop and store areas
- Paving road Type 2 layer works used for parking and walkway areas.
- Concrete road layer works used in the process plant area between the mill building and the filter building
- Gravel road layer works used mainly for maintenance access
- Haul road layer works used for the main hauls required in the rock handling area between the different types of stockpiles.
- Service road layer works used for security access road around the perimeter and area with little expected traffic.

18.8.5 Stormwater Management

The mine area receives clean stormwater runoff draining from the urban areas located to the north and east of the mine site. Regulation 704 of the National Water Act of 1998 requires that clean stormwater runoff is kept separate from the polluted runoff generated on the site. To meet these requirements, clean and polluted stormwater management systems were designed for the site. The stormwater system was sized to convey the flood peak from the 50 year 24 hour storm event as per Regulation 704. Sufficient freeboard was provided on the stormwater channels to enable the runoff from the 100 year 24 hour event to be conveyed in the channels.

The clean stormwater management system consists of concrete lined channels constructed along the north and eastern sides of the mine area (See Figure 18.15). The stormwater system concentrates the flow so that it cannot be safely discharged into the communities located to the south of the mine area. Therefore, two attenuation dams and a channel directing the attenuated flood peaks to the south west corner of the mine area were included in the design. The attenuated flood peaks are discharged into an unpopulated agricultural area to the south west of the mine area (See Figure 18.17).

The polluted stormwater on site is collected in a system of internal concrete lined stormwater drains (See Figure 18.16). This internal drainage system discharges into Stormwater Pond 3 located in the south west corner of the mine area. The stormwater collected in this pond is re-used on site in the process. Stormwater Pond 3 is sized to spill on average once in 50 years as per Regulation 704. The spills from Stormwater Pond 3 report to the south west stormwater drain.

Figure 18.15 Clean Water Drains

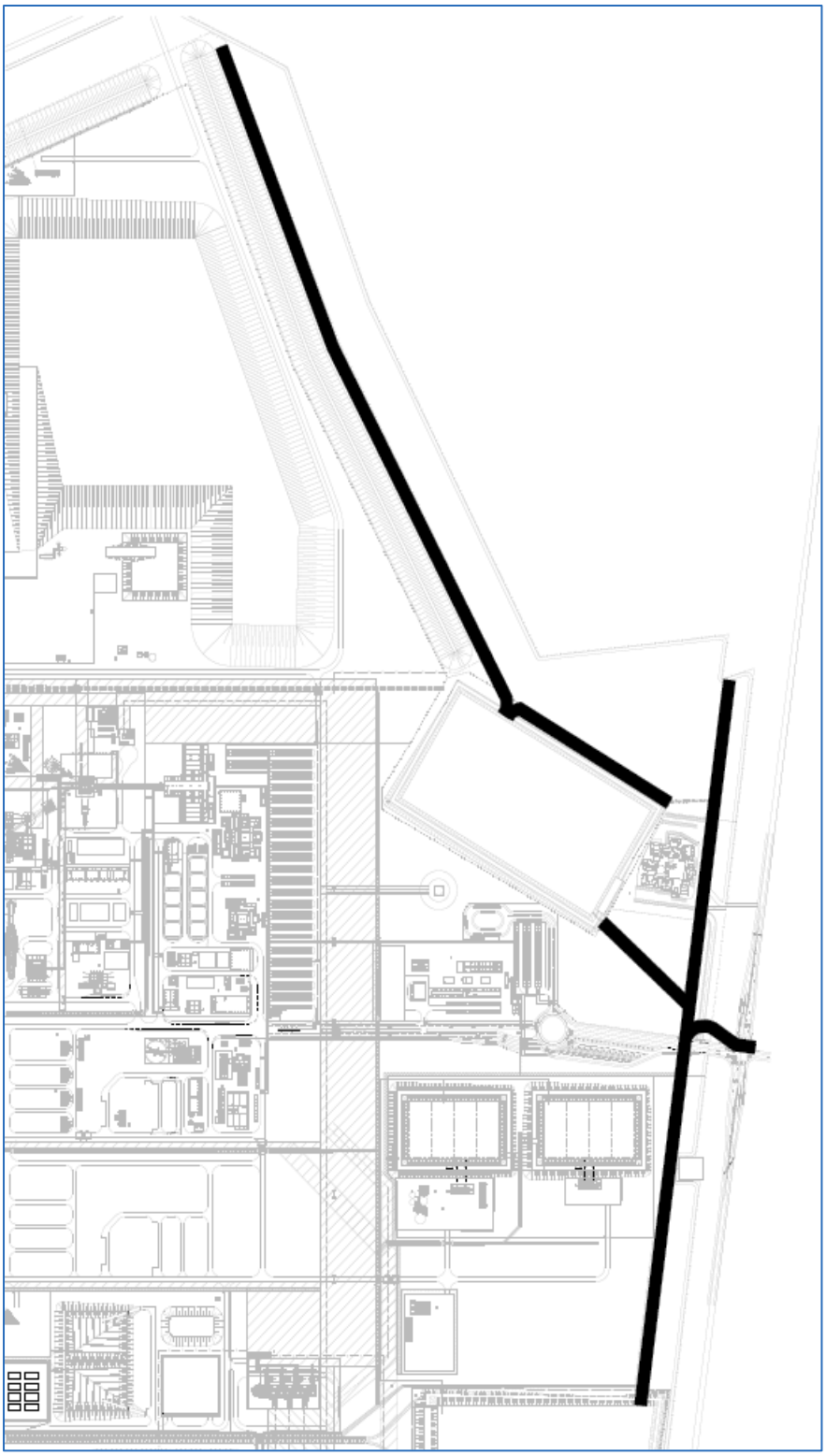


Figure by DRA, 2017.

Figure 18.16 Stormwater Run Off Drains

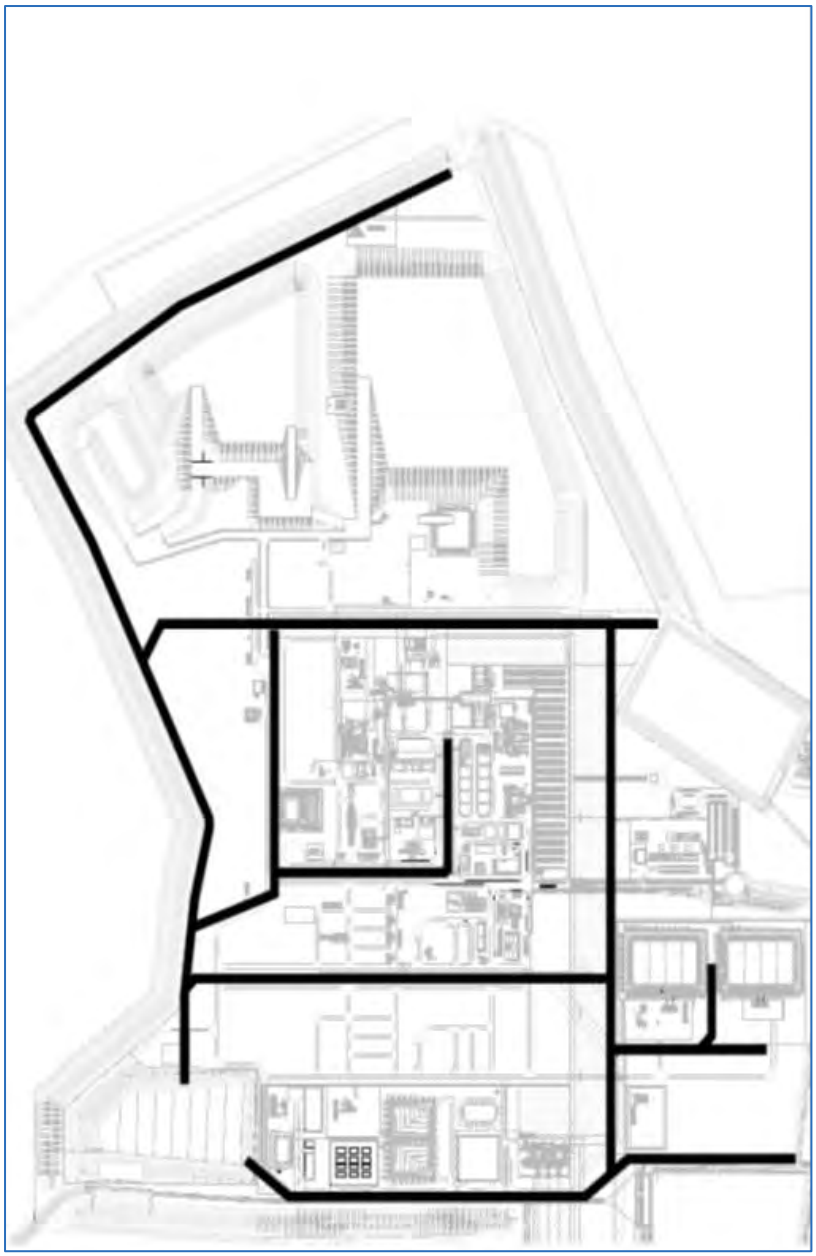


Figure by DRA, 2017.

Figure 18.17 Discharge Drain

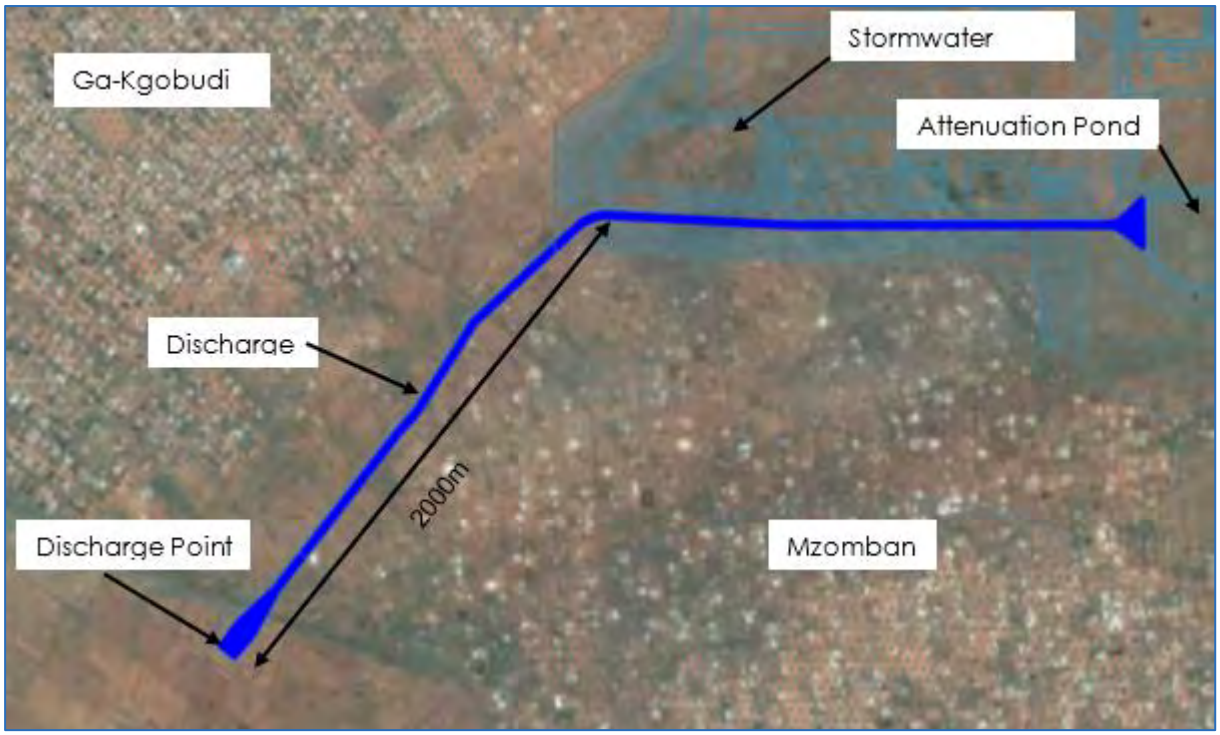


Figure by DRA, 2017.

18.8.6 Water Storage Facilities

Table 18.6 indicates the main water storage facilities, lining type and their capacities in the different areas of the mine.

Table 18.6 Water Storage Facilities and Capacities

Water Storage Facility	Area	Capacity (m^3)	Lined
Raw Water Dam 1	6000	49,000	HDPE Lined
Raw Water Dam 2	6000	49,000	HDPE Lined
Attenuation Pond 1	6000	125,000	Unlined
Attenuation Pond 2	6000	150,000	Unlined
Stormwater Pond 3	6000	225,000	HDPE Lined
Pre-Chlorinated Tank	6000	2,500	Concrete Tank
Filtered Raw Water Tank	6000	5,000	Concrete Tank
Process Water Tank	3000	1,000	Concrete Tank
Clean Water Tank	3000	5,000	Concrete Tank
Fire Water Tank Plant Area	3000	1,000	Sectional Steel Tank
Mine Return Water Dam 1	2000	1,000	Concrete Tank
Mine Return Water Dam 2	2000	1,000	Concrete Tank
Fire Water Tank Shaft Area	2000	1,000	Sectional Steel Tank

18.8.7 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,
- Ore,
- Waste rock uncrushed and crushed,
- Subsoils (soft and hard), and
- Temporarily imported G4 material for platform construction.

The overall stockpiling strategy in terms of design and operation thereof has been done in line with the EMP requirements for minimising dust and visual impact. Thus, the height at which material is discharged from conveyors for stockpiling purposes is to be kept at a minimum. Materials deposited onto low temporary stockpiles are manoeuvred further by means of mobile earthmoving equipment (trucks, loaders and dozers) to build the individual type of stockpiles.

All stockpiles have been designed with the required footing and drainage as per the GAA requirements stipulated in the IWULA application that was lodged with the Department of Water Affairs. The footprint of each stockpile was modelled to optimise the footprint for the required capacity.

A further requirement for waste rock is to re-use the waste rock as earthworks backfill material and as CRF for underground backfilling. To allow for this, waste rock has to be crushed and screened to suitable sized materials.

A mobile crushing and screening plant has been allowed to facilitate this. The plant is made up of a modular jaw crusher complete with a vibrating grizzly which is manually fed by a front end loader. The crushed product then passes over a triple deck screen with the undersized being stockpiled while the oversized is sent to a cone crusher to be recirculated.

If required, the imported G4 material for layer works will also be stockpiled temporarily in the waste rock stockpile area.

The capacities of these stockpiles are indicated in Table 18.7. For the layout and position of these stockpiles, refer to Figure 18.14 for the perimeter berms and Figure 18.18 below for the stockpiles.

Table 18.7 Stockpile Capacities

Stockpile	Capacity (1,000,000 t)	Capacity (1,000,000 m^3)	Max Height (m)
Low-Grade Stockpile	5.15	2.58	25
Ore Stockpile	1.42	0.71	25
Waste Rock Stockpile	10	5	25
CRF Stockpile	0.1	0.05	10
Perimeter Berm 1	1.1	0.55	10
Perimeter Berm 2	0.62	0.31	10
Perimeter Berm 3	0.6	0.3	10
Perimeter Berm 4	0.68	0.34	10

Figure 18.18 Material Storage Stockpiles

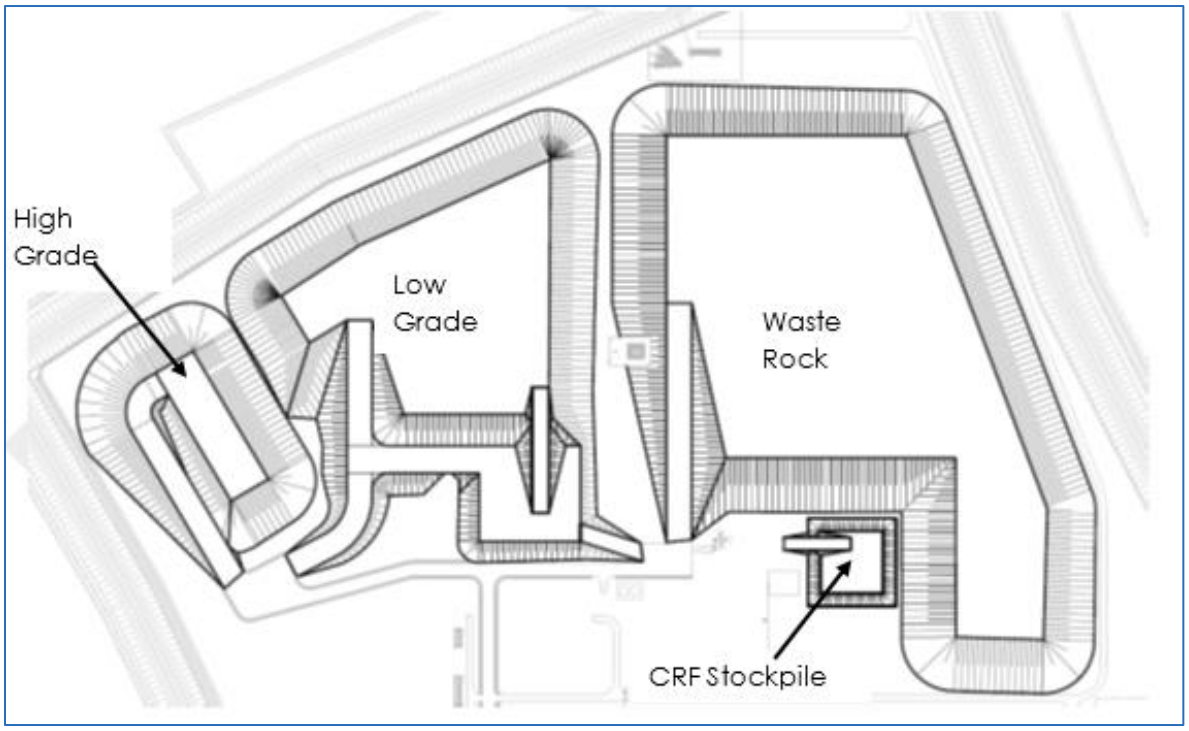


Figure by DRA, 2017.

18.8.8 Services

18.8.8.1 Water Treatment Plant

The area in which the Platreef Mine is situated is commonly known as a water scarce area, and as such the need to conserve water and re-circulate water within the mine site is required.

The raw water source to the mine could potentially originate from a wastewater treatment plant (WTP). Although agreements will be put in place to manage the quality of raw water to the treatment plant, there could potentially be instances where the raw water is off-specification. In order to cater for off specification raw water a treatment plant will be considered at the mine site. The cost for a water treatment plant is included in the capital estimates and IWULA approval has been granted.

Raw water will be pumped to one of two 49,000m³ raw water dams. In order to maximise the amount of water contained within the mine site, Stormwater Pond 3 discharge will be pumped to one of two raw water dams. From here the water will be pumped to lamella settlers and thereafter sand filters. Clean water will then be produced from this section from where it will be pumped site-wide for use within the mine site. A bleed of clean water will then be directed to a reverse osmosis (RO) plant which will produce two types of water. Prior to chlorination at the reverse osmosis plant, pre-chlorinated water will be produced. The pre-chlorinated water will feed the fridge plant for ventilation air purposes. Water that will be sent through the reverse osmosisant, including the chlorination step, will be used as potable water. This potable water will be safe for human consumption. The brine produced from the RO plant will be pumped to the Tailings Disposal Tank situated at the process plant to be pumped away to the TSF. Figure 18.19 below is a schematic showing the water treatment works.

Figure 18.19 Schematic Showing Water Treatment Works

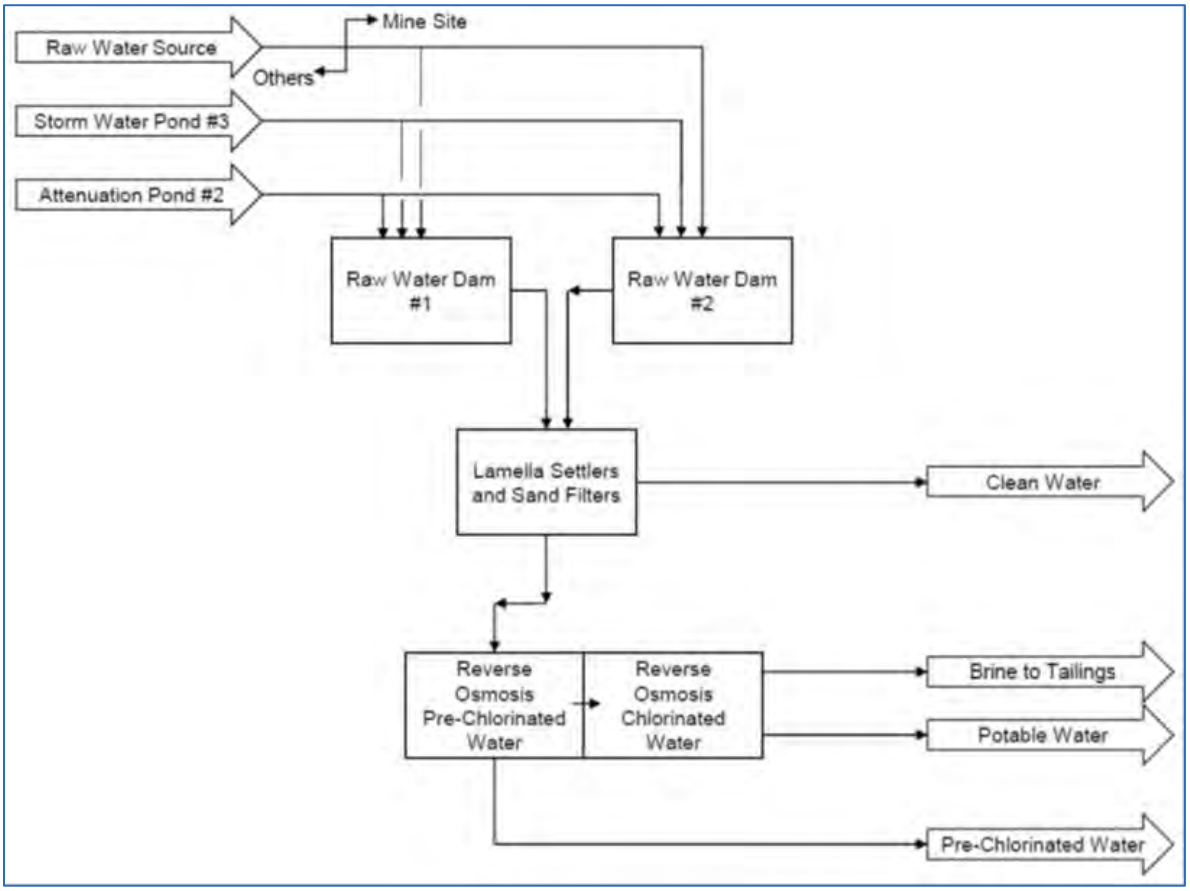


Figure by DRA, 2017.

Clean Water Quality and Quantity

The clean water quality will be produced to the following specification:

- pH 6.5 – 8.0
- Solids Content:
  - Dissolved: 1,000 ppm (mg/L)
  - Suspended: 100 ppm (mg/L)
  - 100% of +250 mesh (60 µm) particles removed
- Maximum Individual Dissolved Ions:
  - Hardness (Ca-, Mg-) 200 ppm (mg/l) as $CaCO_3$
  - Calcium carbonate ($CaCO_3$) 10 ppm (mg/l)
  - Sulphate (SO4) 50 ppm (mg/l)
  - Chloride (Cl) 1,000 ppm (mg/l)

Clean water produced will cater for fluctuations within the design for approximately 400 m^3/hr peak flows required within the mine site.

Potable Water Quality and Quantity

The potable water quality must adhere to the South African National Drinking Water Standard (SANS) 241. Potable water will be produced to cater for peak flows of approximately 150 m^3ph.

Pre-Chlorinated Water Quality and Quantity

Pre-chlorinated water quality will adhere to similar potable water treatment processes, with the exception of the chlorination step. Pre-chlorinated water will be produced to cater for peak flows of approximately 105 m^3ph.

The water treatment plant will adhere to the Platreef engineering design criteria and specifications. Regular quality-control samples will be taken from the water treatment plant in order to ensure that specifications of the products are being met.

18.8.8.2 Assay Laboratory

A centralised mine site laboratory has been allowed for. This laboratory aims at servicing the geology, mining, process plant (concentrator), infrastructure and environmental analysis requirements. This laboratory is a combined laboratory to receive, prepare, analyse and assay all samples from the relevant departments. There will however be certain samples that would require being sent to an external laboratory for control and legislative purposes.

For the geology samples, the grade control channel sampling and evaluation core sampling has been allowed for as part of an internal geology laboratory. This will include for any preparation work required to the point of delivery to the mine site laboratory for assay preparation and analyses.

The 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.

The process plant samples will be sent to the metallurgical laboratory section for sample preparation prior to analysis and assaying.

The infrastructure samples comprises of the raw water samples, treated potable water, clean water and pre-chlorinated samples. These samples will be sent to the mine site laboratory as control samples for monitoring purposes.

The environmental samples comprise of surface water samples, ground water samples, soil samples, occupational hygiene samples and drinking water samples. The control samples will be analysed at the mine site laboratory, and the rest of the samples will be collected and sent to an external laboratory.

The laboratory will have a Laboratory Information Management System (LIMS) and unique barcoding system that will be linked to the main Platreef control network.

18.8.8.3 Buried Services

All buried services are designed according to SANS 1200. These include:

- Earthworks, i.e. trenching (SANS 1200D)
- Bedding for pipes (SANS 1200LB)
- Piping, valves and valve chambers, anchor/thrust blocks and manholes (SANS 1200L)
- Concrete and miscellaneous metal work (SANS 1200G)

A drawing was created for each service, detailing the assumed route. From these drawings, take-offs are generated and priced. The buried services allowed for in the mining area are:

- Potable Water
- Fire Water Reticulation
- Sleeves

18.8.8.4 Sewer Reticulation

The domestic sewage for the mine area is designed so that all sewerage is collected from various points, and then flows through a 100 to 250NB, depending on the flow, polypropylene gravity fed buried pipe system to a concrete-lined sump at the sewer plant.

The sump is adequately sized in order to provide 24 hours of pump downtime.

18.8.9 Weighbridges

Two weighbridges have been allowed for. One is located near the entrance to the plant, shaft and stores area, and the second one is located in the plant area near the concentrate building. The one placed at the concentrate building is specifically to measure concentrate and is more accurate than the one at the entrance to the plant, shaft and stores area. The weighbridges are located so that vehicles entering or leaving the area can be weighed without disrupting traffic flow.

18.8.10 TSF Pipeline Servitude

The tailings pipeline servitude is split in to two portions. The first portion of the servitude runs inside the mine up to the boundary fence in the south-eastern corner, and the second portion runs from this point at the fence line, crossing the N11, all the way up to the TSF 7.5 km north-east of the mine area. Refer to Figure 18.20 for the tailings line servitude and Figure 18.21 for the proposed cross-section of the tailings line servitude.

In Portion 2 of the servitude a fence line runs either side of the servitude, separating the tailings lines, road, power line, berm and paddocks from the surrounding areas. A power line runs parallel to the tailings lines in order to provide power for the booster pump station and the return water pumps.

Berms run alongside the pipelines in order to catch possible spillages and to guide it towards paddocks. Allowance has also been made to install culverts where required to provide crossing access over the tailings line servitude for people and livestock.

Figure 18.20 Tailings Pipeline Servitude

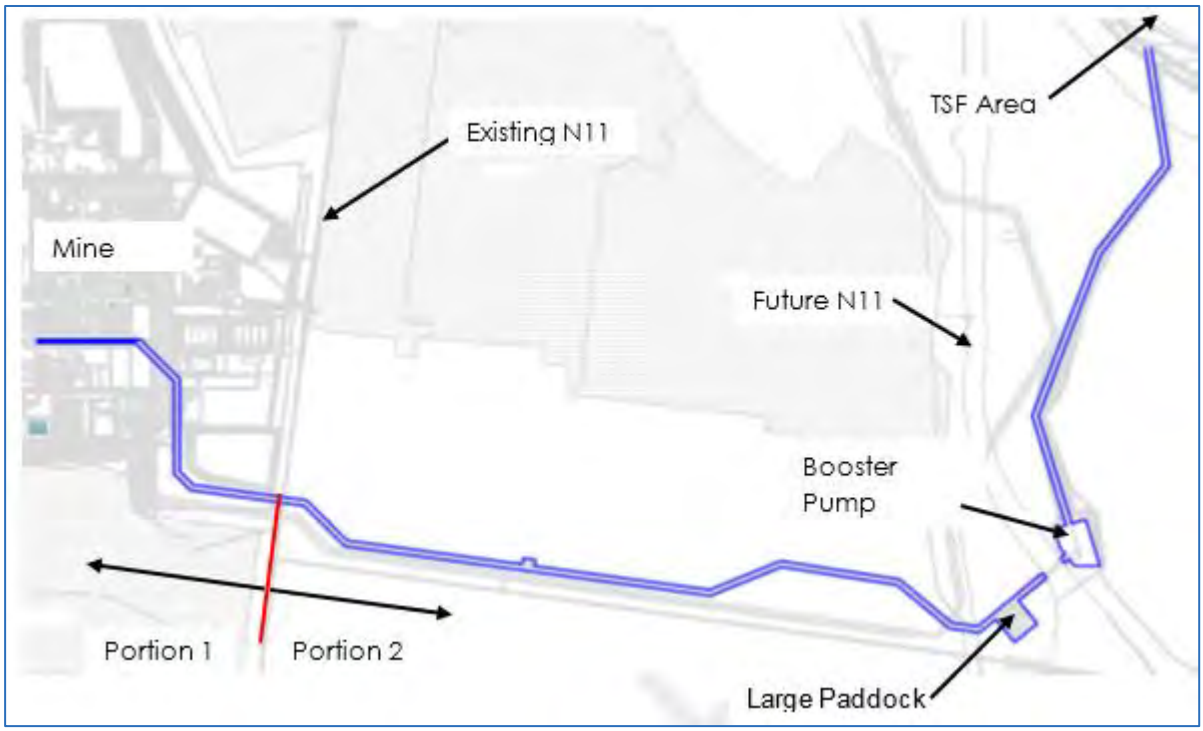


Figure by DRA, 2017.

Figure 18.21 Tailings Servitude Section

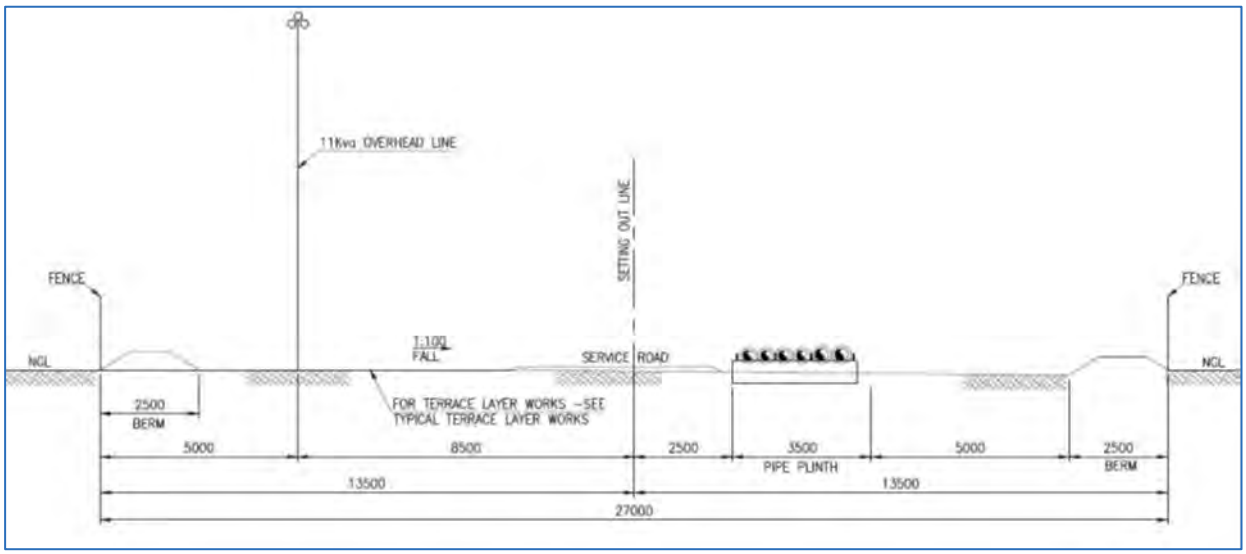


Figure by DRA, 2017. All figures shown in millimetres

18.8.10.1 Overland Piping

Tailings Lines

The tailings lines consist of pipelines to accommodate the 2 Mtpa and 4 Mtpa phases. The first phase is when the first 2 Mtpa plant is running. This will utilise one 250NB HDPE-lined mild steel pipeline. The second phase is when the second 2 Mtpa plant is brought online, providing a total throughput of 4 Mtpa. During the 4 Mtpa operation, provision has been made for one off duty and one off standby 300 NB HDPE-lined mild steel pipeline. Tailings from the process plant is pumped to a booster station situated approximately 5 km from the process plant. The total length of the tailings servitude is approximately 7.5 km between the process plant and the TSF. The tailings pipelines are routed in the tailings servitude and are supported on concrete sleepers at 6 m spacing intervals.

Tailings Water Line

The Tailings Return Water consists of two duty/standby 250 NB HDPE pipelines, which run from the TSF Return Water Dam to the Process Water Tank. This pipeline is routed in the tailings servitude, and the TSF corridor servitude and is supported on concrete sleepers at 6 m spacing intervals. The feed line providing clean water to the booster pump station consists of a 250 NB HDPE pipeline.

18.9 Buildings

The buildings have been categorised:

- Architectural Buildings
- Workshops and Stores
- Change House Buildings
- Electrical Buildings

The buildings are typically constructed out of face brick, complete with aluminium windows and wooden doors. The roof is constructed out of steel and timber roof trusses with inverted box rib (IBR) roof sheeting. Included in the buildings are all small power, lighting and furniture.

18.9.1 Workshops and Stores

The workshops are typically designed as a sheeted steel structure building with civil bases, plinths and a surface bed. These buildings also have filled in brick work on the sides and brick offices and small stores on the inside to facilitate the people working in this building. Roller shutter doors as well as doors and windows are included. Included in the workshops are all small power, lighting, general tools and equipment, furniture and where indicated, an overhead crane.

18.9.2 Change House Buildings

The change house building is also built as a sheeted steel structure building with civil bases, plinths and surface bed filled-in brick work with all of the necessary sanitary facilities, benches, light fittings, extractors, shelves and geyser/boilers. The change house building has been designed in such a way that it encompasses other buildings such as the lamp room, boot wash facility, laundry room, store rooms and offices. In addition, the design provides a logical, sequential flow for employees who are either starting or ending a shift, while ensuring the safety and efficiency of the employees.

18.9.3 Electrical Buildings

All the substation buildings are structural steel buildings with IBR roof sheeting, elevated concrete slab and 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 EDC 1051-DC-50-15, as well as compliance with the relevant codes of practice and the local authorities.

Fire water will be drawn from a dedicated source, ensuring the availability of 675 m^3 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 dual suction, c/w vortex inhibitors. Each tank section will have an infill, overflow, drain, suction, test return and diesel engine cooling water return line nozzles. The tanks have been designed to supply dedicated firewater for a duration of 90 minutes at maximum flow.

Pressurised fire water will be distributed around the mining and shaft area, conveyors and localised general infrastructure buildings. A dedicated fire water pump set shall achieve this.

Where practicable, fire water reticulation pipework will be buried and will be HDPE class 16 pipe work and 160 mm diameter throughout. Above ground reticulation pipe work will be SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be class 16. All isolation/section valves will be UL listed/FM approved. Fire hydrants throughout will be spaced at 90 m intervals.

Fire hydrants are to be fed off the fire water column and to be placed no further than 60 m apart in the required areas. The maximum permissible velocity is 8 mps through any pipe and 6 m/sec through any valve. Cognisance has been taken to ensure compliance with these limitations when sizing the ring main.

Each tank section will have an infill, overflow, drain, suction, test return and diesel engine cooling water return line nozzles.

Pressurised fire water shall be distributed around the process plant area, conveyors and localised general infrastructure buildings. A dedicated fire water pump set shall achieve this. This will include a diesel, electric and jockey pump delivering a duty 450 m^3 @ 850 kPa. The starting mechanism will be automatic, via a series of differential pressure switches connected to a fire pump panel and annunciator. The electric pump will act as the primary pump. The diesel pump will act as a standby pump, operating should the electric pump fail to start.

The jockey pump will deliver a flow of approximately 1,200 Lpm at the designed duty pressure, maintaining system pressure in the event of fire hydrant and/or hose reel operation or the presence of small leaks. Prior to delivery to site, the pump will be tested in accordance with ASIB codes of practice and under the supervision of the SABS.

All fire hydrant, hose reel and protection system risers will be steel and protected accordingly against corrosion.

18.10.1 Fire Detection

All substations and motor control centres (MCCs) have both a smoke detection system in the room as well as a Very Early Smoke Detection Apparatus (VESDA) in the cabinets.

Each building will have its own panel for remote monitoring of its status via potential free contacts.

Each building will be zoned accordingly, requiring a double knock (two adjacent zones) in simultaneous fire condition prior to the discharge of the gaseous suppression system, thus preventing the possibility of accidental discharge. Each panel will also contain potential free contacts used for the shutting down of associated equipment (main incomer, air conditioning system, etc.)

In general buildings such as the control room, a smoke detection system has been included. Each building will have its own panel for remote monitoring of its status via potential free contacts.

Linear Heat Detection cable will be installed along the full length of the conveyors and will be used for belt shutdown only. Due to the fact that the protection system along the conveyors is by means of sprinkler systems, the stopping of the belt in the presence of fire is of paramount importance in order to allow sufficient time for the sprinklers to activate.

The Linear Heat Detection cable is stainless steel braided and has a confirmed temperature initiation system. A simple break in cable or loss of resistance will not initiate belt shutdown unless there is a confirmed high temperature, eliminating the risk of unnecessary or accidental shutdown.

The conveyors have been separated into 200 m sections or zones. Each zone has its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

Flame detectors have been placed at strategic locations and will detect a moving fire in its incipient stage. The detection system will initiate belt shutdown as well as activate the solenoid on the associated deluge valve. Each detection system will have its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

18.10.2 Fire Suppression

Due to the variety of risks associated with this project, a vast number of suppression system types have been designed and catered for. In all instances, the systems comply strictly with the applicable codes of practice, both locally and internationally. Each system has been designed as a fit for purpose solution which protects the equipment and personnel and does not restrict the operation. Table 18.8 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

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: 11,000 V AC resistively earthed
- Nominal frequency: 50 Hz

- Low voltage systems:
  - Motor operating voltage: 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. This solution provides greater flexibility in terms of incremental introduction as the site load increases. PFC harmonic filter banks will be installed at the following locations:

18.11.3.1 Main Consumer Substation

Three harmonic filter PFC banks are to be installed at the consumer substation:

- A single third harmonic filter bank on the left bus.
- A single third harmonic filter bank on the middle bus.
- A third harmonic filter bank together with a fifth harmonic filter bank on the right bus.

18.11.3.2 Mining Substation

Two harmonic filter PFC banks are to be installed at the mining substation:

- A single fifth harmonic filter bank on the left bus.
- A single seventh harmonic filter bank on the right bus.

18.11.3.3 Plant Substation

Two harmonic filter PFC banks are to be installed at the plant substation:

- A single fifth harmonic filter bank on the left bus.
- A single seventh harmonic filter bank on the right bus.

A power quality study will be required, before the final detailed PFC design can be completed.

18.11.4 MV Distribution

The MV distribution from the 40 MVA 110 kV/11 kV Eskom transformers and outdoor yard are included within in the general surface infrastructure. Figure 18.22 indicates the planned MV distribution.

Figure 18.22 MV Distribution

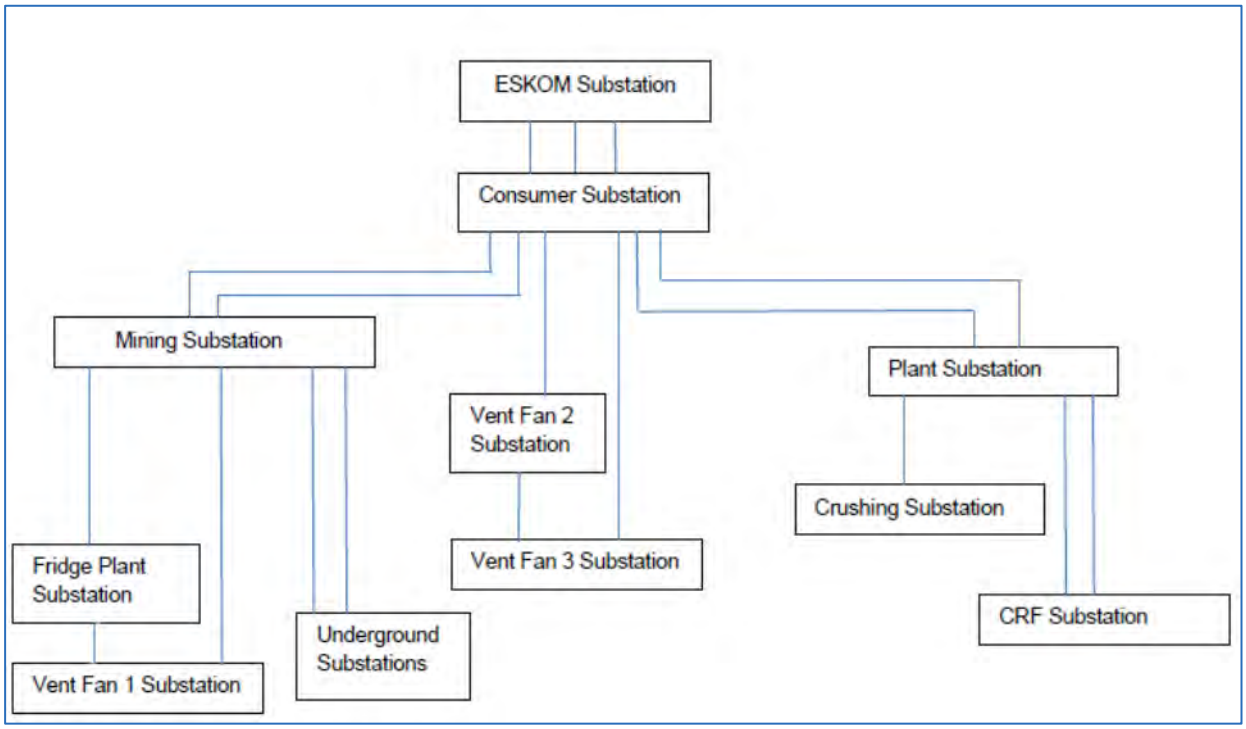


Figure by DRA, 2017.

In the consumer substation double bus 11 kV indoor switchgear is provided, for maximum load distribution flexibility. From there power is distributed on an 11 kV network to the 11 kV mining, plant, and other substations as illustrated diagrammatically in Figure 18.22. Single bus switchgear is utilised in all other substations.

18.11.5 LV Distribution

Power at 550 V has been derived from a suitable number of 630 kVA, 2000 kVA, 11 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 from11 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 550 V/400 V transformer fed from the MCC.

Only energy-efficient forms of lighting have been utilised, with facilities for person presence detection and/or automated remote switching included as appropriate for further energy savings.

18.11.7 Control System

The control system design is based on the equipment specifications and control and instrumentation design criteria, developed for the project. The control system architecture is designed around a fully distributed Programmable Logic Controller (PLC) and central Supervisory Control and Data Acquisition System (SCADA).

Two surface process control rooms have been catered for the site-wide operations.

The mining control room is located on surface near the main shaft area for the control of daily mining operations on surface and underground.

The surface process control room is located on surface at the plant area for control of the daily plant operations.

The equipment provided within these facilities is detailed in the control and instrumentation design criteria.

18.11.8 Instrumentation

The instrumentation system design is based on the equipment specifications and control and instrumentation design criteria developed for the project.

In general, with the exception of belt-scales and density meters that communicate via a Fieldbus, conventional "hard wired" type instrumentation is used in the design. Instrumentation is based on standard signal types. Instrumentation will be wired directly to weather-proof, field-mounted I/O marshalling boxes (remote I/O boxes or RIO boxes), located strategically around the plant. All RIO boxes will be connected on a fibre link back to the relevant control room.

18.12 Mining Surface Infrastructure

18.12.1 Surface Rock Handling

The rock handling area includes the following main areas and equipment:

- Headgear load and transfer conveyors,
- 6,000 t emergency ROM and 6,000 t waste rock temporary stockpiles,
- Permanent ore, low-grade and waste rock stockpiles,
- 5,200 t ROM silos,
- Rock handling mobile fleet,
- Waste rock crushing.

18.12.1.1 Process Description

The rock handling system handles both ROM and waste rock streams. The waste rock stream has a larger maximum material lump size compared to the ROM stream, resulting in different technical design perimeters.

The shaft headgear bin feeds a wide sacrificial conveyor at a design rate of 1,350 tph. The sacrificial conveyor is deliberately designed wider to allow for a slower belt speed that assists with the extraction of scrap metal by over-belt magnets and further assists with accurate sampling by the cross-belt hammer sampler.

The sacrificial belt feeds a splitter transfer station, intended to split ROM and waste streams. From the transfer tower ROM is fed to the ROM silos, and waste is fed to a waste transfer conveyor. The waste transfer conveyor feeds another splitter transfer station to allow for a dual stockpile system. The dual stockpiles allow for emergency ROM storage if the ROM silos are not available, but could also allow for two waste rock stockpiles. Two waste rock stockpiles allow for equipment and personnel safety during stockpile removal operations. Rock is dumped on one stockpile, while removal operations continue on the other stockpile.

Silos provide storage and buffering capacity. Silos were 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 18.23 indicates the surface rock handling layout.

Figure 18.23 Surface Rock Handling

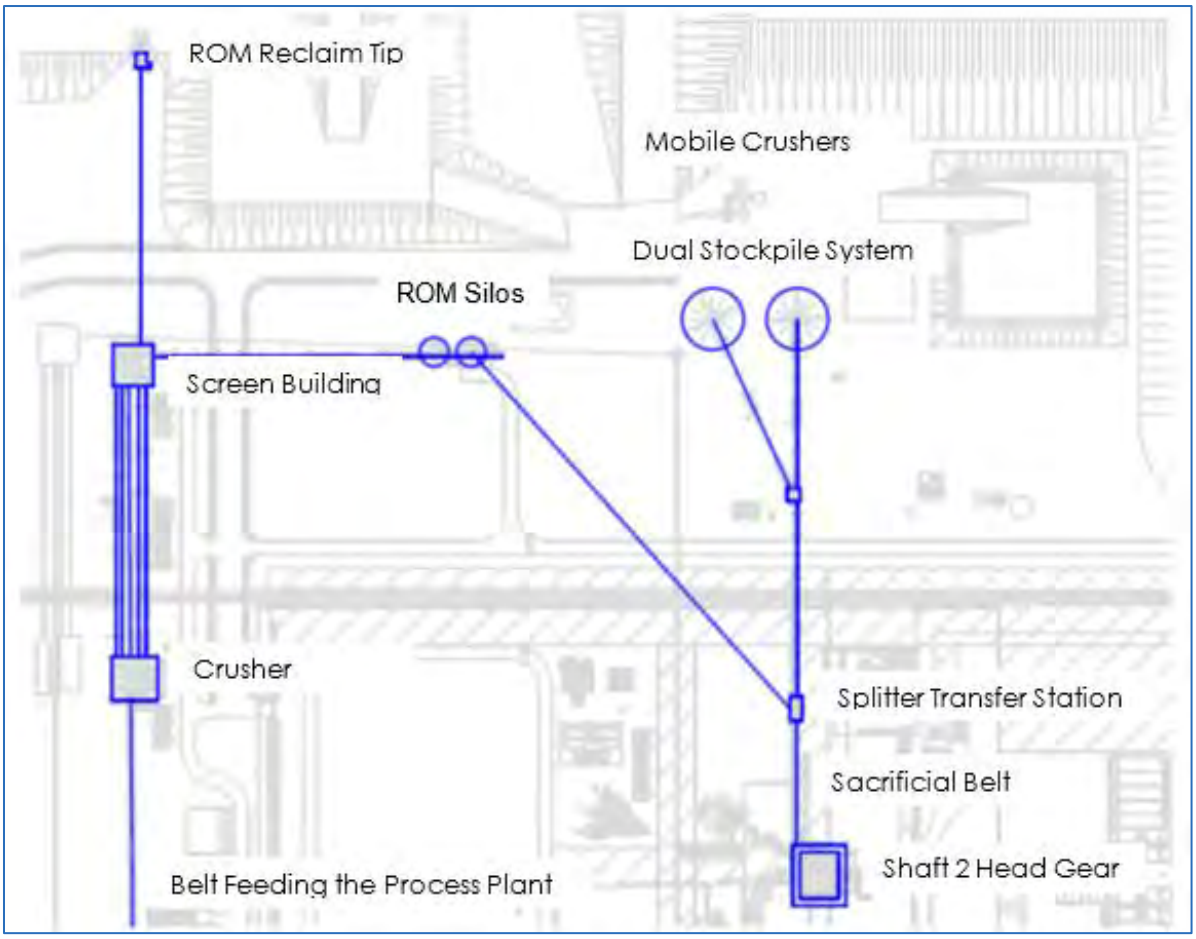


Figure by DRA, 2017.

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,
- 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 a 4 Mtpa concentrator plant which is expandable to an 8 Mtpa plant. The layout considered the constructability of the 4 Mpta and 8 Mpta cases allowing for minimal disruption to the then-already operating mine shafts and 4 Mpta plant.

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.

18.12.1.4 Waste Rock Crushing

The waste rock crushing material will be used for cemented rockfill for underground usage and G4 type of material for surface usage. A mobile waste rock crusher has been allowed for and suitably sized at 250 tph to deliver the required CRF and G4 quantities. The crushing and screening circuit is mobile and diesel-electrically driven to provide for early establishment on site, prior to availability of electrical infrastructure.

The mobile crusher that has been allowed for could produce both types of materials required for CRF and G4 quantities. The crusher will be fed and products removed using the mobile rock handling fleet previously described. The CRF stockpiles and underground transfer shafts are in close proximity to the waste rock crusher to allow for rapid delivery and optimised haulage cost.

Prior to the availability of the concentrator plant tailings for paste backfill of the underground mine, CRF will be utilised. This method requires the crushing of waste rock on surface and its return underground. To meet the required CRF quantities requires the establishment of the crusher 22 months prior to initiation of CRF consumption. Figure 18.24 indicates the grading analysis of the material that could be produced by the mobile crusher allowed for.

Figure 18.24 Grading Analysis of Waste Rock Material

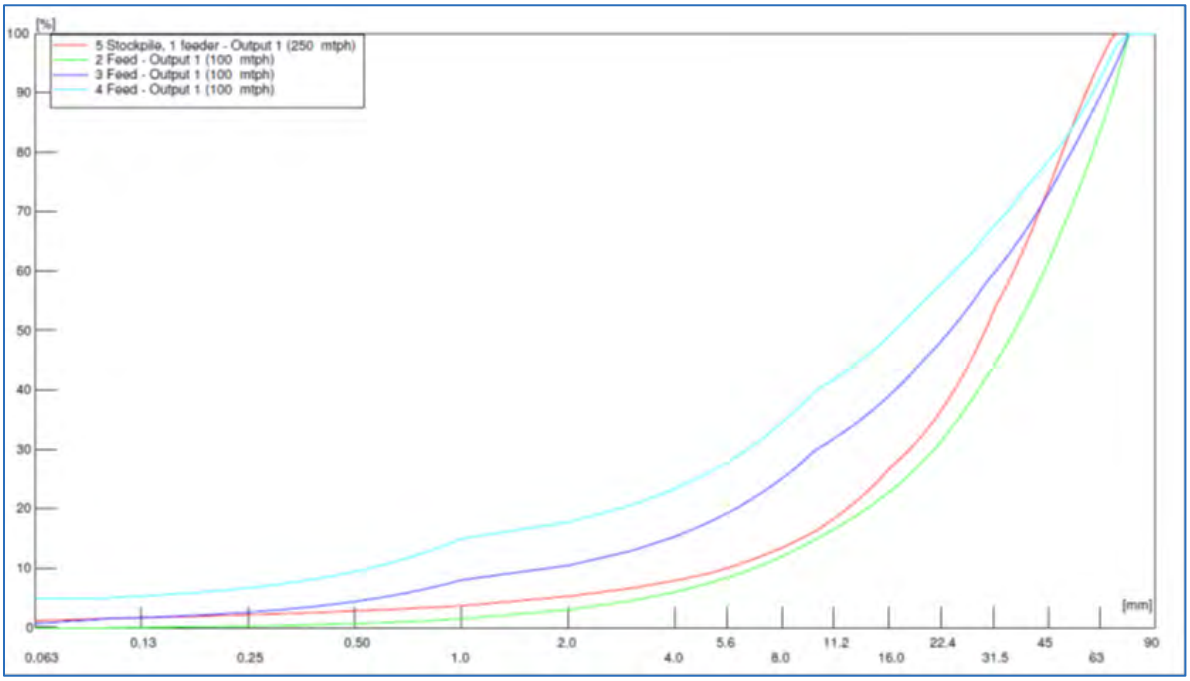


Figure by DRA, 2017.

18.12.2 Explosives Handling and Distribution

The existing surface magazine will be utilised in conjunction with the emulsion system. The emulsion system is based on the sum of all costs of the various engineering disciplines that are affected, i.e. earthworks, civil, structural steel, plate work, mechanical, piping and electrical. 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

Emulsion is delivered on surface and stored in emulsion storage silos. Four tanks are required with a storage capacity of 35 m³ each. Figure 18.25 and Figure 18.26 show the layout of these silos.

Figure 18.25 Emulsion Storage Tanks - Plan View

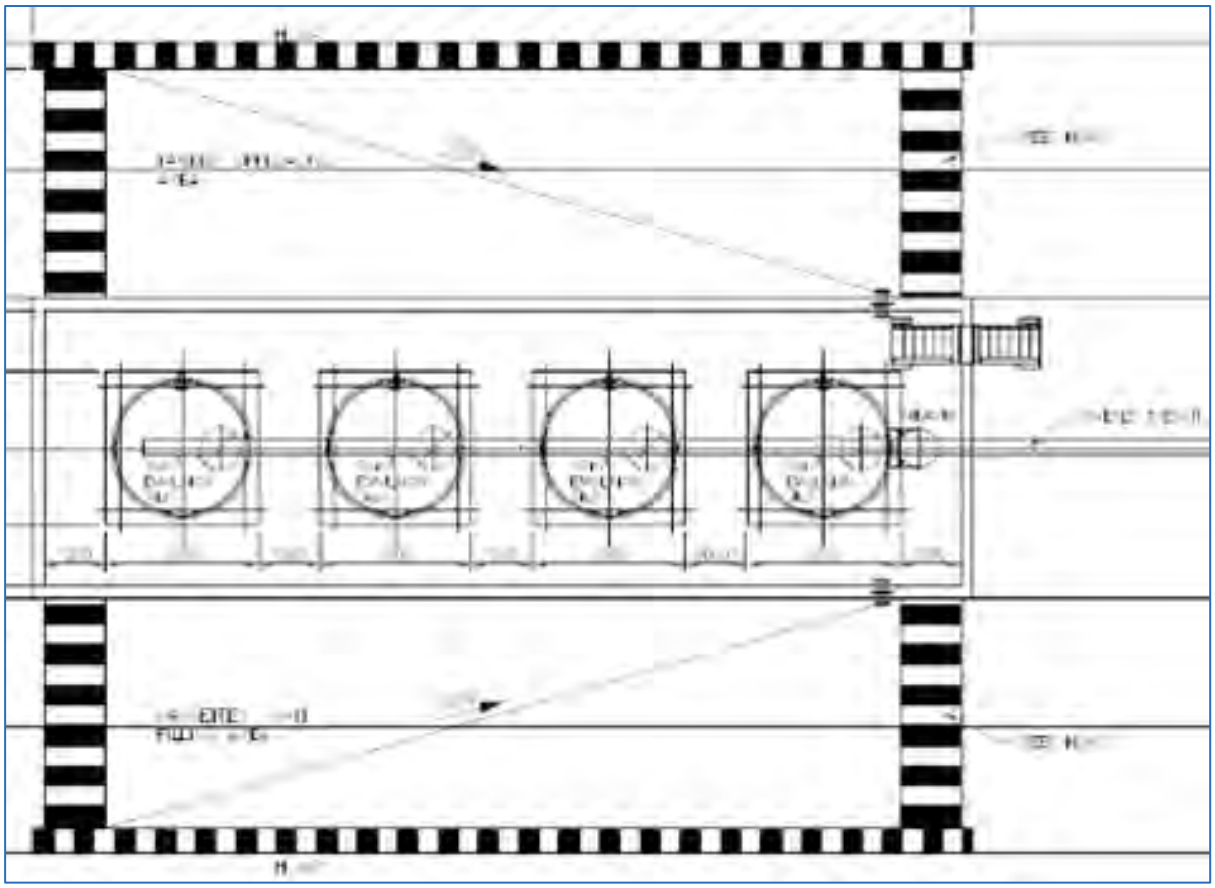


Figure by DRA, 2017.

Figure 18.26 Emulsion Storage Tanks - Section View

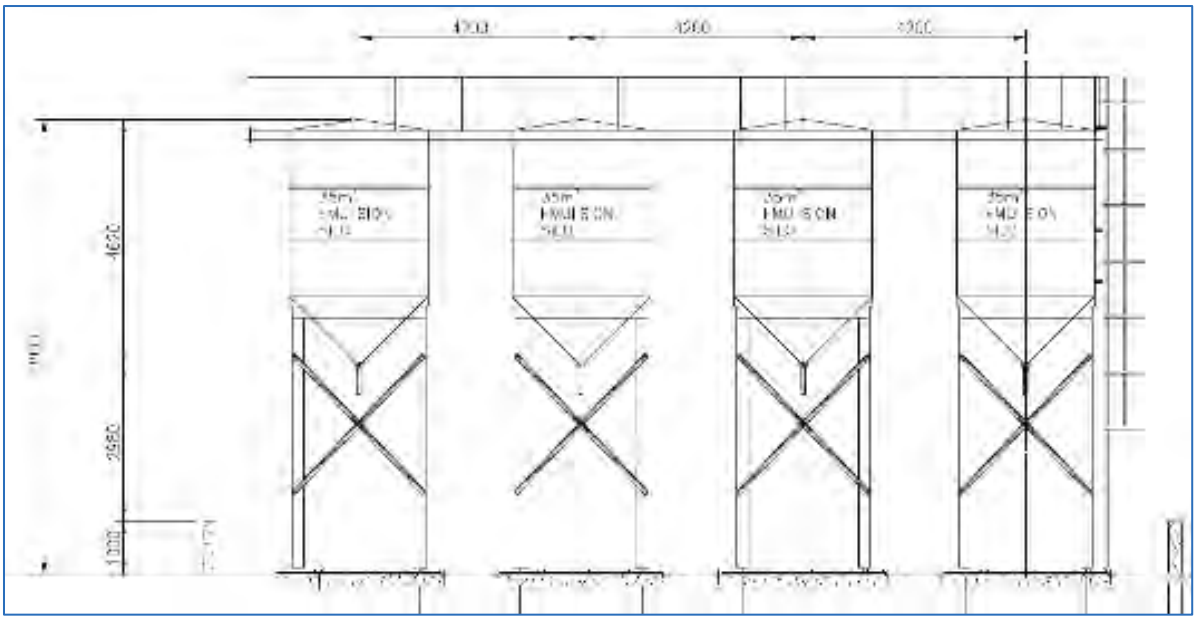


Figure by DRA, 2017.

Purpose-built cassettes will be filled from these silos and transported underground.

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 18.27 to Figure 18.29 show the layout of the offloading area and indicate the dock leveller, the pallet stacking and the scissor lift.

Figure 18.27 Explosives Offloading Facility – Plan View

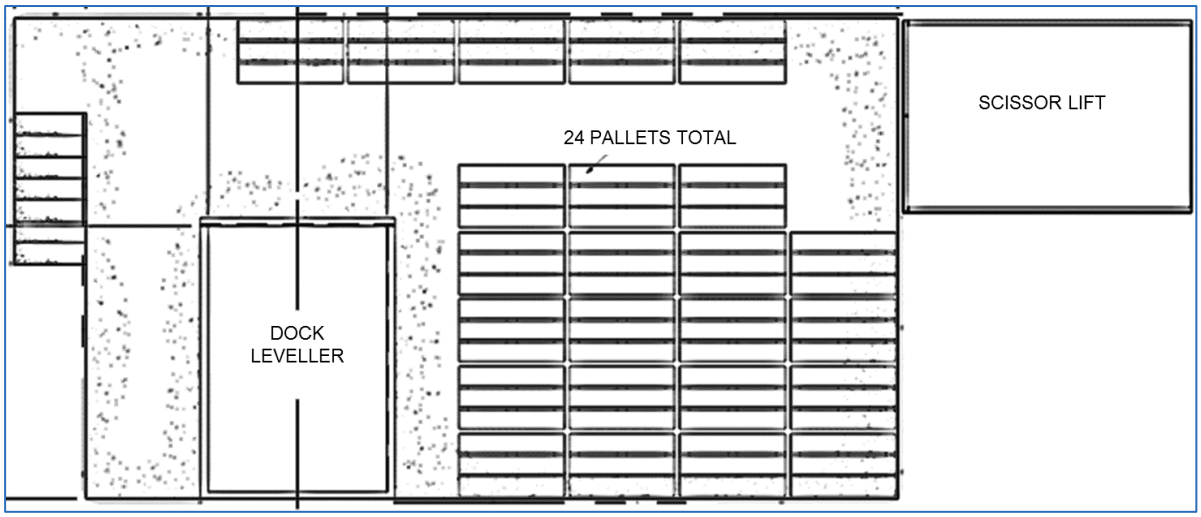


Figure by DRA, 2017.

Figure 18.28 Explosives Offloading Facility – Front View

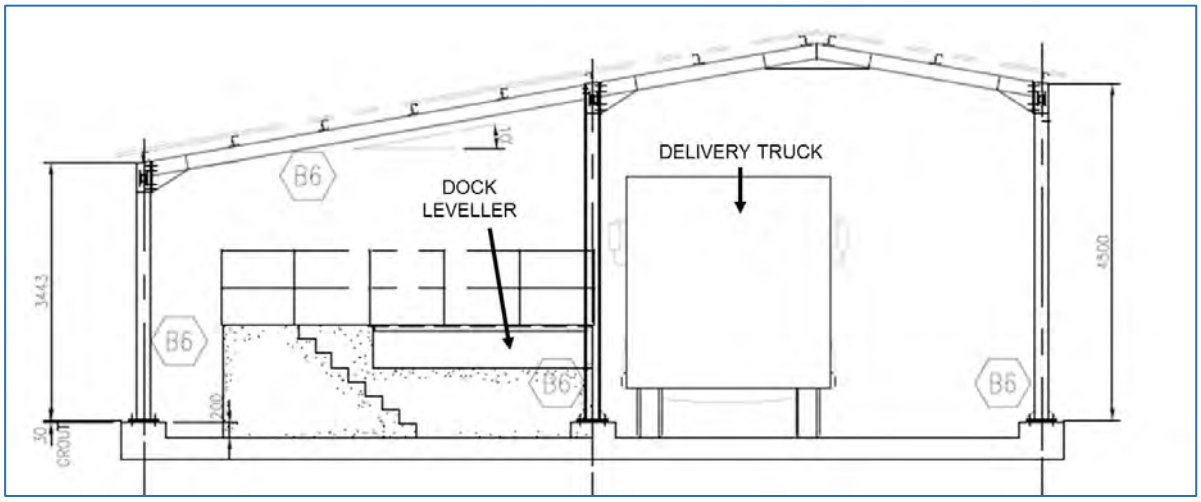


Figure by DRA, 2017.

Figure 18.29 Explosives Offloading Facility – Side View

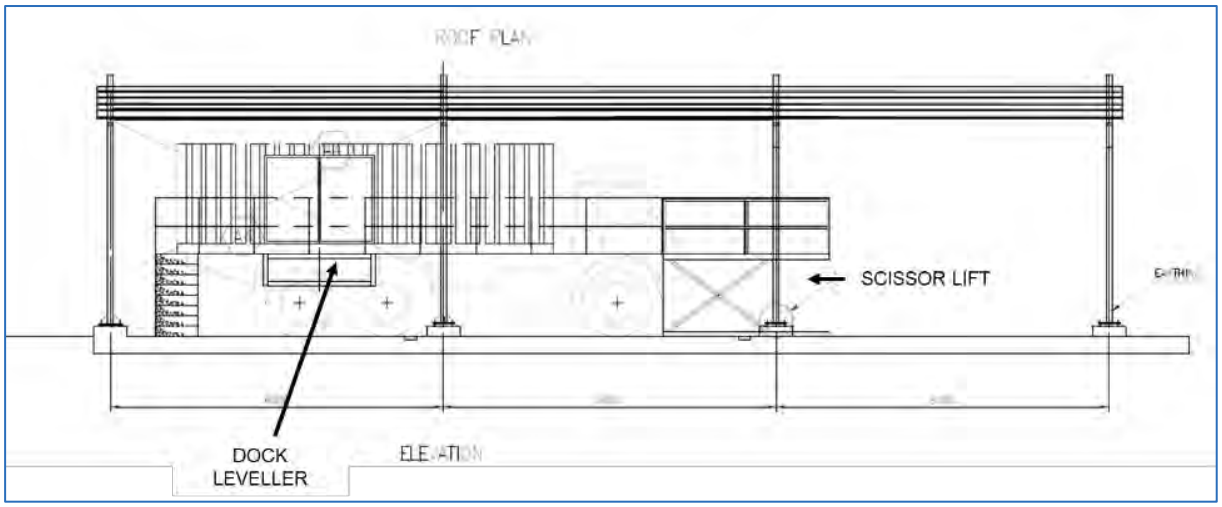


Figure by DRA, 2017.

18.12.3 Fuel and Lubrication

The fuel and lubrication system is a shared service between the surface and mobile underground fleet.

A separate fuel system is included at the emergency generators to supply fuel to the generators only.

The surface vehicles will refill at the depot. Underground vehicles will refill at refuelling stations. Fuel will be piped to the three main levels, and lubes will be transported in lube cassettes through the shaft material transport system.

Figure 18.30 illustrates the plan layout of the surface refuelling and storage depot and indicates the lubricants offloading and dispensing. Figure 18.31 is the same layout, indicating the diesel receiving and dispensing.

Figure 18.30 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Lubes Receiving and Dispensing

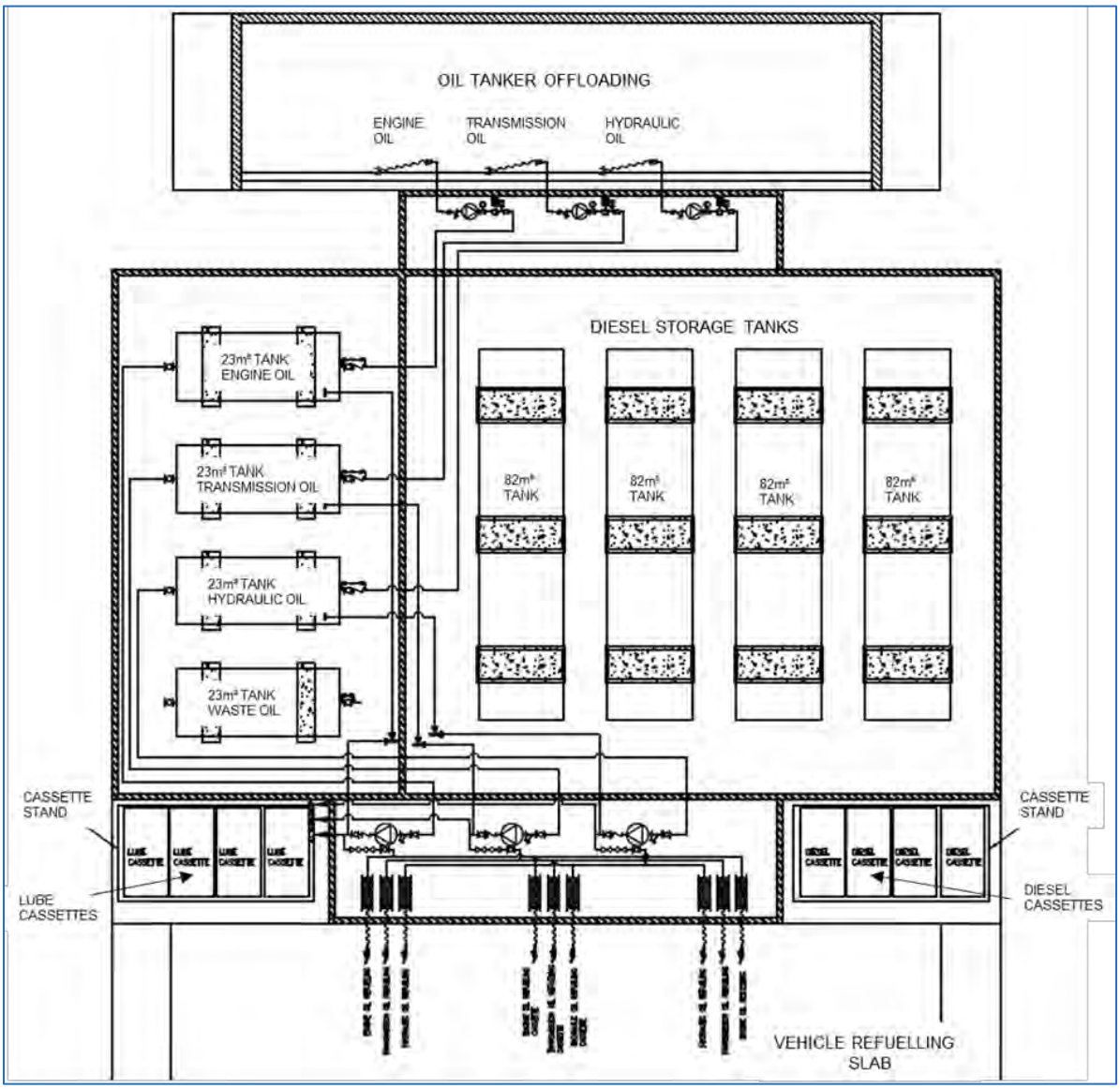


Figure by DRA, 2017.

Figure 18.31 Surface Fuel and Lube Receiving, Storage and Dispensing Depot – Diesel Receiving and Dispensing

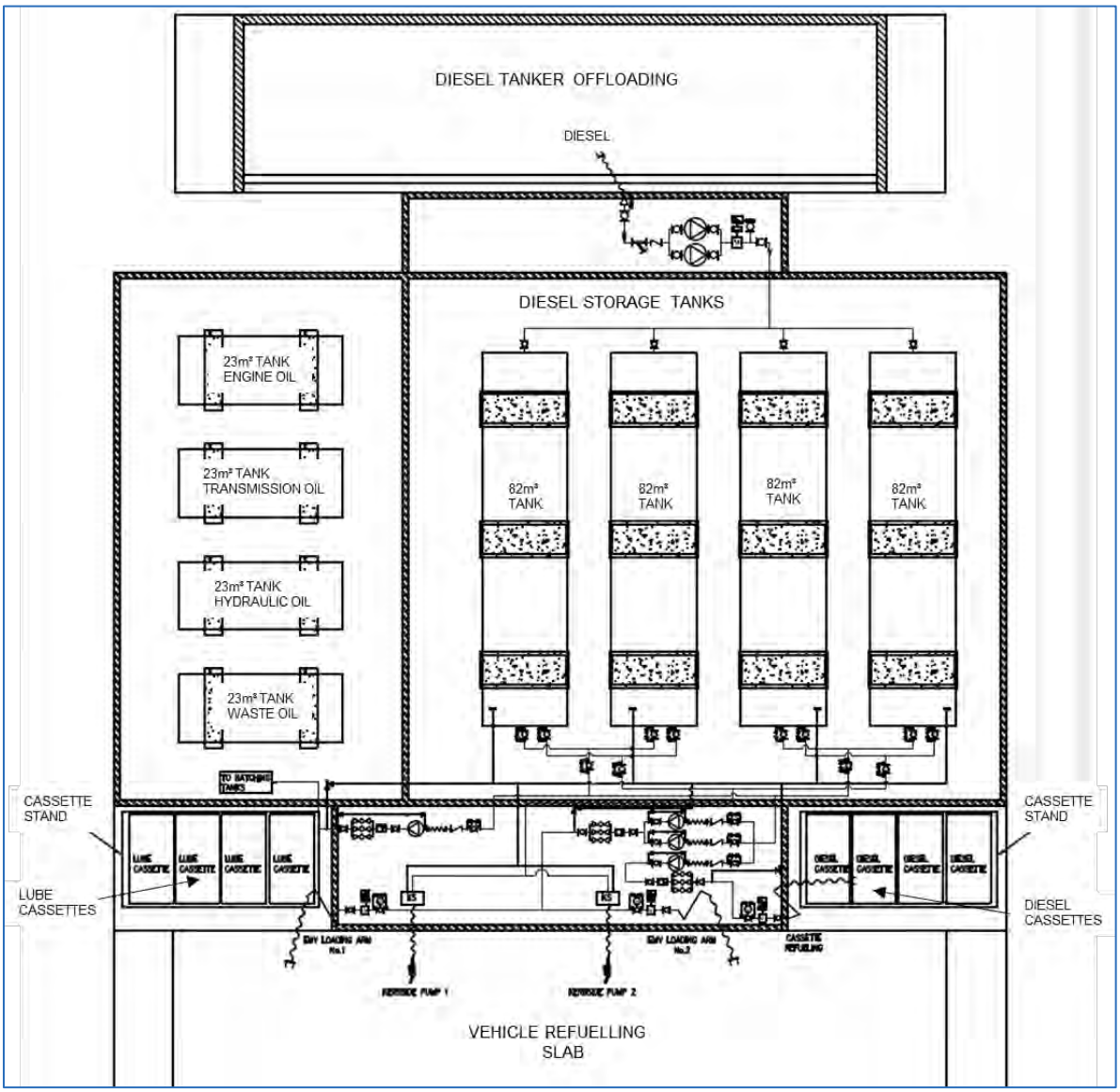


Figure by DRA, 2017.

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 transferred to the underground fuel storage tanks using "cassettes" on a continual basis – no batching will take place via piping.

Three grades of oil will be used

- Engine Oil
- Transmission Oil
- Hydraulic Oil

Surface storage tanks were designed with sufficient fuel storage capacity to operate the mine for approximately 30 days (per grade of oil). The tank sizes per grade of oil are as follows:

- Engine Oil: - 1 x 23 m^3 tank
- Transmission Oil: - 1 x 23 m^3 tank
- Hydraulic Oil: - 1 x 23 m^3 tank
- Waste Oil: - 1 x 23 m^3 tank

18.12.4 Concrete and Shot Crete Facility and Distribution System

Shot Crete and concrete will be mixed in a batch plant located on surface.

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 18.32 show the layout of the surface shotcrete and concrete batch plant.

Figure 18.32 Surface Shotcrete and Concrete Batch Plant

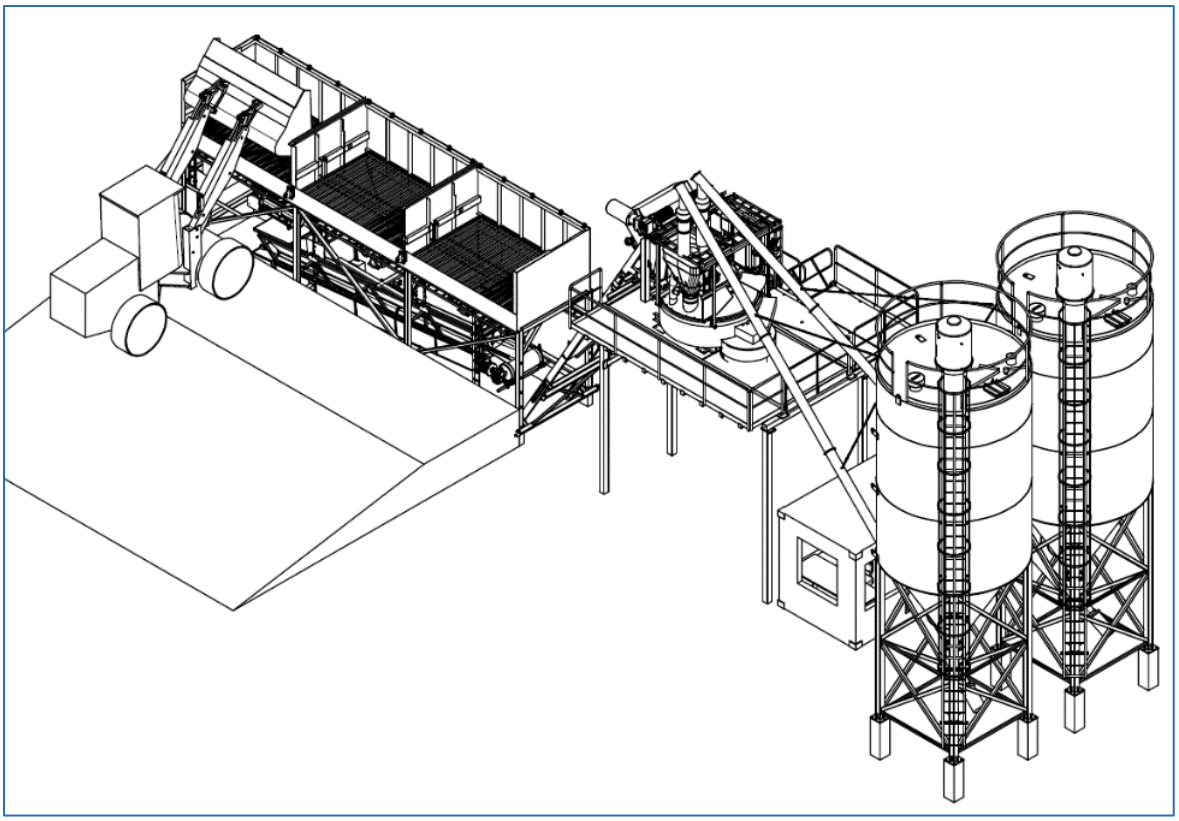


Figure by DRA, 2017.

The batch plant is equipped with strong compulsory planetary mixer 1,000 L, which can mix all kind of concrete recipes including steel and synthetic fibres. The mixer can make high quality shotcrete, ready-mix, mortar and earth dry concrete. Production capacity is from 10 m^3/h up to 40 m^3/h depending on the mixing time, used materials and recipe.

The plant includes three aggregate silos which are filled by wheel loader.

Cement is dosed by screw from the cement silo.

There is an optional additive system that is 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 weight and dose fibres automatically according to recipe. Fibres are mixed in the mixer to ensure homogeneous concrete and the best spraying result.

Optional micro silica dosing system makes possible to use even more specific concrete recipes. Micro silica is dosed from big bags to scale.

The concrete is mixed into a wet sludge and discharged into one of two boreholes feeding the underground receiving areas located in dedicated excavations on 850 and 950 Levels.

18.12.5 Vent Raise and Fans

The ventilation system is designed as an exhausting pull system. The two main intake shafts (Shaft 1 and Shaft 2), located at the centre of the mining district, will provide fresh air, while Ventilation Raise 1 (near the intake shafts), Ventilation Raise 2 (in the north), and Ventilation Raise 3 (in the south) will be exhausts.

The overall ventilation system for the project uses two fresh air intake shafts (10 mø and 7.5 mø) and three primary ventilation raises (6 mø each).

Fresh air is reticulated through the underground mine via a pull (exhaust surface fans) system. Each ventilation raise will have three similar exhaust centrifugal fans located on surface as shown in Figure 18.33. The smaller intake shaft (7.5 mø) will provide chilled fresh air (~50% of the total intake air cooled) from the surface-located BAC. Airflow quantity for the mine at full production is calculated as 1,200 kg/s when the air is distributed to the underground operation via the two intake shafts.

Figure 18.33 Main Ventilation Fans Layout

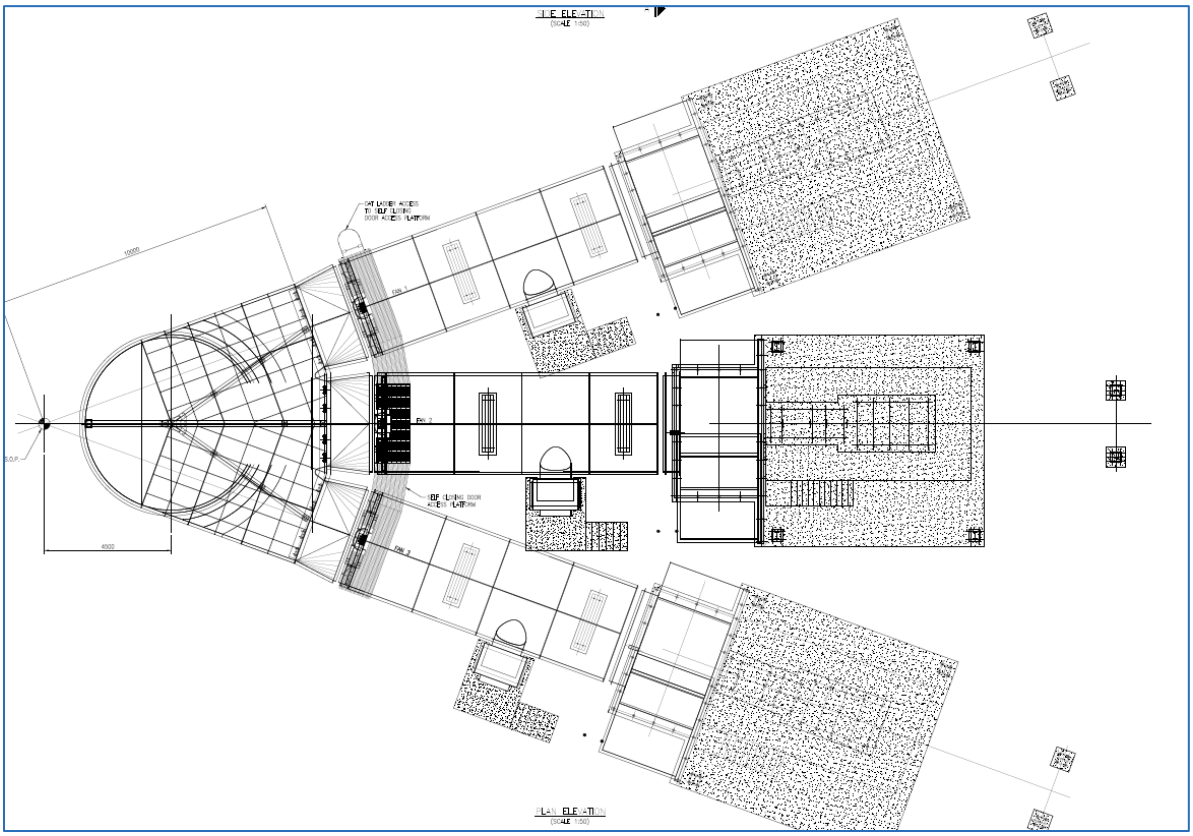


Figure by DRA, 2017.

18.12.6 Backfill Plants

Ivanplats requested Golder Associates Africa Ltd. to perform a feasibility-level engineering study for the Cemented Rock Fill (CRF) and Cemented Paste Fill (CPF) Plants (collectively referred to as Backfill) for the Platreef Project. Refer to Figure 18.34 for the surface position of the back-fill plant in relation to the rest of the mining and process plant surface infrastructure

CRF would be the most suitable method of backfill for the initial stages of mining (also referred to as the "Interim Backfill System"), prior to the construction of a concentrator capable of supplying tailings to be used in the formulation of backfill. CPF was selected as the as the "Long-Term Backfill System" that would serve the life of mine backfilling requirements.

Figure 18.34 Shaft Area Services with Backfill Plants

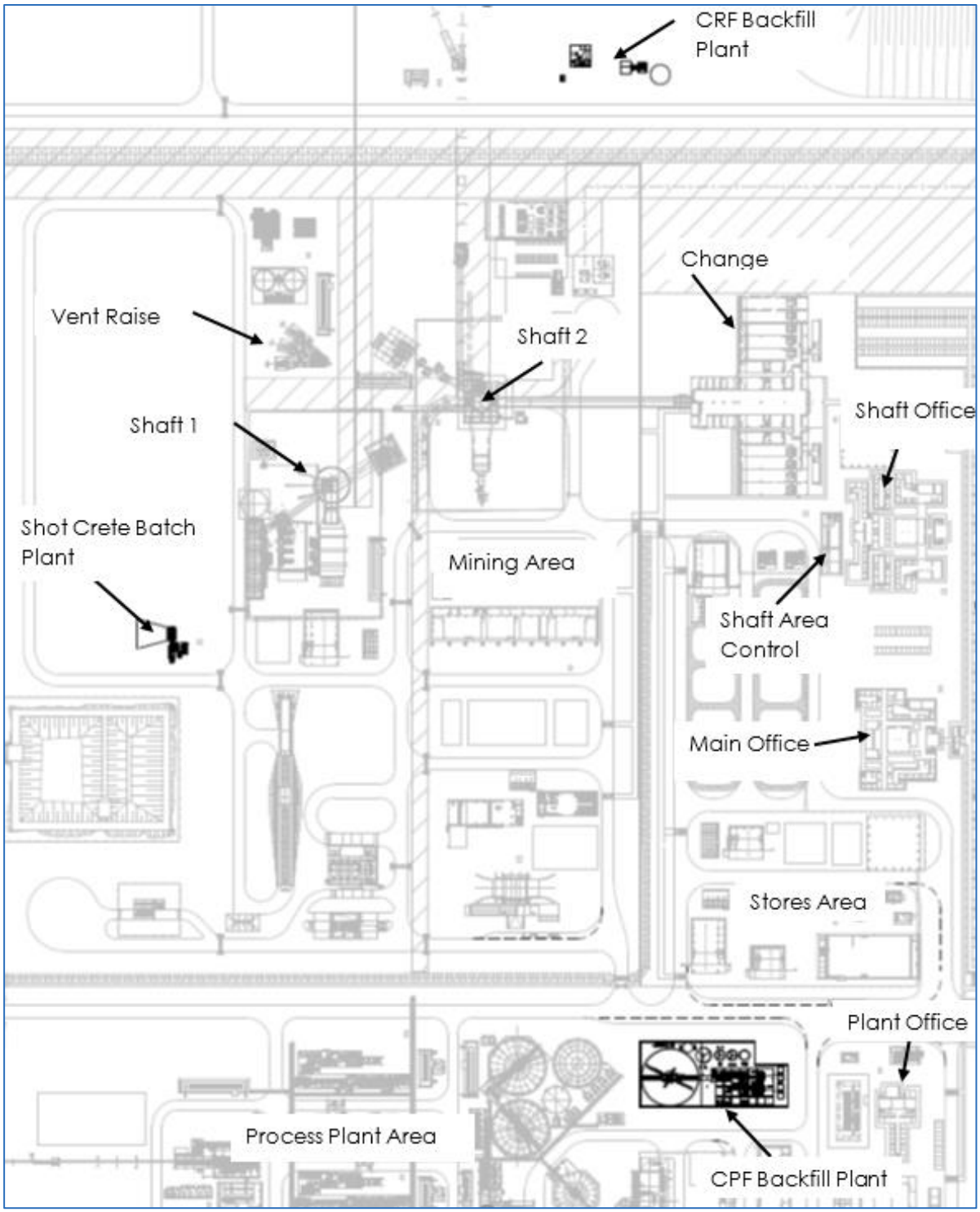


Figure by DRA, 2017.

18.13 Surface Logistics

18.13.1 Traffic Flow Study

Simulation Engineering Technologies (SET) was engaged to construct a simulation model representing the traffic flow into and out of the mine.

The purpose of the simulation study was to provide a check of the design parameters to identify any potential design bottlenecks or risk areas. The surface infrastructure was modelled to determine if the planned infrastructure is sufficient to support surface vehicle operations in terms of arrivals, movement and parking especially during peak periods. The base case was based on a two-way stop at the main road (N11) intersection.

The surface infrastructure simulation indicated that for the base case, inbound traffic is constrained and results in vast queuing up to the intersection and onto the N11. This impacts mine and N11 traffic as well as personnel arrival times and vehicle travel times.

Outbound traffic flow is also constrained and results in vehicle queuing, especially for cars. The planned parking capacity for arriving delivery trucks at the main gate is sufficient. Furthermore, there are minor parking capacity concerns for certain delivery trucks of which the most important is concentrate deliveries. The weighbridge also experiences intermittent queuing.

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 vast queuing. The two-way stop traffic rules constrain traffic flow out of the mine.

Cars account for 96.5% of personnel vehicle arrivals and therefore have the most significant impact on 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 found 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.

18.13.2 Surface Fleet

Table 18.9 lists all the surface fleet equipment that has been allowed for.

Table 18.9 Surface Fleet Equipment

Type	Description	Capacity	Area	Qty
Telehandler	Telehandler MHT - X 14,350	35 t	2000,3000	3
	Telehandler MRT - X 32,55 Privilege	5,500 kg	6000	1
Site Forklift	M-X 70-2 Rough Terrain Forklift	7,000 kg	2000,6000	4
	MH 20 Rough Terrain Forklift	2,000 kg	6000	1
Reagent Forklift	ME 430 AC Electric Forklift	3,000 kg	3000	1
Skid Steer Loader	Gehl 4240E Skid Steer Loader	2,100 kg	2000, 3000	4
Waste Skip Tractor and Trailer	Massey Fergusen 440Se Xtra 61 kW, two wheel drive tractor with airbrake compressor and sun canopy		2000, 3000, 6000	1
	AIM Power X 6 t Chassis Trailer with airbrakes and Road Ordinance electrics		2000, 3000, 6000	1
	AIM Power X Scowback Container with tailgate		2000, 3000, 6000	15
Front End Loader	CAT Large Wheel Loader 990 K		6000	2
	CAT Medium Wheel Loader 950H		3000	2
Articulated Trucks	CAT Three Axle Articulated Truck 745C	41 t	6000	5
Excavators	CAT Medium Excavator 330D2L	30,305 kg	6000	1
	CAT Large Excavator 340D2L	40,000 kg	6000	1
	CAT Large Excavator 374F L (2017)	70,000 kg	6000	1
Dozers	CAT Medium Dozer D7R		6000	2
Truck with High up	16 t Flatbed Truck with High up		6000	1
Motor Grader	CAT M Series Motor Grader 120M		6000	1
Backhoe Loader	CAT Backhoe Loader 416F2		6000	1
Roller	CAT Vibratory Soil Compactor CS64B		6000	1
Water Bowser	15 000 l Water Bowser		6000	1
Diesel Bowser	8000 l Diesel Bowser		6000	1
Ambulance	ECW Ambulance Two Bed		6000	1
	ECW Ambulance Four bed		6000	1
Fire Truck	Medium-Size Fire Truck 4 x 4	3,000 l	6000	1
	Large Fire Truck 6 x 4	8,000 l	6000	1
Rescue Vehicle	RIV Land Cruiser		6000	1
Rescue Platform	TL Hydraulic Platform		6000	1

18.14 Security and Access Control

18.14.1 Control

18.14.1.1 Access Control

The access control system is a software-based system with a database which will be accessed on the network on a fibre link from the server to the field switches via the main controllers and to the sub-controller. Every door will need a controller with entry and exit of outdoor and/or indoor biometric readers in buildings, boom gates and turnstile.

The system works as a dual tag system, meaning both fingerprints and a card reader will be required to grant a personnel entry or exit.

The access report database will be stored on the server with Time and Attendance and its software integrated in the system. All the personnel who will be accessing the mine will be enrolled on the fingerprint system and be issued with a card.

The CCTV and access control system will be integrated into all main entry and exit points, allowing for the ability to view the video recordings of a personnel when badging on the biometric/card reader's access control system.

18.14.1.2 CCTV

The closed-circuit television (CCTV) system is an Internet Protocol (IP) based network system with a 10 GB backbone design on a fibre link from the server to the field switches. This includes the server and storage for every 50 cameras with 30 days live and 30 days archive recordings respectively (total of 60 days recordings). All switches will be power over Ethernet to cater for all the IP cameras.

There will be outdoor static cameras for all the outdoor areas, including workshops and for overview coverage, while there will be indoor fixed cameras for all indoor offices and buildings. High-speed dome cameras will be used mainly for parameters and big overview coverage.

The CCTV and access control system will be integrated at all main entry and exit points with the ability to view the video recordings of personnel when badging on the biometric/card reader's access control system.

18.14.1.3 Security Alarm

The security alarm system design is easy to service and maintain. This includes the monitoring of the fire escape doors with door contacts and passives (PIRs) to be located in a logical manner in the buildings. The system will include and integrate with the following:

- Radio and GPS link
- 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.

18.14.2 Fencing and Access Control Buildings

Fencing

The mining lease area is fenced off, as shown in Figure 18.36, 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 have been 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 mine area perimeter and internal fence lines are shown in Figure 18.35.

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. Refer to Figure 18.36 for details.

Access Control Points

Various access control buildings have been allowed within the mine area. Table 18.10 lists them all and Figure 18.37 indicates their location.

Table 18.10 Access Control Buildings

AREA 2000			
WBS Number	Drawing Description	Building Description	Area/m²
257805	Shaft Area Gatehouse	Gate House with turnstiles controlling the entrance of pedestrians into the shaft area.	110
AREA 3000			
341105	Plant Area Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the plant and store area as well as vehicles that want to enter the mine and store area. The weighbridge control room also forms part of this building.	80
AREA 6000			
618105	Main Entrance Vehicle Gate House	Main vehicle access control at the main entrance to the mine controlling all vehicles that want to access the mine including deliveries and visitors.	30
618128	Main Entrance Pedestrian Gate House	Gate House with turnstiles controlling the entrance of visitors to the mine and employee access to the engagement centre.	360
618129	Bus Stop Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the mine area arriving with buses and taxis.	90
618130	Main Office Gate House	Gate House with turnstiles controlling the entrance of pedestrians into the main office area.	50

Figure 18.35 Mine Area Perimeter and Internal Fence Lines

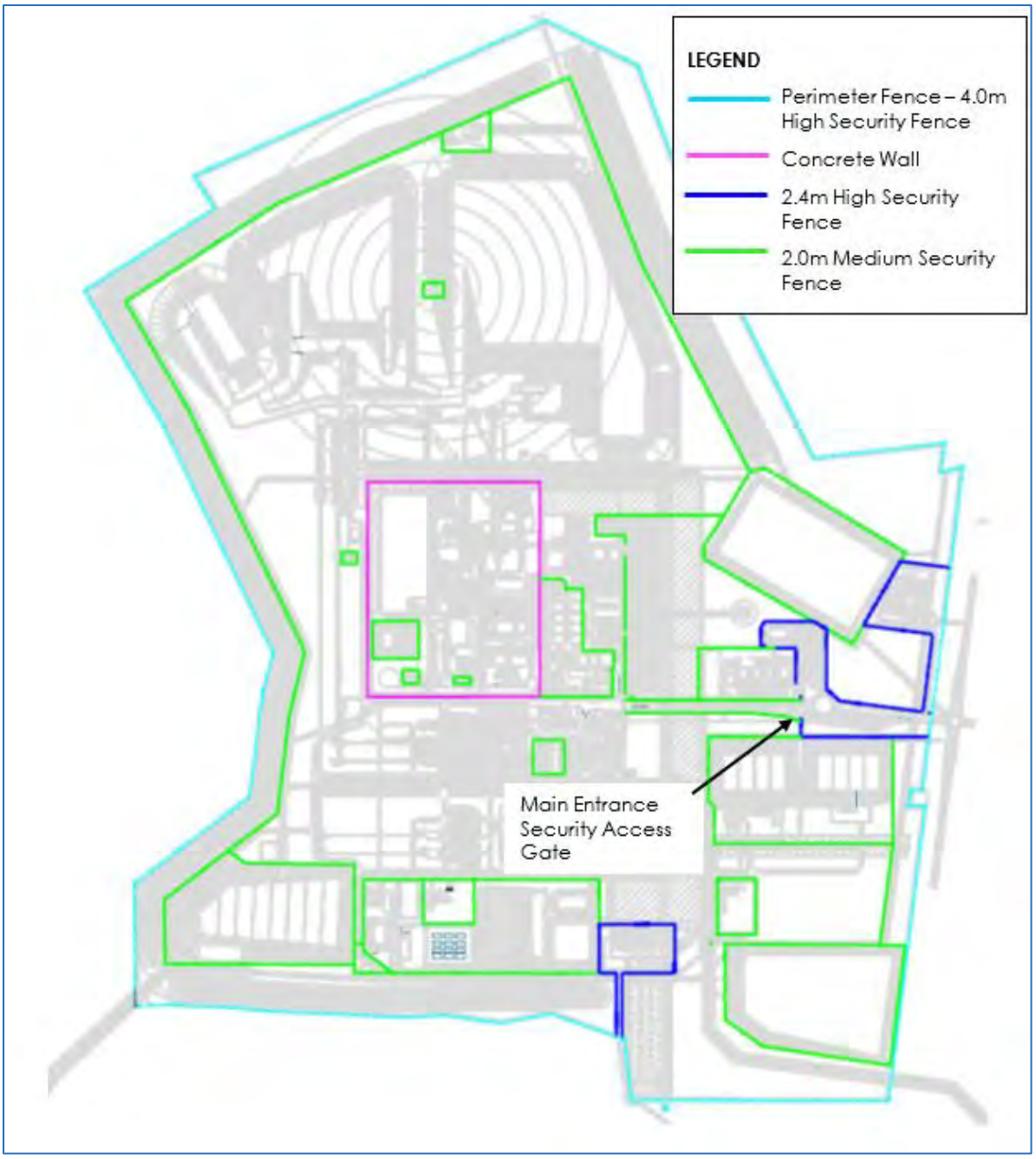


Figure by DRA, 2017.

Figure 18.36 TSF Perimeter Fence Line

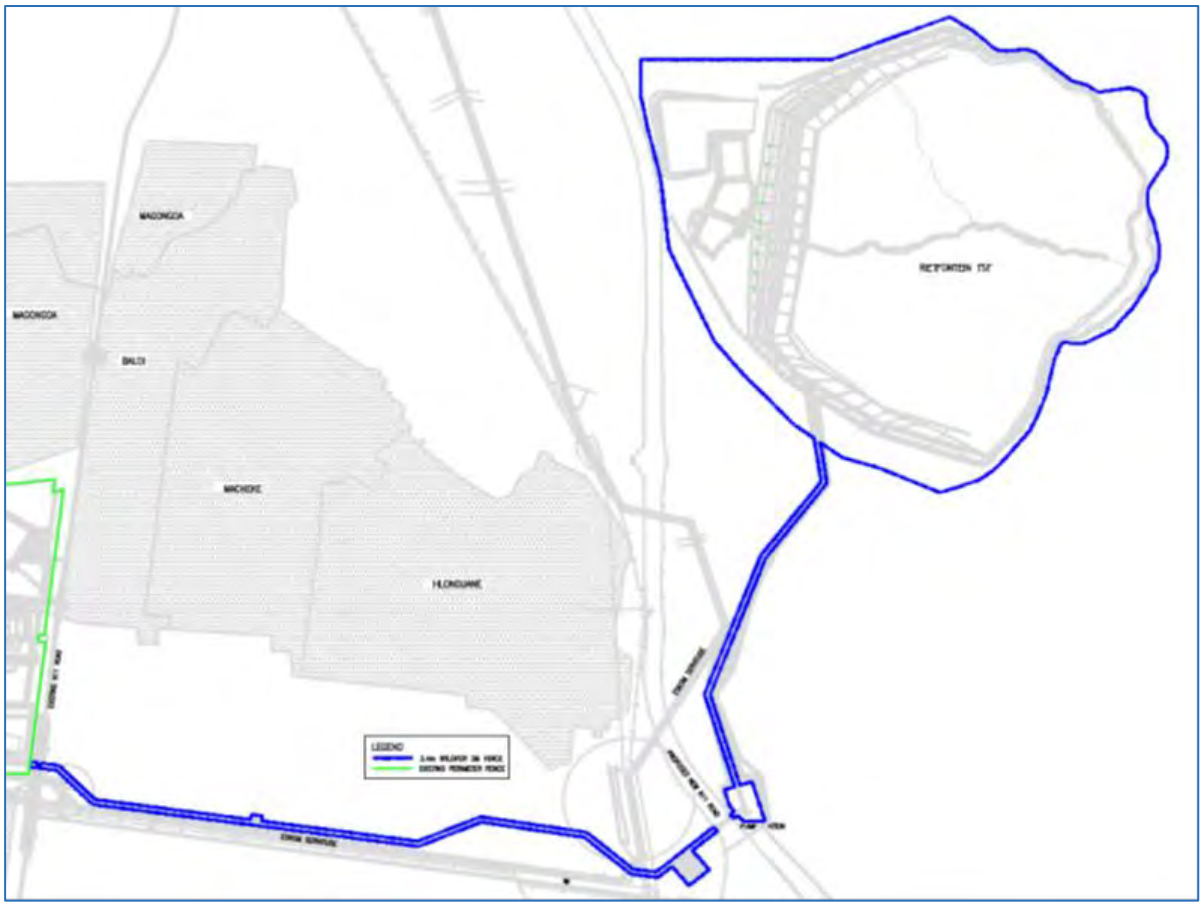


Figure by DRA, 2017.

Figure 18.37 Mine Area Access Control Points

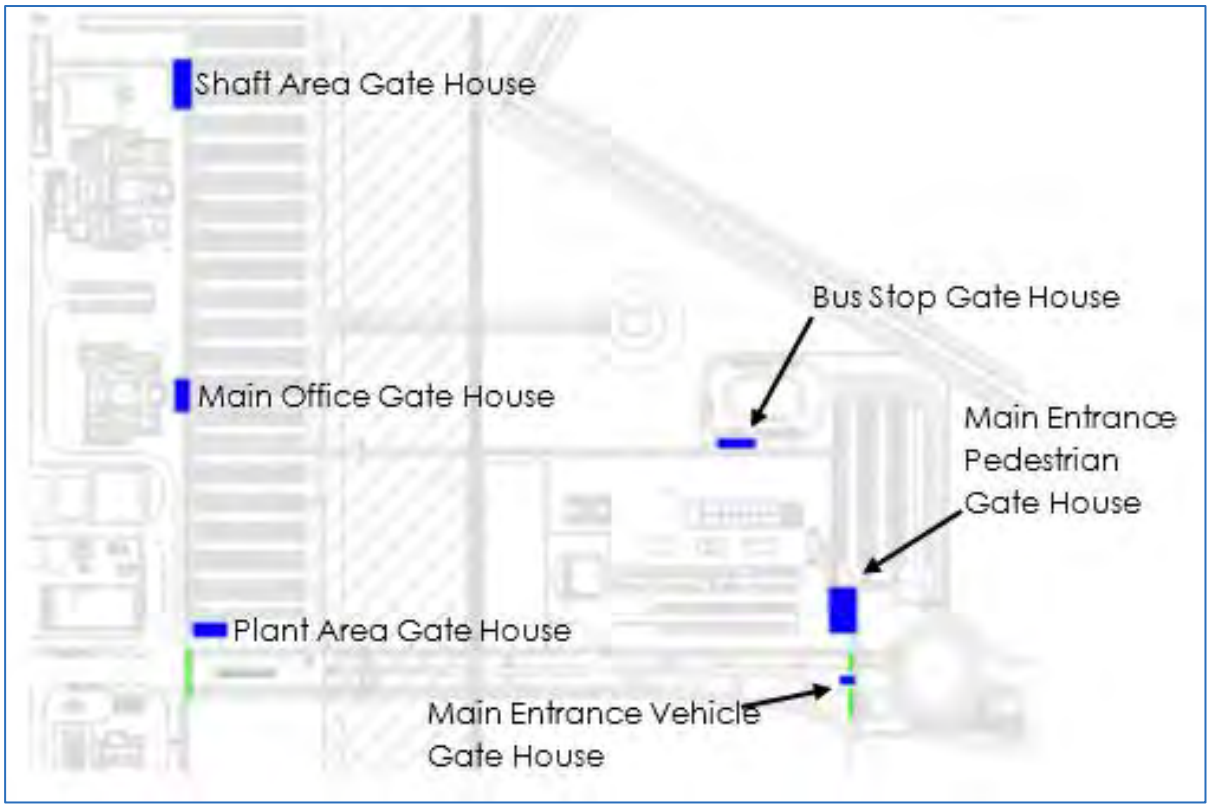


Figure by DRA, 2017.

18.15 Waste Management

18.15.1 Introduction

The Platreef Project will generate a number of general and hazardous waste streams that will be managed on site. As part of the proposed mining operation, the following waste-related facilities will be constructed to manage the various waste streams:

- Hazardous waste storage area for the safe and compliant temporary storage of various hazardous waste materials;
- General waste storage area for temporary storage of sorted material before disposal;
- Waste tyre storage area for the short- to medium-term storage of used tyres;
- Recycling/Material Recovery Facility (MRF) for sorting and separate storage of recyclable materials;
- Oil traps and oil storage tanks for oil interception/recovery, and temporary storage of used oil;
- Salvage yard/scrap yard for temporary storage of scrap metal and re-usable mechanical parts;

- One stormwater pond for capture and storage of contaminated water generated on site;
- Sewage Treatment Plant to treat black water produced on site;
- Mine Filter Plant to treat water to be used in mining process (waste sludge generated at the WTP will be disposed to on-site landfill or licensed hazardous disposal, pending classification);
- Plant Filter Plant to treat water to be used in concentrator process (waste sludge generated at the WTP will be disposed to on-site landfill or licensed hazardous disposal, pending classification); and
- General Waste Disposal Facility (landfill), including leachate pond and sediment trap, for on-site disposal of general waste.

Within the central mining area, small intermediate solid waste transfer open storage yards are provided for general waste and salvage. Associated with the flotation process is a tailings disposal sump which permits mineral sludge to be transferred via the tailings delivery line to the TSF.

Some of the above facilities require a Waste Management Licence (WML) in terms of the National Environmental Management Act, 2008 (Act 59 of 2008) (as amended). The initial application form for the WML was submitted to the competent authority for waste licensing, namely the Department of Environmental Affairs (DEA) on 15 April 2013. The EIA process was undertaken and the final EIA report submitted to the DEA for decision-making on 15 January 2014. The conceptual designs of the waste facilities were approved by the DWS (based on a review of the designs and subsequent amendments made to the designs); on 13 March 2015, the DEA issued the WML for the project.

A variation application for the WML has been submitted to the Department of Mineral Resources (DMR), the new competent authority for waste licensing on mine sites (in April 2016). The application includes requests for amendments to the licence, specifically with regard to correcting co-ordinates of the waste facilities. Platreef is awaiting the amended licence from the DMR.

Since the initial WML application was submitted in 2013, prior to the amendment of the National Environmental Management Waste Act (NEMWA) to include mine residue stockpiles and deposits, the issued WML does not include the waste rock facilities and TSF. These facilities are authorised through the approved EMP in terms of the Mineral and Petroleum Resources Development Act (MPRDA), 2002 (Act 28 of 2002) and the Water Use Licence to be issued by the DWS.

18.15.2 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), 1998 (Act 107 of 1998);
- Environmental Impact Assessment (EIA) 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;
- Principles:
  - Integrated Waste Management Hierarchy;
  - Extended Producer Responsibility;
  - Duty of Care; and
  - Cradle to Grave.

18.15.3 Description of Waste Facilities

In the waste area allowance was made for the following waste facilities:

- General Waste Landfill
- Composting Plant
- General Waste Storage and Recycling Area
- Hazardous Waste Storage Area
- Salvage Yard
- Waste Tyre and Conveyor Belt Storage
- Oil Traps
- Sewage Treatment Plant
- Leachate Pond
- Waste Collection Bins

Figure 18.38 below indicates the position of the different type waste facilities within the waste area.

Figure 18.38 Waste Facilities Area

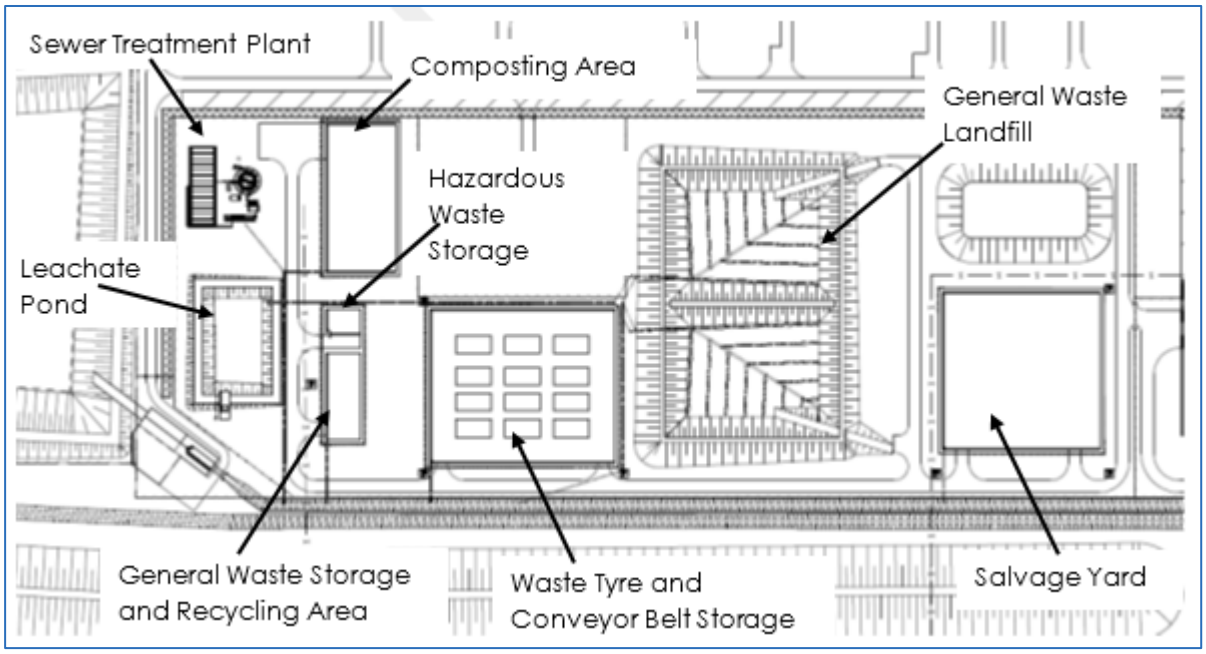


Figure by DRA, 2017.

18.15.3.1 General Waste Landfill

It has been established that Platreef will construct an on-site landfill with a designed lifetime of 36 years from the start of shaft sinking. The landfill model considers three rates of deposition:

- Bulk shaft rate of deposition (3 years) =672 tpa (1.84 tpd)
- Construction rate of deposition (3 years) =1,530 tpa (4.19 tpd)
- Operation phase rate of deposition (30 years) =708 tpa (1.94 tpd)

The landfill will be dedicated to the disposal of general waste from the mine and mine-related activities only, with the removal of recyclable materials (tins, glass, plastic and paper) as well as organic material for composting before disposal of the residual waste.

The landfill site is expected to receive general waste, packaging waste and garden waste that are regarded as residual after source separation and further sorting at the materials recovery facility.

Due to low levels of organics in the landfill, odour and vermin are considered to be a minor issue. Cover material has been assumed to be provided in a 1:5 ratio, which may be excessive due to the reduced organic composition of disposed waste.

The design is of a set of two cells with a combined airspace of 80,000 m^3. The cells will be developed with base at 3 m below ground level and reaching a maximum height of 5 m above ground level.

18.15.3.2 Composting Plant

A composting plant is to be built in the vicinity of the landfill for the processing of organic wastes into a usable product for various landscaping functions around the mine. The composition of material from the general waste stream at the mine site which will enter the composting system is estimated using the following conservative figures:

- Potential 30% of domestic waste (rubbish) if organics are separated;
- Sewage sludge in operation phase after drying beds;
- 80% of wood and garden waste;
- Potential 80% of packaging; and
- Sludge from an on-site potable water treatment plant (3-6% solids) if a provision for tankage and pumping is made at the composting facility.

The plant design will consider a first phase of 2,400 m^3 per year with a second phase starting in 2020 with capacity for a total of 4,800 m^3 per year. Material will be supplied from the general waste holding yard or material recovery facility where separation, identification and quantities delivered to composting will be recorded.

18.15.3.3 General Waste Storage and Recycling Area

All of the general waste, including recyclables and organic material, that is disposed into the various bins on site, will be taken to a centralised storage and sorting area for further sorting and consolidation before the various types of materials are dispatched to their respective destinations.

The general waste storage area will have an adjoined recycling area also referred to as a Materials Recovery Facility (MRF) as this will be the source of all recyclable materials that are sent off site. The general waste storage and recycling area should comprise the following elements:

- Recyclable skips and cages for recyclable material accumulation by type;
- Area for separate disposal of packaging waste for subsequent separation to recyclable, composting and general landfill portions;
- Area for processing of wood and garden waste for use in composting via a shredder or for direct recycling where appropriate and, for disposal of residual to landfill; and
- Area for receiving the fraction of domestic waste (rubbish) not source separated as recyclables, for processing of organics for use in composting, for direct recycling of further recyclables from the mixed waste where appropriate and for disposal of residual to landfill.

18.15.3.4 Hazardous Waste Storage Area

Hazardous waste that is generated at various locations around the mine will be taken to a centralised hazardous waste storage area. The hazardous waste storage area will be accommodated within the general waste storage area, but with separate security and entrance control as a stand-alone facility.

The hazardous waste yard will include appropriate covered storage area and will have a liner protected concrete pad with surface water containment and provision for final disposal of collected surface water. Each hazardous waste will be separately stored in the appropriate container, as discussed and provided by the hazardous waste removal contractor.

The hazardous waste storage area should be accessible to the hazardous waste collection contractors, and volumes of any hazardous waste material leaving the storage area should be recorded and put into the mine-wide waste information system.

18.15.3.5 Salvage Yard

A salvage yard is an area dedicated to the collection and sorting of re-usable parts, particularly metal parts. It is expected that a salvage yard will be alongside the scrap metal yard in the integrated waste management area (in the south-western corner of the site).

The salvage material should be identified by persons appropriately qualified to do so, and should be stored in a way that does not hamper their re-usability, and protects the salvageable parts according to their value to the mine.

A detailed record of all salvageable parts in the salvage yard should be kept up to date and in the possession of the procurement department, in order to avoid ordering new parts, while the parts exist in the salvage yard. This list should also be available to the managers of the maintenance workshops.

Any used metal part which is beyond repair or may not be re-useable in any way on-site, should be sent to the designated area in the salvage yard for scrap storage before it is sold or auctioned off to scrap metal dealers. Platreef should ensure that scrap metal is not sold to unlicensed scrap dealers. A record of sale, recording price and volume, should be kept along with company registration documents of the purchasing scrap dealers.

18.15.3.6 Waste Tyre and Conveyor Belt Storage Area

Waste tyres and conveyor belts will be stockpiled until a downstream application is found and a waste transport contractor removes them. Tyres and conveyor belting will be stored in a separate facility with the required stacking and spacing arrangements to comply with the relevant legislation, particularly with respect to firebreaks.

Firefighting equipment will be required to be located in close proximity to the facility and the following site specifications should be met:

- Ground surface is cleared, levelled and compacted;

- Run off shall be diverted from upslope by a 1 m-high diversion berm;
- A collector drain should be constructed on the boundaries to catch the run off from the site. This should discharge through a silt trap to the contaminated water system;
- Appropriate demarcation and signage will be provided to indicate responsible person details and fire prohibition;
- The site is to be fenced with two gates for access;
- Internal firebreaks/access of 5 m between stockpiles and 6 m outside the stockpiles, i.e. outside the fence; and
- Tyres to be neatly stacked, not exceeding 3 m height, 10 m width and row length to not exceed 20 m.

18.15.3.7 Oil Traps

Oil traps will be required to remove oil from surface water run-off, and thus a few of these facilities will be located on-site mainly at wash bays, workshops, maintenance areas and fuelling stations. While no manual labour is required to operate the oil traps, they will require cleaning at regular intervals, depending on the rate of oil capture. Most of the oil caught in the oil trap can be recycled and should be stored in a used oil storage tank for collection by a used oil collection contractor. An oily sludge is also produced and should be taken to a bioremediation plant off-site.

18.15.3.8 Leachate Pond

The leachate pond will receive leachate emanating from the landfill. The pond has been sized to have a volume of approximately 2,300 m³. The pond will be excavated to a depth of 3 m below ground level, and side slopes will be 1:2.5. From the pond, leachate can be pumped and sprayed over the waste body if required, in order to reduce the level of leachate in the leachate pond.

18.15.3.9 Waste Collection Bins

It has been proposed that Platreef will install waste collection points at various localities around the site for the convenient separation and disposal of various waste materials by mine employees. These collection points should each consist of five 210l wheelie bins of different colours and with distinguishing labels for the separate disposal of the main groups of recyclable waste streams: paper and cardboard (together), tins and glass (together), plastic, and mixed domestic waste, while a fifth bin should contain hazardous wastes for safe disposal to hazardous landfill. Platreef should also consider having additional bins for the separate collection of certain hazardous wastes, such as batteries and oily rags. Extra bins should only be implemented in areas where significant volumes of the relevant waste streams are expected. Refer to Table 18.11 for details.

Table 18.11 Waste Bin Suggesting Colour Coding System

Waste Category	Colour of Containers
Hazardous waste	Red
Domestic	Black
Paper and cardboard	Blue
Tins and glass	Green
Plastic	Yellow

18.16 Construction Facilities

Where possible permanent infrastructure will be utilised to minimise temporary construction facilities. Temporary construction facilities and services has been allowed for during the construction phase of the project where permanent infrastructure has not been constructed or allowed for.

These facilities are a crucial part of construction management, as sites can be very demanding involving the co-ordination and movement of large quantities of materials, as well as high-value products, plant and people. Effectively and accurately laying out a site can help ensure that the construction works are undertaken safely and efficiently. Correct sizing and positioning of temporary facilities can help to reduce travel times, congestion and waiting times making the site a more effective workplace.

The following temporary utilities have been allowed for:

Construction Power

Before grid power from the 70 MVA ESKOM supply is available, all construction power is to be supplied by local diesel generators provided by the contractors as required. A 100 kVA generator and associated reticulation has been allowed to power the EPCM construction team office.

Provision has also been made to reticulate grid power the construction offices, the construction laydowns and the concentrator plant construction areas from a series of mini-substations. Remote areas for example the tailings facility, will continue to be supplied by local diesel generators as required.

Construction Water

Initially the bulk water supply, for construction of the mine, will be provided by local groundwater from boreholes. The water will be carted to site with water bowsers and discharged where required.

Construction Laydowns

The future plant area has been identified to be used as the main construction laydown area where all surface contractors will establish their main construction camp. Allowance was made for clearing the area, removal of top soil and layer works (2 x 150 mm compacted G6 material type layers) to create a hard stand. 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. Refer to Figure 18.39 for details.

Figure 18.39 Construction Laydown Area

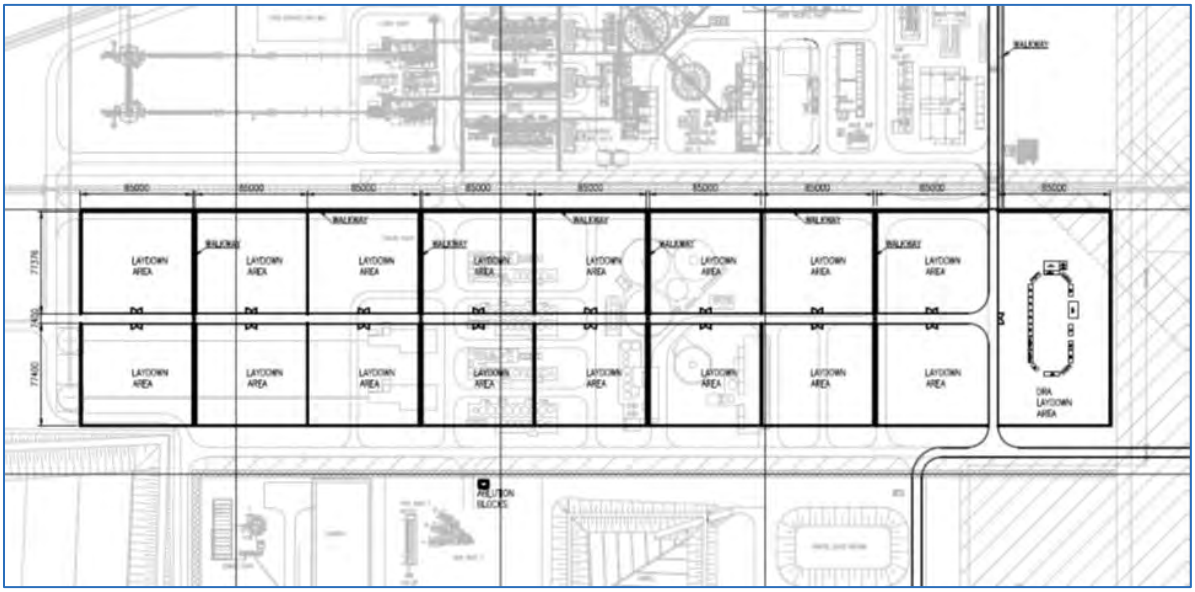


Figure by DRA, 2017.

Construction Office

The Main office block will be utilised as construction offices.

Construction Stores

The permanent Capital and Main store will be constructed and utilised during the construction as storage space for equipment.

Construction Communication

An independent IT network in the construction offices with Internet Data communications (by Satellite) is catered for the EPCM project team.

Handheld mobile radios and two base stations are provided for site voice communications.

Construction Vehicles

Construction vehicles have only been allowed for the EPCM Project team. Existing project vehicles will be utilised.

Construction SHEQ

Allowance were made for lighting detection equipment, breathalyser's, first aid equipment, permit and inspection books, construction office firefighting equipment, working at height rescue kit, PPE and environmental spill kits.

Construction Signs

Allowance was made for standards signs required for a construction site.

Construction Ablution Facilities

Ablution facilities during construction is usually each contractor's own responsibility and can sometimes create hygienic difficulties. The main aim of the Platreef Project is to do away from the chemical type temporary toilets and rather install prefabricated ablution units with a water and septic tank next to it providing flushing toilets rather than chemically cleaned toilets. 20 of these units have been allowed for that will be placed in all main construction areas of the mine. For the remote areas, mobile flushing units that is mounted on a trailer have been allowed for. A contractor will be appointed to clean and maintain these units.

Construction IT and Computer Equipment

Allowance was made for an independent IT network including file server, multifunction printer/copier, A0 plotter and WIFI networking within the construction offices. Office personal computers, audio visual projectors and a video conferencing facility is also catered for.

Construction Access and Security Facilities

Temporary container type turnstile units with card readers have been allowed for to assist with the access control of employees to the site. Vehicle access to site will be controlled by boom gates also linked to the card read system.

Construction Waste Facilities

Allowance was made for a construction salvage yard and waste skip laydown areas. A contractor will be appointed to clean and maintain the waste bin skips.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Platreef Phase 1 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/demand deficits. Favourable positioning on the cost curve, base metal diversification and a natural South African rand (ZAR) hedge should all conspire to make the concentrate attractive to South African toll smelters. Discussions between Ivanplats and interested parties are well underway, and Ivanplats expects this to result in a definitive tolling agreement.

Ivanplats prepared and updated marketing studies for the Platreef 2015 PFS and updated both content and assumptions for the Platreef 2017 FS. This Platreef 2017 FS is based on an examination of options around the Phase 1 production scenario of 4 Mtpa which would yield approximately 175,000 tpa concentrate. Ivanplats has not yet agreed specific offtake agreements for this Phase 1 production, as it is it is not typical industry practice to have such contracts in place five years before production commences. Ivanplats has a clearly defined development strategy to secure smelting and refining capacity that has considered potential treatment capacities, primarily in South Africa, and has mapped a development path for placing concentrates and expects capacity to become available by the time that steady state production is achieved.

With the establishment of a number of smaller PGM mining firms, toll smelting and refining contracts and 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 sulphate. 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/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 less 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. 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 900 vehicles. In China, the largest vehicle market, ownership levels are currently less than 100, thus, growth prospects are expected to be strong. Ivanplats engaged SFA (Oxford) Ltd (SFA) to provide a review of the PGM industry and outlook. It is estimated by SFA that automobile demand for Pt alone could rise by over 10% post-2020 while China could be using two million more oz of Pd.

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. 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. HSBC states that it is reasonable to expect industrial demand growth for PGMs to be sustained in the long term. Platinum jewellery accounts for one third of total Pt 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 drawing close to that of Russia, where reduced production from Norilsk (as a by-product of Ni) and depletion of above-ground stockpiles have a direct effect. South Africa's traditional PGM mines in the Western Bushveld are challenged by high costs, diminishing grades and labour issues around mining methods. It is envisaged that production will remain high cost and will probably peak over the next two years prior to a drop over the following five- to ten-year period. It is forecast that there will be a need for replacement ounces (oz) from new sources, including the Northern Bushveld where Platreef is located. Current South African mine economics and the consequent deferment of projects leads to a forecast that Pt and Pd markets will be in deficit when Platreef Phase 1 production commences. In a recent report, PricewaterhouseCoopers showed a platinum production profile from existing mines

dropping from over 4 Mozpa in 2017 to less than 2 Mozpa in 2040. SFA suggests that the reduced supply is a clear trend, although there may be differing interpretations on quantities. SFA approximates almost one million ounces of PGM will be lost from the predicted supply between 2020 and 2026, including over 300,000 oz from South Africa alone. This suggests there is a need for new developments and projects to come into production to fill this shortfall.

Short-term capital constraints with many of the existing mining companies and escalating costs, albeit alleviated to a degree by a weak ZAR, will push these shaft developments out. These factors, together with Platreef's favourable cost profile, would make the Phase 1 concentrate attractive for South African smelters. It is duly recognised that the industry could see both organic expansion and new projects compete for this capacity, however Platreef enjoys the benefit of a favourable Pt: Pd ratio when the declining price spread, that many forecast, will incentivise new production from the Northern Limb over the Pt-dominant areas of the Bushveld.

19.3 Trade in Flotation Concentrates

The sale of PGM flotation concentrates from various Merensky and UG2 mining operations in Southern Africa has increased over the past decades as producers seek to optimise existing metallurgical facilities and offer surplus capacity to smaller mines developed outside of the traditional larger mining lease areas. These agreements can be quite variable in net payment for metal contained but will have some of the following elements: smelter charges; refining charges; metal accountabilities; pipelines; penalties; delivery terms; assay charges; metal accounting provisions; dispute resolution mechanisms; length of contract periods; renewal conditions; other considerations. Most significant of these in economic terms are the metal accountabilities, any treatment charges, penalties and the pipelines or delays between concentrate delivery and payment. This pipeline for some metals is as long as nine months (i.e. rhodium), such that these terms can have an impact on the sellers' cash flow although platinum, palladium and the base metals are generally paid within eight to ten weeks. Flotation concentrates, because of their bulk and the South African government's focus on beneficiation, do not find a market offshore and are all sold locally. Ivanplats has seen substantial interest from offshore smelters for Phase 1 concentrate offtake, and has stated it is fully committed to the beneficiation process in South Africa and that it plans to work with the local smelters in the period between feasibility study and first concentrate production to secure suitable offtake agreements.

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 sulphates 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/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 sulphur 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. It is forecast by Ivanplats that there would be adequate smelting capacity across all of the South African operations for the first phase of Platreef production as defined in the Platreef 2017 FS. Ivanplats expects that a reduction in supply post 2020, will free up further capacity.

Base metal refining capacity requires further comment. Total nickel refining capacity in South Africa has been estimated to be 53 ktpa Ni in 2016. As Platreef ramps up from commissioning to steady state in 2023, it is forecast sufficient capacity would be freed up across all operations commensurate both with the forecast drop-off in Pt oz and continued developments in producing a more pure Nickel Sulphate for direct marketing rather than refining in South Africa.

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.

Ivanplats has entered into a confidential Letter of Intent/Cooperation Agreement with parties interested in treating Platreef Phase 1 concentrate. A joint study is currently underway to look at treatment of between 50 % and 100% of production. Ivanplats is also undertaking discussions with other parties where capacity is forecast to be available.

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 Offshore Toll Treatment Options

Many offshore smelters and refineries now recognise the benefits of optimising capacities by toll treating third party products. Many of these facilities have expressed a keen interest in working with Platreef both in supplying treatment terms for various products, including concentrate, matte, base metal products and in exploring ways to cooperate metallurgically in South Africa. This interest highlights the importance of the Platreef Project to the PGM industry. Ivanplats is committed to working within the South African government's guidelines on domestic beneficiation where local capacities and treatment options are available, thus, whilst there would be offshore options as backup, there is a high likelihood that the full Phase 1 concentrate production profile will be processed in South Africa.

19.8 Treatment Contract and Cost Structures

There are many ways of structuring concentrate-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%. For the purposes of this study the assumption has been made of a forecast total net payable of 82%.

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/copper to be refined, although the cost of removing sulphur 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.

19.9 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.

19.10 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.

19.11 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.

19.12 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.

19.13 Platreef 2017 FS Smelter Terms and Price Assumptions

The prices in the economic analysis for the Platreef 2017 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 US$1,250/oz Pt, US$825/oz Pd, US$1,300/oz Au, US$1,000/oz Rh, US$7.60/lb Ni, and US$3.00/lb Cu. For the Platreef 2017 FS the average transport distance for the concentrate has been assumed to be a distance of 154 km.

19.14 Conclusions and Development Plan

Platreef Phase 1 steady state production is forecast to be 2023 with ramp up commencing 2022. Consequently, firm commitments from third party toll smelters will not be given at the feasibility study stage, 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 Phase 1 offtake may be placed with more than one customer depending on interest, capacities and commercial terms.

The studies by Ivanplats indicate that there will be sufficient smelting capacity in South Africa for Phase 1, 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.

The Ivanplats strategy is to seek indications of interest from these third parties for Phase 1 concentrate production and to continue the current technical discussions with a view to finalising offtake agreements. The requirement and timing of these agreements would be assessed and offshore treatment scenarios factored to support in-country sales as Platreef moves closer to production.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Impact Assessment and Environmental Management Plan

In 2013, Ivanplats undertook the Environmental Impact Assessment (EIA) 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 EIA/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 and the re-alignment of the TSF pipeline route, will require an amendment application process to be undertaken on the EA (estimated cost of R750,000.00); and
- 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 EIA 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 EIA 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 EIA 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.

Ivanplats made commitments in terms of its Social and Labour Plan (SLP) for a five-year period between 2015 and 2019.

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

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

The main potential social impacts associated with the Platreef Project include some economic displacement due to a loss of access to cultivated land or other livelihood resources, influx in job seeking which, combined with the additional workforce, will place considerable pressure on local infrastructure and services, negative perceptions of project impacts and increased traffic volumes on roads in the vicinity of the local project area. Further to this, there are social risks due to the social environment under which the Platreef Project operates as well as stakeholder fatigue resulting from ongoing mining and exploration activities within the area. Community unrest poses the risk of striking, property destruction and interruptions of operation schedules. The various stakeholder engagement processes revealed a reoccurrence of issues raised by stakeholders regarding the Platreef Project. Stakeholder engagement is an ongoing process, and a grievance mechanism has been developed to manage stakeholder concerns.

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

20.2 Water Use Licence (WUL)

20.2.1 General Authorisation, Integrated Water Use Licence Application

An assessment of the General Authorisation (GA) from four properties (1,077 ha) on the Uitloop 3 KS farm was prepared and submitted to the DWS Limpopo Region (21 October 2013) and an acknowledgement of receipt was obtained. A request from DWS for Ivanplats to submit copies of agreements for the taking of water from two privately-owned properties on the Uitloop 3 KS farm was subsequently received. Agreements were drafted for each of the two landowners. These were submitted to Ivanplats for signature by the landowners and returned. Subsequently the signed agreements have been submitted to DWS and the registration of the water use on the WRMS data base finalised.

The IWULA and IWWMP for bulk sampling were submitted to DWS, Limpopo Office on 6 November 2013. Receipts were obtained and copies provided to Ivanplats.

The Bulk Sampling Shaft (BSS) IWUL was signed on the 9 March 2017, IWUL Licence Number: 01/A61F/AGJ/5021.

20.2.2 Integrated Water Use Licence (IWUL) for Main Mine

An engineering design review meeting took place with Mr Kelvin Legge and his team from the National DWS Department. All the engineering designs including those for the Tailings Storage Facility (TSF) were reviewed. The Platreef 2017 FS design included all the requirements of the regulator. The DWS issued a letter of acceptance of the TSF design, as well as the other water and waste management facilities on the site.

The Main IWUL application is in progress and the application will be done following Regulation No. R 267 of 24 March 2017. This is a 9-step process and the first steps of the pre-application meeting and submission of the DW755A forms with supporting information relating to the mine has been submitted and a letter of acknowledgement received from DWS, accepting the application and requesting that the next step (technical information now take place).

An application for a Dam Safety Permit for Pond 3 is still in progress.

20.2.3 Integrated Waste Management Licence

Platreef was issued with a Waste Management Licence (Ref. No. 12/9/11/L1224/5) in terms of the National Environmental Waste Management Act on 15 January 2014.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating costs have been divided into functional cost areas based on the project work breakdown structure (WBS) as follows:

- Area 1000 – Geology
- Area 2000 – Mining
- Area 3000 – Concentrator
- Area 6000 – Infrastructure, Utilities and Ancillaries
- Area 7000 – Site Costs
- Area 8000 – Owners Costs and General and Administration

DRA compiled the overall capital and operating cost estimates. The estimates were prepared by the following for the areas shown below:

- DRA Projects SA (Pty) Ltd
- Stantec
- Murray and Roberts Cementation (Pty) Ltd
- SRK (Pty) Ltd
- Golder Associates Africa (Pty) Ltd
- Digby Wells Environmental (Pty) Ltd.
- OreWin 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 2017, ramp-up to full production steady state in 2023 and finally mine closure requirements in 2053.

21.2 Discipline Specific Estimates

The methodology applied to the discipline specific estimates was:

- Earthworks, Earthworks quantities were measured from completed drawings and a BOQ, in standard Committee of Land Transport Officials format was produced. A significant portion of the earthworks estimate is the use of G4 materials for terracing. Rather than using commercially sourced materials, the estimate is based on using materials produced by the waste rock crushing enterprise development project. For the earthworks estimates the costs of the G4 materials included in the Enterprise Development estimate for the Area – 8000 Owner's Costs. The surface fleet associated with the waste rock dump management was estimated as contractor supplied on a rental basis and are therefore operating costs.

- 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 were grouped into procurement packages, and a request for quotations issued accordingly. Quotations were received, adjudicated and applied to develop the estimate. Larger mechanical equipment such as Winders, Mills, Filter Presses and Underground Mobile Fleet was enquired as formal tenders, with specific terms and conditions. Detailed adjudication was performed and pricing for estimating purposes was selected on this basis.
- Piping. Although no detailed piping drawings were done, quantities were measured from the material take-offs and line lists. Underground piping supply was also estimated by means of a priced BOQ, with installation included in the mine development contractor rates.
- Electrical, Instrumentation and Control (EC&I). The EC&I estimate was derived using a combination of database rates, factorization and quotations received.

For each discipline, a formal request for quotations was issued to the market place complete with:

- Detailed scope of work,
- Standard FIDIC terms and Conditions,
- BOQ; and
- High Level Schedule.

A rates adjudication was performed, inclusive of current execution project rates as a benchmark. Suitable rates were selected and applied to the final quantities in developing the overall estimate.

21.3 Labour

Ivanplats has put in place a recruitment and selection process to be fair and transparent and to ensure that South Africans will be employed as part of the National Development Plan and local potential employees trained through skills transfer from expatriate labour. Ivanplats will deploy compensation and benefits that are of international standard and compensate employees within benchmarked labour rates with additional appropriate and transparent target-driven incentive schemes.

The community skills audit has identified that there is a shortage of mining skills required for the operation in the labour sending areas. This requires Ivanplats to set the objective to develop and reskill community members to ensure that the required skills will be available. This process will occur through skills transfer and by implementing several other development plans. The mine will be developed and mined by contractor model in its initial stages and be taken over by the owner at steady state in phases.

The mine will employ in total 2,000 employees at steady state with an emphasis on high technical skill levels. To train and maintain the workforce, integrated Human Capital Development Programmes will be set focussing specifically on the hard-to-fill positions but not neglecting others. Training will be conducted in a Mine Qualification Authority accredited centre that will have skills programmes in relation to the mine's operation. This will be done by ensuring that each job profile is well developed and that each employee will have a development plan on how he or she can progress through the relevant career path. The training will be conducted in an environment where a student can comply with the lead times but still achieve the correct level of competency by means of the appropriate mix of theoretical and practical training. This will be realised through e-learning, simulation and virtual training.

Employee relations will be a key factor in making a success of this mega mine. Organised labour will be managed inside set and agreed-upon recognition agreements and in an environment of mutual respect. By employing a well-defined joint communication strategy, the associated threats of an uninformed workforce can be addressed and if a dispute arises it can be mitigated through the set strategy and guidelines. The community as stakeholder will be managed by a well-developed engagement strategy supported by a fully-staffed department. At the end of the mine life the project will downscale and close. It has already planned and made financial provision for this event in the SLP.

21.3.1 Salary and Wage Structure

The structure of salaries will be based on the current Total Cost Package (TCP) principle in the Paterson D to E levels. C, B and A levels are currently unionised and will be structured packages tailored according to the specific level's requirement and agreed in annual wage negotiations. All packages will be 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) will be deducted. All employees will partake in the company provident scheme currently at 14% of TCP. The provident deduction includes a company funeral fund and a free funeral fund (currently 2016). All employees will be covered by the group Disability and Life Cover related to the mining risk; the rates will be deducted from the TCP and paid over by the company for each active employee.

Paterson C, A and B employees will have the following allowances included in their TCP:

- Housing Shift Allowance or Home Ownership
- Transport Shift Allowance (dependent on own or company transport)
- Food Allowance (dependent on own or company provided meal)
- Shift Allowance (when on shift cycle)

21.3.2 Shift Cycles

The Project will operate on a continuous operation model - in planning the most cost-effective and optimised shift cycles the company did a trade-off study of several shift cycles. The optimised cycle decided on was a four-team 11.5 hour four-day shift on four-night shift and four days off. There will always be two teams off. This cycle was chosen as it supports the fatigue management programme to ensure effective handover between shifts in the mining cycle.

21.3.3 Labour Plan

The Labour Plan for the mine has been designed for the mine steady state projected at 2024. The construction and development phase of the mine and later on the plant will take place consecutively within the first seven years up until steady state is reached. Labour will increase as the phases of construction and development progress until mining commences and progresses until the 4 Mtpa production target is reached. Employment of workers will take place in such a way that the right people are in the right place at the right time when the mine is operationally ready. In period from 2017 to 2024, all construction, development, mining and processing will take place by contractor model.

The following histogram (Figure 21.1) represents the projected organogram of Ivanplats as at the height of production of the mine and the plant in 2024. It must be noted that these are only projected figures and exclude surface construction employees.

Figure 21.1 Project Labour

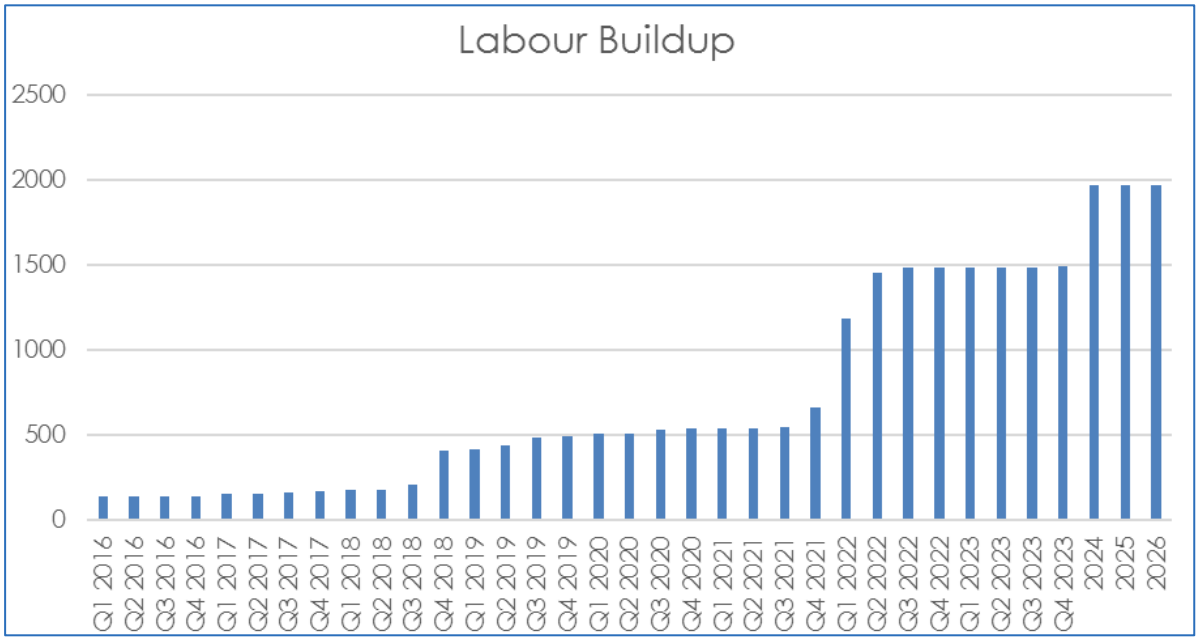


Figure provided by Ivanhoe, 2017.

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 life of mine.

Operating costs include the following:

- Mining Geology
- Mine Technical Services
- Geotechnical Engineering (underground (UG))
- Laboratory, Sampling and Assaying

21.4.2 Area 2000 – Mining

The capital costs associated with:

- Shaft 1:
  - Sinking of shaft,
  - Changing of shaft infrastructure to cater for temporary early development hoisting and
  - Converting to permanent duty as ventilation shaft.
- Shaft 2
  - Construction of headgear,
  - Sinking of shaft, and
  - Equipping of shaft and associated infrastructure as required for production.
- Ventilation Raises
  - Raise boring of raises; and
  - Installation of infrastructure to establish each of the three ventilation raises into up-cast ventilation shafts

- Shaft Surface Infrastructure
  - Establishment of all surface infrastructure, required for steady state mine operation which include but is not limited to:
  - Surface backfill plants,
  - Surface building and Earthworks,
  - Electrical reticulations,
  - Rock handling and associated stockpiling; and
  - Air Cooling
- Underground Mining –
  - Underground Development and Mining (Waste Development and Ore Development)
  - Establishment of all underground infrastructure required to achieve steady state mining operations, which include but is not limited to:
  - Underground backfill systems,
  - Underground Fleet (Primary and Secondary)
  - Settling and Dewatering,
  - Crushing; and
  - Underground Workshops
- Commission and spares;
- EPCM

The mining operating costs include the following:

- Labour – All mining related labour, including technical services;
- Power – Bulk Eksom power supply, excluding generator power;
- Water – Bulk water supply applicable to Area 2000, obtained from the site wide water balance;
- Waste development – All lateral waste development, , after the preproduction period, including permanent materials, mobile equipment running costs and consumables;
- Ore development – Ore drift development including permanent materials, mobile equipment running costs and consumables;
- Vertical development – Contractor raiseboring and in-house drop raising;
- Stoping – Permanent materials, mobile equipment running cost and consumables, for drift and fill stoping and longhole stoping;
- Consumables – maintenance consumables on engineering infrastructure;
- Construction – ongoing construction of underground engineering infrastructure, such as pumps, ventilation controls, escape routes, tips, chutes and refuge bays;
- Backfill - All costs related to backfill manufacturing and distribution

Operating costs for mining include all associated costs for ore and waste development, longhole stoping, and drift-and-fill stoping, labour, power, water, fleet running costs, backfill.

21.4.3 Area 3000 – Concentrator

The capital cost required to establish the Concentrator plant and associated infrastructure to treat 4 Mtpa at steady state ore production, producing concentrate to the correct volumes and specification. EPCM and spares are also included.

A capital cost benchmarking exercise was performed in order to compare capital costs for a number of projects in the DRA database, over the period 2003 – 2016.

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

Concentrator operating costs were benchmarked against other process plants of a similar size, using the DRA studies database. The studies and projects were concluded during 2012 to 2016. These costs compare the concentrator operating costs, inclusive of laboratory costs for assaying of concentrator samples, but exclude concentrate transport and tailings management costs. Estimates derived for the study compared well with operating costs for projects successfully concluded during this period.

21.4.4 Area 6000 - Infrastructure, Utilities and Ancillaries

Key assumptions relating to the plant and to the infrastructure capital to establish bulk, external and common infrastructure supporting mine and concentrator steady state production are:

Bulk Power

The bulk power estimate is developed based on the Eskom Self Build option, where Ivanplats will build and erect the power lines on behalf of Eskom.

Bulk Water

The bulk water estimate includes the offtake agreement with Mokopane Municipalities' Masodi water supply, and no allowance has been made for Olifants River Water Resource Development Project (ORWRDP) water.

Preliminary and General

The P&G estimates were developed both as factorized and first principle estimates. Due to the level of detail associated with Shaft 2 designs and schedule, associated P&G estimates were developed from first principles.

The remainder of the P&G estimates were all factored based, from quotations provided by the contractors.

Tailings Storage Facility,

The Tailings Storage estimate is based on formal tendered quotes from two contractors on a formal BOQ, conforming to SANS 1200 with a set of tender drawings.

Project Construction Facilities

The project philosophy is to ensure as much as possible of the permanent infrastructure is built early, and used as construction facilities.

Note: By building infrastructure early, not all facilities could be catered for. In these instances, associated costs were estimated using historical information and analogue estimation. Costs from previous projects have been included as part of the estimate.

Spares

Spares were collated into three groups, comprising capital spares, operating spares and commissioning spares. Capital spares is provided by the project capital, and therefore included in this estimate. Large capital spares were included as individual items in the procurement packages and priced accordingly. For the smaller capital spares, a 3.5% over and above allowance was made on the capital costs of each package. Operating spares was included as part of the operating cost estimate, and commissioning spares was priced as part of the 3.5% allowance.

EPCM

For the Platreef 2017 FS the EPCM estimate was developed as a factored estimate, and not based on any labour hour schedules. The EPCM estimate was broken down into facility level, and estimated accordingly.

21.4.5 Area 7000 – Site Costs

Site based capital costs required during building the project:

- Site Safety, Health and Environment,
- Ivanplats Project Development team
- Project Development Fleet, and
- Site security

21.4.6 Area 8000 – Owners Costs, Administration and Overheads

The Capital costs associated with:

Project Services

Project execution services other than EPCM. This includes but not limited to:

- Commissioning,
- Project consultant fee's (other than EPCM),
- Financing fees; and
- Logistics and transport.

Owners Team Administration and Overheads

Capital costs associated with owner's team administration and overheads, this includes:

- Traveling costs,
- Compliance and government fees,
- Mine IT costs,
- Legal and Joint Venture (JV) fees; and
- Management fees.

Environmental and Social

Capital costs commitments as required by the environmental management plan and implementation, together with all future enterprise development and social and labour plan commitments, as required by legislation.

Closure Costs

The Guideline Document for Evaluation of the Quantum of Closure-Related Financial Provision Provided by a Mine, was used to assess the applicant's environmental liability at the feasibility stage. The Department of Mineral Resources (DMR) Guideline format makes use of a set template, for which defined rates and multiplication factors are used. Multiplication and weighting factors which ultimately define the rate to be used, are determined by topography, classification of the mine according to mineral mined, the risk class of the mine and its proximity to build up, or urban areas.

21.4.7 Area 9000 – Contingency

The estimate is presented in real terms based on Quarter three 2016 pricing received. This includes all capital estimate contingency allowed for.

21.5 Cost Basis and Exchange Rates

The base date for the capital estimate is end quarter three 2016. 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 2017 Exchange Rates

Currency	Rate
US$/US$	1.00
ZAR/US$	13
EUR/US$	0.79
AUD/US$	1.18
CNY/US$	5.91
GBP/US$	0.67
JPY/US$	92.86
NOK/US$	7.39
SEK/US$	7.56

21.6 Project Execution and Life of Mine Schedule

Two critical paths exist within the project schedule. The first critical path affects the first concentrate date in that any slippage on this path will result in a delay in first revenue and concentrate. The second critical path affects the steady state production date, in that any slippage on this path will result in a delay in meeting the 4 Mtpa steady state production target. Figure 21.2 illustrates the first critical path from project start to first concentrate. Figure 21.3 illustrates the second critical path from project start to steady state production.

Figure 21.2 Project Critical Path 1 – First Concentrate

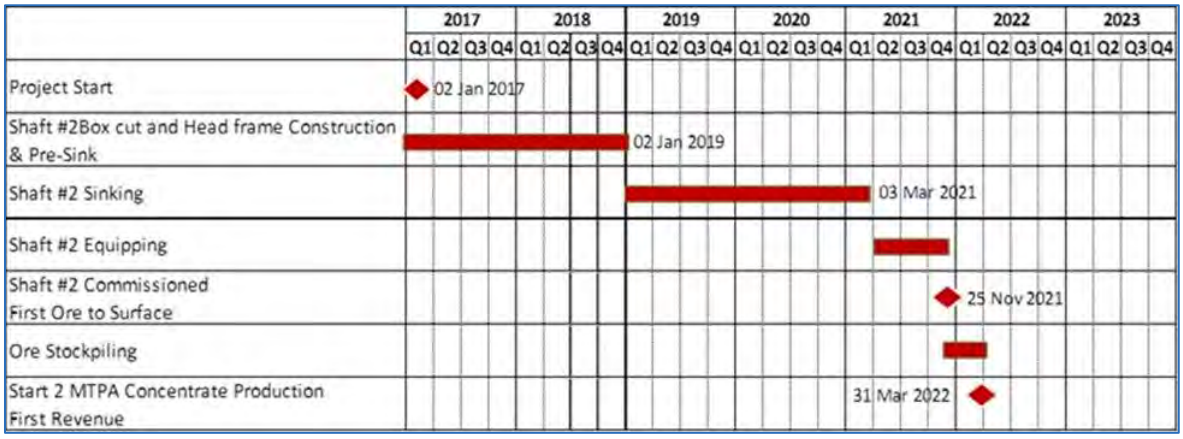


Figure provided by DRA, 2017.

Figure 21.3 Project Critical Path 2 – steady state production

Figure provided by DRA, 2017.

The key activities and dates within the project execution schedule are shown in Figure 21.4 and Figure 21.5.

Figure 21.4 Platreef Project Key Milestones

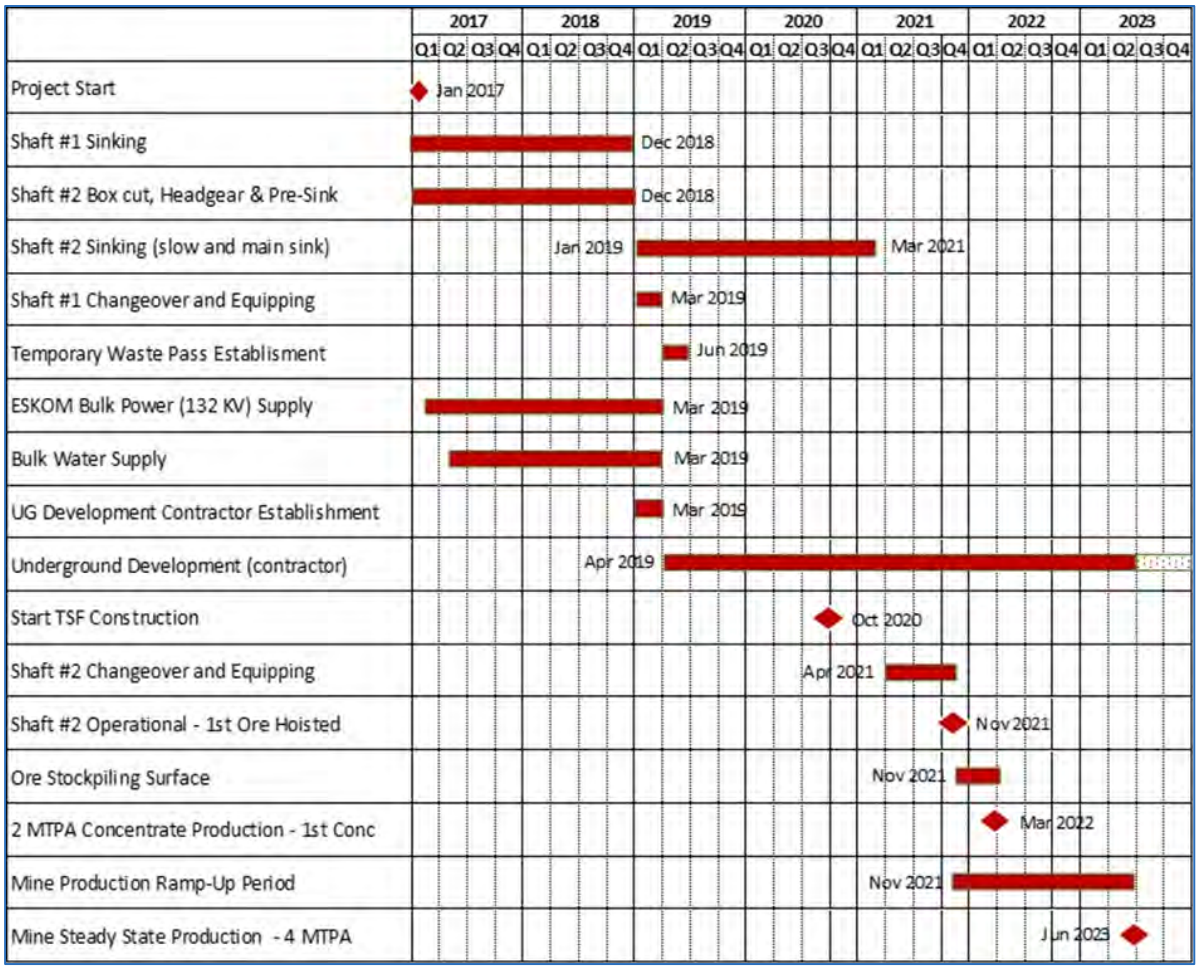


Figure provided by DRA, 2017.

Figure 21.5 Platreef Project Life of Mine Activities

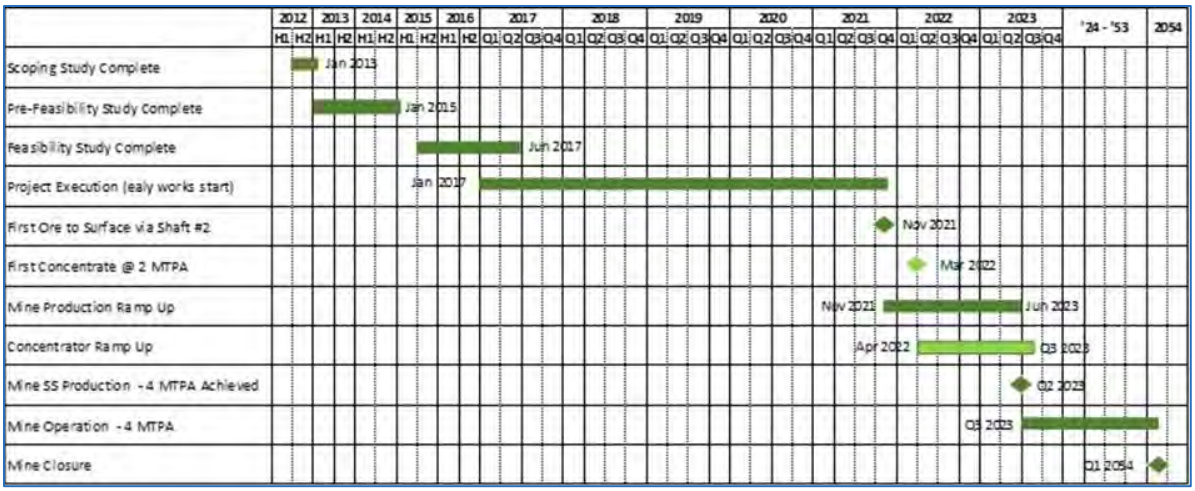


Figure provided by DRA, 2017.

21.7 Cost Estimation Methodology

The following techniques were used in preparation of the capital and operating cost estimates:

- 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 market place 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 market place. 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 principle estimates. Owing to the level of detail associated with the designs and schedule of Shaft 2, associated P&G estimates were developed from first principles. The remainder of the P&G estimates were all factored from quotations received.

Spare parts estimates were accounted for in three principal groups:(i) capital, (ii) operating and (iii) commissioning. The high value and long lead time capital spares were included as individual items in the procurement packages and priced accordingly. An allowance of 3.5% of the capital value was included as an allowance for other capital spares.

Operating spares are included as part of the operating cost estimate and commissioning spares are priced as part of the 3.5% allowance.

Mine closure capital costs are based on the Department of Mineral Resources guideline entitled Evaluation of the Quantum of Closure-Related Financial Provision Provided by a Mine.

The cost estimate is a project basis costing, it does not include escalation and the cost of project financing.

All expenditure prior to January 2017 were classified as sunk costs and have not been included in the capital estimate.

Benchmarking of the concentrator plant capital costs was carried out using the DRA database. As there is not another project with the same design, layout, schedule and operating scenario as the Platreef mine and general infrastructure, direct benchmarking comparisons of the capital costs was not undertaken. Operating cost benchmarking was performed on each of these areas.

A number of review sessions were held for purposes of quantifying the overall estimate quality, including detailed reviews throughout the development of the Platreef 2017 FS.

21.8 Capital Cost Summary

The total capital cost includes both pre-production capital and sustaining capital. Pre-production capital includes all direct and indirect mine development and construction costs prior to the start of production. 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 21.2.

Table 21.2 Capital Expenditure Summaries

US$M	Pre-Production	Sustaining	Total
Mining			
Geology	12	9	20
Mining	779	349	1,129
Capitalised Operating Costs	63	–	63
Subtotal	854	358	1,213
Concentrator & Tailings			
Concentrator	240	6	246
Capitalised Operating Costs	0.05	–	0.05
Subtotal	240	6	246
Infrastructure			
Infrastructure	230	23	253
Site Costs	9	3	11
Capitalised Operating Costs	36	–	36
Subtotal	275	26	300
Owners Cost			
Owners Cost	44	8	52
Closure	1	16	17
Subtotal	45	24	69
Capex Before Contingency	1,413	414	1,827
Contingency	131	3	135
Capex After Contingency	1,544	418	1,962

1. Sustaining capital expenditure also includes 2023 construction capital expenditure
2. Totals vary due to rounding.

21.9 Operating Cost Summary

The operating costs and revenues are summarised in Table 21.3. The proportion of costs by category for mining, concentrator and infrastructure are shown in Figure 21.6 to Figure 21.8.

Table 21.3 Operating Costs

	Life of Mine TOTAL US$M	US$/t Milled		
		YEARS 1-5 AVERAGE	YEARS 1-10 AVERAGE	LOM AVERAGE
Concentrate Transport	76	0.70	0.68	0.61
Site Operating Costs				
Mining	4,150	46.39	37.89	33.29
Processing & Tailings	1,303	10.91	10.64	10.45
Infrastructure	295	2.91	2.60	2.37
Site Cost	56	0.37	0.42	0.45
General & Administration	279	2.29	2.30	2.24
Total	6,083	62.87	53.86	48.79

Totals may vary due to rounding.

Figure 21.6 Area 2000 - Mining Operating Cost Split

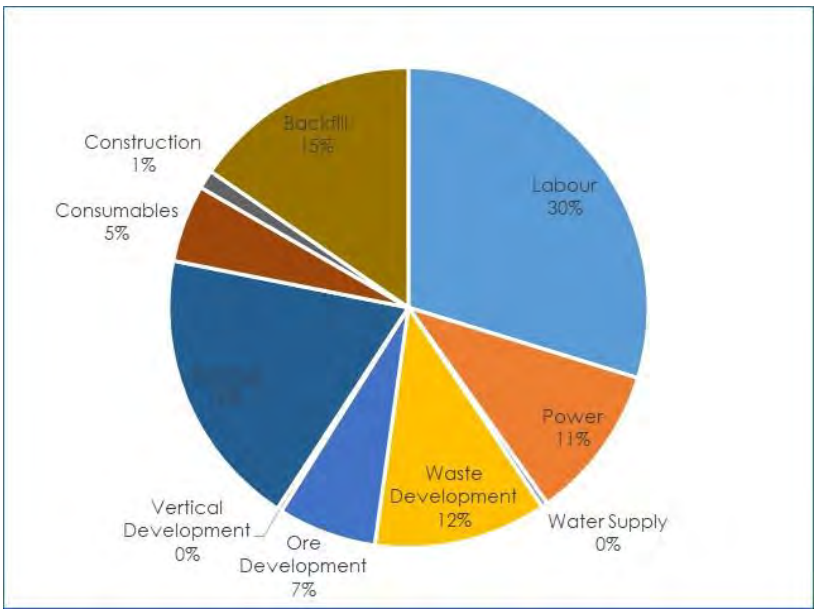


Figure provided by DRA, 2017.

Figure 21.7 Area 3000 – Concentrator Operating Cost Split

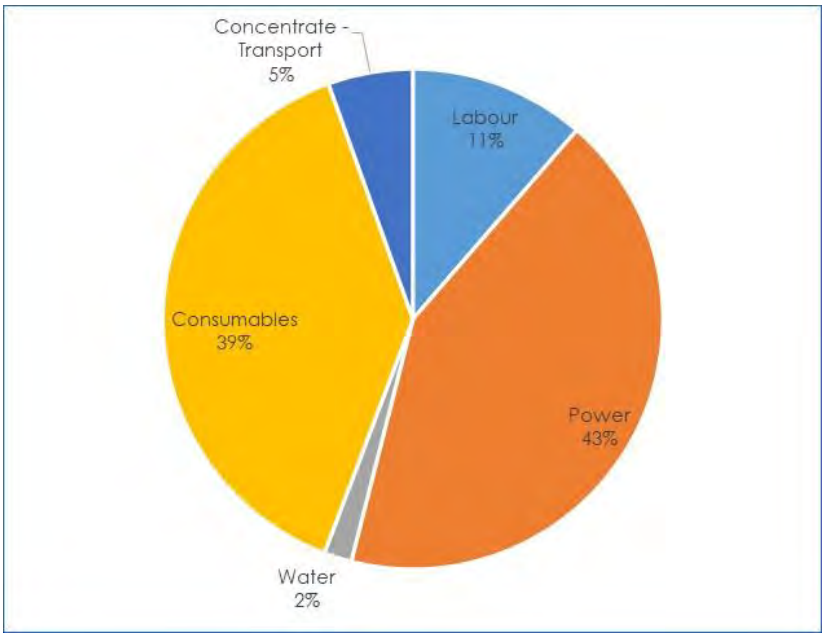


Figure provided by DRA, 2017.

Figure 21.8 Area 6000 - Infrastructure, Utilities and Ancillaries Operating Cost Split

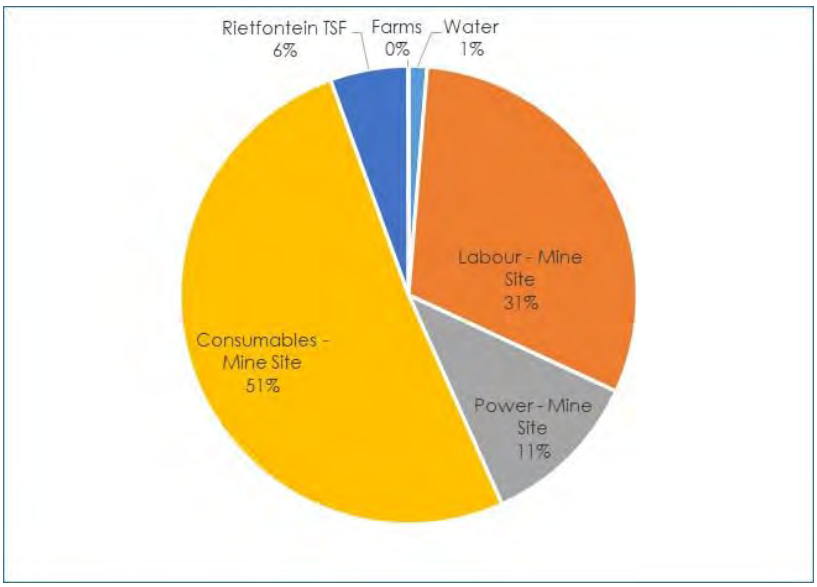


Figure provided by DRA, 2017.

22 ECONOMIC ANALYSIS

22.1 Summary of Financial Results

The key features of the Platreef 2017 FS include:

- Development of a large, mechanised, underground mine is planned at an initial 4 Mtpa throughput scenario.
- Planned average annual production rate of 476 koz of platinum, palladium, rhodium, and gold (3PE+Au).
- Estimated pre-production capital requirement of approximately US$1.5 billion, including US$131 million in contingencies.
- After-tax Net Present Value (NPV) of US$916 million, at an 8% discount rate.
- After-tax Internal Rate of Return (IRR) of 14%.
- The Platreef 2017 FS maintains options available to accelerate expansions, to the 8 Mtpa or the 12 Mtpa scenarios, as the market dictates.

Mine production is shown in Figure 22.1, and the after tax cash flow is shown in Figure 22.2. The key production and financial results including Net Present Value at 8% Discount Rate (NPV8) and Internal Rate of Return (IRR) of the Platreef 2017 FS are shown in Table 22.1.

Figure 22.1 Mining Production

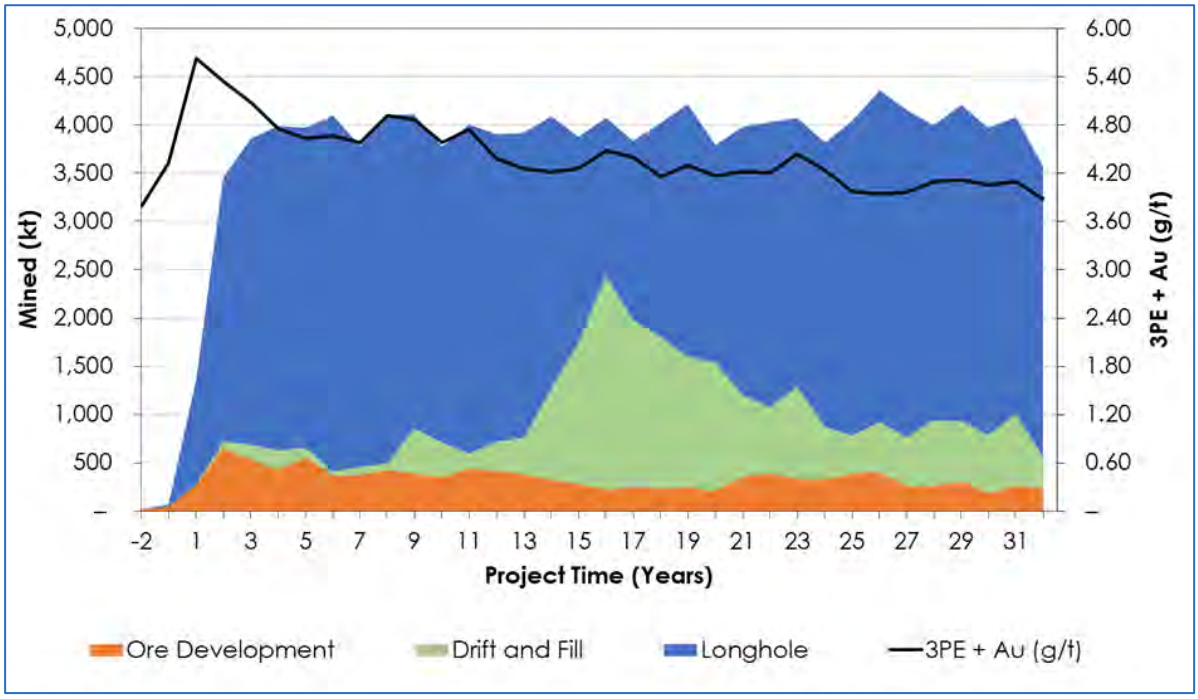


Figure by OreWin, 2017.

Figure 22.2 Cumulative Cash Flow After Tax

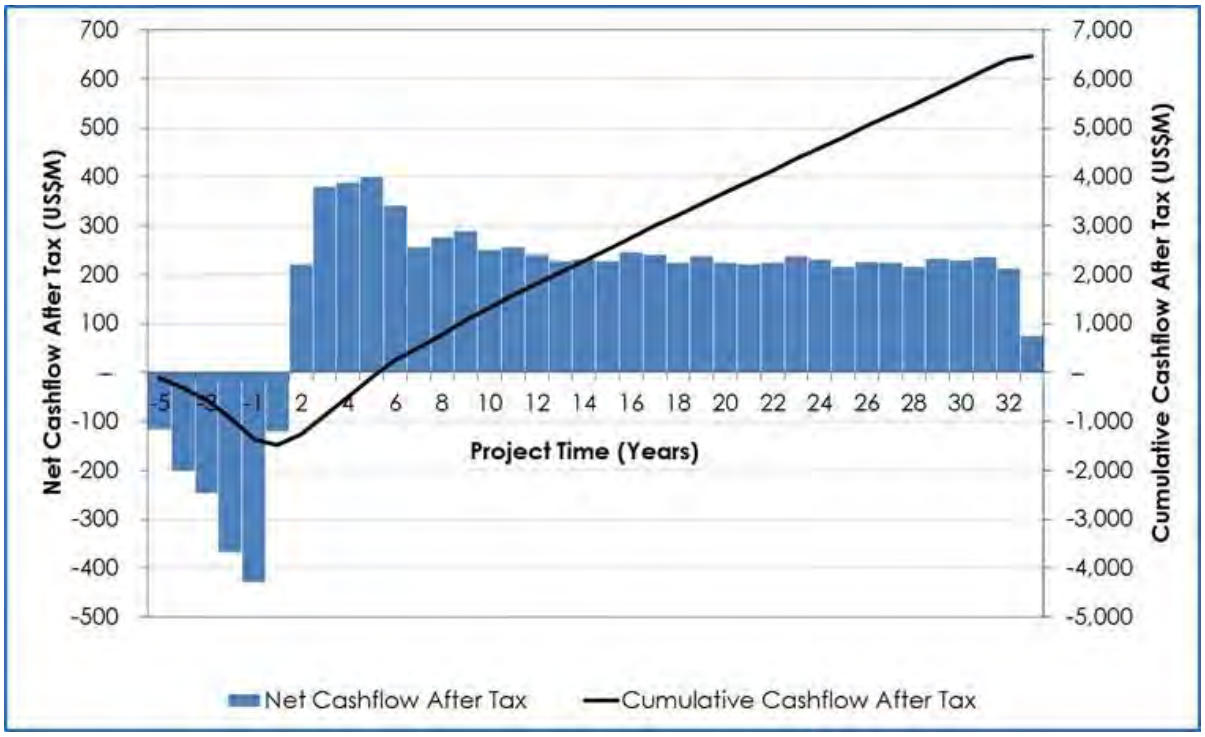


Figure by OreWin,2017.

Table 22.1 Platreef 2017 FS Results

Item	Unit	Total
Mined and Processed	Mt	125
Platinum	g/t	1.95
Palladium	g/t	2.01
Gold	g/t	0.30
Rhodium	g/t	0.14
3PE + Au	g/t	4.40
Copper	% Cu	0.17
Nickel	% Ni	0.34
Concentrate Produced	kt	5,568
Platinum	g/t	38.24
Palladium	g/t	39.07
Gold	g/t	5.32
Rhodium	g/t	2.43
3PE + Au	g/t	85.07
Copper	% Cu	3.27
Nickel	% Ni	5.46
Recovered Metal		
Platinum	koz	6,846
Palladium	koz	6,994
Gold	koz	952
Rhodium	koz	436
3PE + Au	koz	15,228
Copper	Mlb	402
Nickel	Mlb	670
Key financial results		
Life of Mine	Years	32
Pre-Production	US$M	1,544
Mine-Site Cash Cost	US$/oz Recovered 3PE + Au	399.5
Total Cash Costs After Credits	US$/oz Recovered 3PE + Au	325.7
Total Cash Costs After Credits & Sustaining Capital	US$/oz Recovered 3PE + Au	350.7
Site Operating Costs	US$/t Milled	48.79
After-Tax NPV8%	US$M	916
After-Tax IRR	%	14.2
Project Payback Period	Years	5.3

Totals may not add due to rounding.

1. The economic analysis is based on Probable Mineral Reserves only.
2. 3PE+Au = platinum, palladium, rhodium and gold.
3. Metal prices used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper.
4. A declining Net Smelter Return (NSR) cut-off of US$155/tonne–$80/tonne was used in the Mineral Reserve estimate.
5. Metal price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.

6. All-in cash costs include sustaining capital costs.

22.2 Model Assumptions

22.2.1 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2017. It is presented in 2017 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 2017 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 US$1,250/oz Pt, US$825/oz Pd, US$1,300/oz Au, US$1,000/oz Rh, US$7.60/lb Ni, and US$3.00/lb Cu. Costs estimated in ZAR have been converted to US$ at an exchange rate of 13 ZAR/US$. The key economic assumptions for the analyses are shown in Table 22.2.

Table 22.2 Economic Assumptions

Parameter	Unit	Financial Analysis Assumptions
Platinum	US$/oz	1,250
Palladium	US$/oz	825
Gold	US$/oz	1,300
Rhodium	US$/oz	1,000
Copper	US$/lb	3.00
Nickel	US$/lb	7.60
Copper Refining Charges	% Gross Sales	18%
Nickel Refining Charges	% Gross Sales	18%
Precious Metals Refining Charge	% Gross Sales	18%

22.2.2 Republic of South Africa Fiscal Environment

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

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

22.3 Management Fees

Ivanhoe shall earn a management fee, to be paid by the Platreef Project, equal to 3% of all items, directly or indirectly, incurred or accrued in relation to operating the Platreef Project other than capital costs. The NPV8% of the management fee is $47 M.

22.4 Project Results

The results of the financial analysis show an After Tax NPV8% of US$916 M. The case exhibits an after tax IRR of around 14% and a payback period of around five years. The estimates of cash flows have been prepared on a real basis as at 1 January 2017 and a mid-year discounting is taken to calculate Net Present Value (NPV). A summary of the financial results is shown in Table 22.3. The mining production statistics are shown in Table 22.4.

Table 22.3 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	8,897	6,471
	5.0%	2,794	1,961
	8.0%	1,392	916
	10.0%	838	500
	12.0%	461	217
	15.0%	99	-57
	20.0%	-212	-291
Internal Rate of Return		16.2%	14.2%
Project Payback Period (Years)		5.2	5.3

Table 22.4 Mining Production Statistics

Item	Unit	Total LOM	Years 1-5 Average	Years 1-10 Average	LOM Average
Mined and Processed	Mt	125	3.35	3.66	3.90
Platinum	g/t	1.95	2.19	2.14	1.95
Palladium	g/t	2.01	2.34	2.25	2.01
Gold	g/t	0.30	0.31	0.31	0.30
Rhodium	g/t	0.14	0.16	0.15	0.14
3PE + Au	g/t	4.40	5.00	4.85	4.40
Copper	% Cu	0.17	0.17	0.17	0.17
Nickel	% Ni	0.34	0.35	0.35	0.34
Concentrator Recoveries					
Platinum	%	87.37	89.37	88.94	87.37
Palladium	%	86.93	89.14	88.70	86.93
Gold	%	78.63	79.87	79.65	78.63
Rhodium	%	80.47	83.11	82.54	80.47
3PE + Au	%	86.36	88.47	88.04	86.36
Copper	%	87.88	88.61	88.03	87.88
Nickel	%	71.88	73.27	72.59	71.88
Concentrate Produced	kt (dry)	5,568	173	184	174
Platinum	g/t	38.24	37.82	38.03	38.24
Palladium	g/t	39.07	40.28	39.79	39.07
Gold	g/t	5.32	4.83	4.88	5.32
Rhodium	g/t	2.43	2.52	2.51	2.43
3PE + Au	g/t	85.07	85.44	85.21	85.07
Copper	% Cu	3.27	2.97	2.95	3.27
Nickel	% Ni	5.46	4.99	5.01	5.46
Recovered Metal					
Platinum	koz	6,846	211	224	214
Palladium	koz	6,994	224	235	219
Gold	koz	952	27	29	30
Rhodium	koz	436	14	15	14
3PE + Au	koz	15,228	476	503	476
Copper	Mlb	402	11	12	13
Nickel	Mlb	670	19	20	21

22.5 Capital and Operating Cost Summary

Mine site cash costs are summarised in Table 22.5. The revenues and operating costs are presented in Table 22.6.

The higher nickel and copper grades contribute to lower operating cash costs for the Northern Limb as illustrated by Figure 22.3. Among the current and future Northern Limb producers, Platreef's estimated cash cost of US$326/oz recovered 3PE+Au , net of copper and nickel by-product credits, ranks at the bottom of the cash-cost curve.

Table 22.5 Cash Costs After Credits

	US$/oz Recovered 3PE + Au		
	Years 1-5 Average	Years 1-10 Average	LOM Average
Mine Site cash costs	442.3	392.1	399.5
Realisation	266.6	304.3	339.8
Total Cash Costs Before Credits	708.9	696.4	739.2
Nickel Credits	304.5	306.5	334.4
Copper Credits	71.5	71.1	79.1
Total Cash Costs After Credits	332.9	318.9	325.7
Sustaining Capital Costs	25.4	26.1	25.0
Total Cash Costs After Credits & Sustaining Capital	358.3	345.0	350.7

Figure 22.3 Cash Cost Comparison Platreef 2017 FS and 2017 Producers

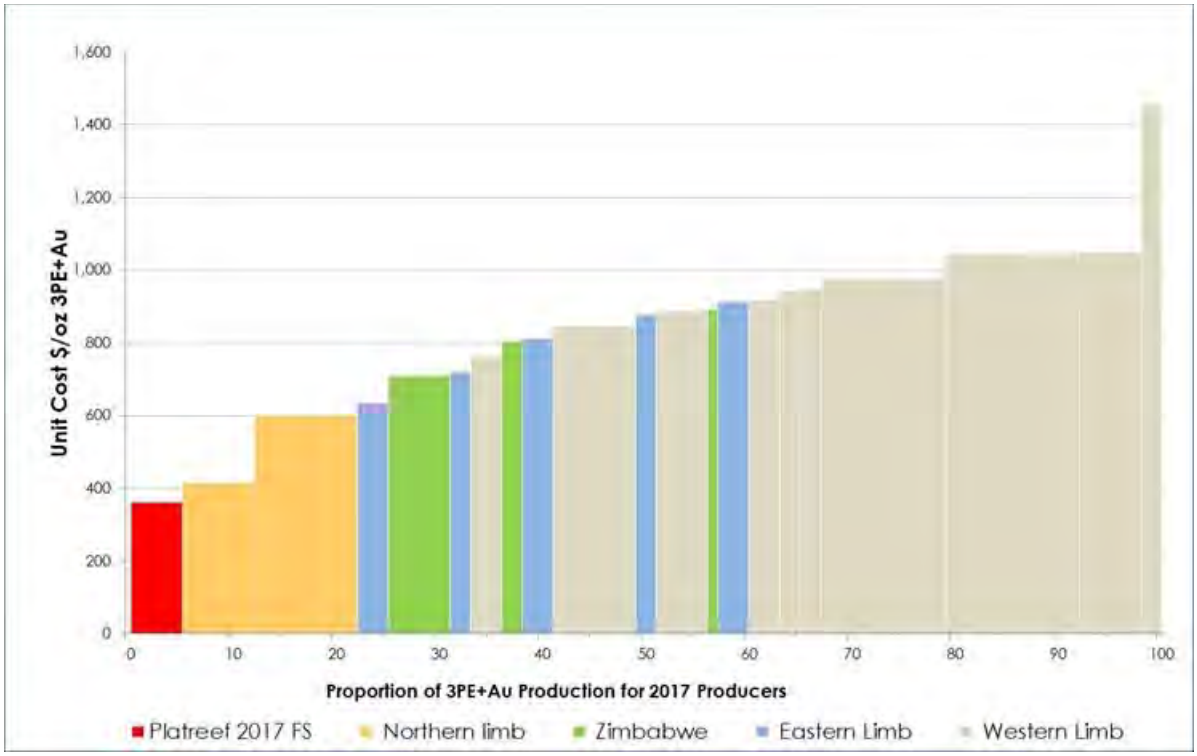


Figure by OreWin Chart shows cash cost including sustaining costs for producer cost data. Source: SFA (Oxford).

Table 22.6 Operating Costs and Revenues

	LOM Total US$M	US$/t Milled		
		Years 1-5 Average	Years 1-10 Average	LOM Average
Gross Sales Revenue	22,298	202.01	195.65	178.84
Less: Realisation Costs				
Transport	76	0.70	0.68	0.61
Treatment & Refining	4,014	36.36	35.22	32.19
Royalties	1,084	0.83	5.89	8.70
Total Realisation Costs	5,174	37.89	41.79	41.50
Net Sales Revenue	17,124	164.12	153.86	137.34
Site Operating Costs				
Mining	4,150	46.39	37.89	33.29
Processing & Tailings	1,303	10.91	10.64	10.45
Infrastructure	295	2.91	2.60	2.37
Site Cost	56	0.37	0.42	0.45
General & Administration	279	2.29	2.30	2.24
Total	6,083	62.87	53.86	48.79
Operating Margin	11,041	101.24	100.01	88.55
Operating Margin	50%	50%	51%	50%

The total pre-production and sustaining capital costs required are shown in Table 22.7.

Table 22.7 Total Project Capital Cost

US$M	Pre-Production	Sustaining	Total
Mining			
Geology	12	9	20
Mining	779	349	1,129
Capitalised Operating Costs	63	–	63
Subtotal	854	358	1,213
Concentrator & Tailings			
Concentrator	240	6	246
Capitalised Operating Costs	0.05	–	0.05
Subtotal	240	6	246
Infrastructure			
Infrastructure	230	23	253
Site Costs	9	3	11
Capitalised Operating Costs	36	–	36
Subtotal	275	26	300
Owners Cost			
Owners Cost	44	8	52
Closure	1	16	17
Subtotal	45	24	69
Capex Before Contingency	1,413	414	1,827
Contingency	131	3	135
Capex After Contingency	1,544	418	1,962

1. Sustaining capital expenditure also includes 2023 construction capital expenditure
2. Totals vary due to rounding.

22.6 Project Production and Cash Flows

Cumulative cash flow after tax is depicted in Figure 22.4. Year -5 is 2017 and Year 1 is 2022. Year 1 is the commencement of production and the plant is only in operation for part of the year commencing in Quarter 2 2022. The details of mine and process production and the project cash flow are shown in Table 22.8 to

Table 22.11 Platreef 2017 FS Concentrate Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrate Produced	kt	5,568	–	–	–	–	–	75	200	206	196	189	970	1,748	1,984
Concentrator Grade															
Platinum	g/t	38.24	–	–	–	–	–	40.10	37.62	37.41	37.40	37.99	38.21	38.13	38.54
Palladium	g/t	39.07	–	–	–	–	–	42.30	40.56	40.23	40.07	39.45	39.35	39.16	38.34
Gold	g/t	5.32	–	–	–	–	–	4.43	4.48	4.85	5.07	5.11	4.93	5.32	5.73
Rhodium	g/t	2.43	–	–	–	–	–	2.70	2.57	2.52	2.46	2.45	2.51	2.40	2.39
Copper	% Cu	3.27	–	–	–	–	–	2.44	2.68	3.07	3.20	3.14	2.93	3.40	3.47
Nickel	% Ni	5.46	–	–	–	–	–	4.07	4.46	5.13	5.42	5.33	5.03	5.68	5.68
Sulphur	% S	13.12	–	–	–	–	–	9.05	10.62	12.71	13.11	13.17	11.85	13.48	13.87
3PE + Au	g/t	85.07	–	–	–	–	–	89.53	85.23	85.00	85.00	85.00	85.00	85.00	85.00

Table 22.12 Platreef 2017 FS Metal Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Recovered Metal															
Platinum	koz	6,846	–	–	–	–	–	97	242	247	236	231	1,192	2,143	2,458
Palladium	koz	6,994	–	–	–	–	–	102	260	266	253	240	1,227	2,201	2,445
Gold	koz	952	–	–	–	–	–	11	29	32	32	31	154	299	365
Rhodium	koz	436	–	–	–	–	–	7	17	17	16	15	78	135	153
Copper	klb	01,728	–	–	–	–	–	4,045	1,779	3,923	3,841	3,095	2,585	0,899	1,561

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Nickel	klb	69,993	–	–	–	–	–	6,733	9,656	8,273	8,425	2,212	7,512	8,807	8,375
Sulphur	klb	10,633	–	–	–	–	–	4,976	6,770	7,630	6,650	4,933	8,490	9,564	6,621
3PE + Au	koz	15,228	–	–	–	–	–	216	547	562	536	517	2,651	4,777	5,421

Table 22.13.

Figure 22.4 Cumulative Cash Flow After Tax

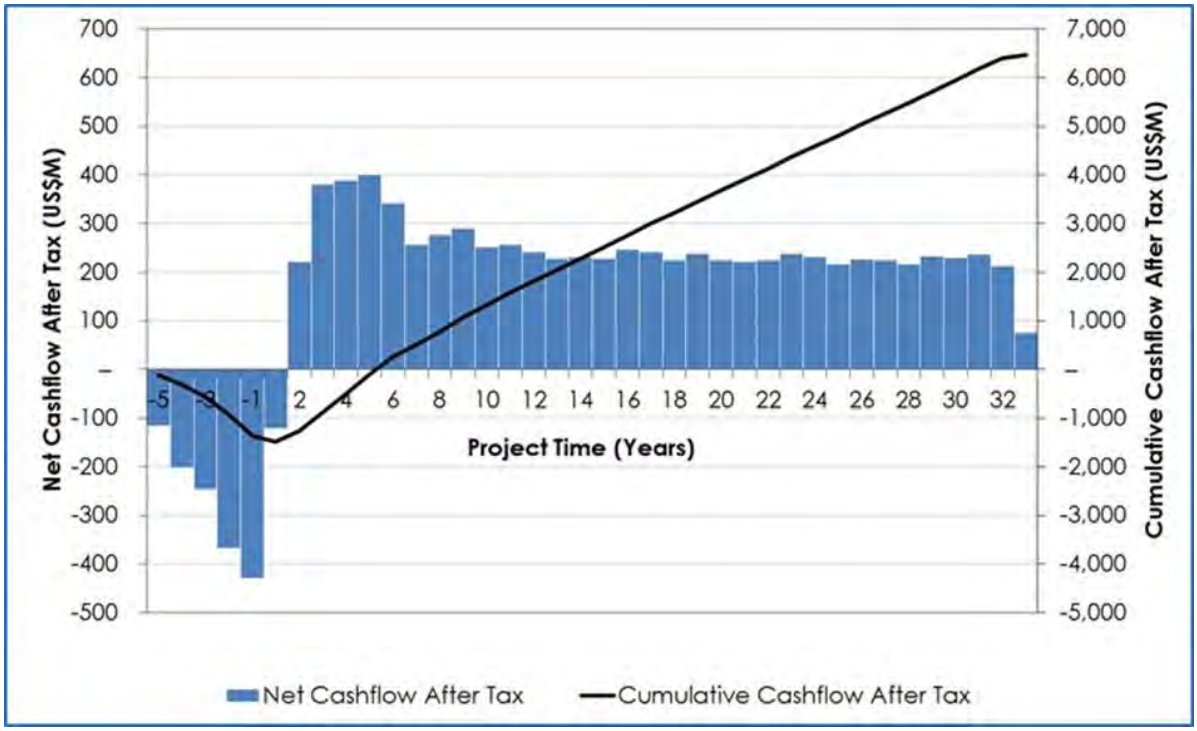


Figure by OreWin, 2017.

Table 22.8 Platreef 2017 FS Mine Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Total Ore Mined	kt	124,681	–	–	–	15	65	1,369	3,457	3,867	3,987	3,971	19,890	39,751	48,307
NSR BDT20	$/t	158	–	–	–	140	155	189	185	180	170	166	167	157	148
Platinum	g/t	1.95	–	–	–	1.68	1.88	2.50	2.34	2.22	2.07	2.05	2.10	1.92	1.84
Palladium	g/t	2.01	–	–	–	1.74	2.05	2.64	2.53	2.39	2.23	2.14	2.17	1.99	1.84
Gold	g/t	0.30	–	–	–	0.24	0.28	0.31	0.31	0.32	0.31	0.31	0.30	0.30	0.30
Rhodium	g/t	0.14	–	–	–	0.12	0.13	0.18	0.17	0.16	0.15	0.14	0.15	0.13	0.12
Copper	% Cu	0.17	–	–	–	0.16	0.17	0.16	0.17	0.18	0.18	0.17	0.16	0.17	0.16
Nickel	% Ni	0.34	–	–	–	0.33	0.34	0.32	0.35	0.37	0.36	0.35	0.34	0.34	0.33
Sulphur	% S	0.83	–	–	–	0.84	0.86	0.74	0.84	0.92	0.89	0.87	0.82	0.83	0.81
3PE + Au	g/t	4.40	–	–	–	3.79	4.33	5.63	5.35	5.08	4.76	4.64	4.73	4.34	4.10

Note NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.

Table 22.9 Platreef 2017 FS Process Production Plant Feed

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrator Feed	kt	124,681	–	–	–	–	–	1,367	3,540	3,867	3,987	3,971	19,890	9,751	48,307
NSR BDT20	$/t	158	–	–	–	–	–	186	185	180	170	166	167	157	148
Platinum	g/t	1.95	–	–	–	–	–	2.46	2.34	2.22	2.07	2.05	2.10	1.92	1.84
Palladium	g/t	2.01	–	–	–	–	–	2.60	2.53	2.39	2.23	2.14	2.17	1.99	1.84
Gold	g/t	0.30	–	–	–	–	–	0.31	0.31	0.32	0.31	0.31	0.30	0.30	0.30
Rhodium	g/t	0.14	–	–	–	–	–	0.18	0.17	0.16	0.15	0.14	0.15	0.13	0.12
Copper	% Cu	0.17	–	–	–	–	–	0.16	0.17	0.18	0.18	0.17	0.16	0.17	0.16
Nickel	% Ni	0.34	–	–	–	–	–	0.32	0.35	0.37	0.36	0.35	0.34	0.34	0.33
Sulphur	% S	0.83	–	–	–	–	–	0.74	0.84	0.92	0.89	0.87	0.82	0.83	0.81
3PE + Au	g/t	4.40	–	–	–	–	–	5.55	5.35	5.08	4.76	4.64	4.73	4.34	4.10

Note NSR is reported for BDT20. BDT20 metal prices were used in the Mineral Reserve estimate are as follows: US$1,600/oz platinum, US$815/oz palladium, US$1,300/oz gold, US$1,500/oz rhodium, US$8.90/lb nickel and US$3.00/lb copper. Metal-price assumptions used for the Platreef 2017 FS economic analysis are as follows: US$1,250/oz platinum, US$825/oz palladium, US$1,300/oz gold, US$1,000/oz rhodium, US$7.60/lb nickel and US$3.00/lb copper.

Table 22.10 Platreef 2017 FS Concentrator Recoveries

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrator Recoveries															
Platinum	%	87.37	–	–	–	–	–	89.46	90.69	89.85	88.67	88.21	88.56	87.11	86.20
Palladium	%	86.93	–	–	–	–	–	89.24	90.47	89.64	88.42	87.92	88.29	86.66	85.55
Gold	%	78.63	–	–	–	–	–	79.49	80.70	80.23	79.53	79.25	79.46	78.53	77.93
Rhodium	%	80.47	–	–	–	–	–	83.46	84.67	83.68	82.20	81.60	82.04	80.14	78.84
Copper	%	87.88	–	–	–	–	–	85.68	88.38	89.63	89.05	88.24	87.50	88.23	87.46
Nickel	%	71.88	–	–	–	–	–	69.09	72.67	74.69	74.07	72.85	72.01	72.46	70.80
Sulphur	%	70.99	–	–	–	–	–	67.38	71.38	73.52	72.65	72.18	70.75	71.20	70.49
3PE + Au	%	86.36	–	–	–	–	–	88.61	89.81	88.95	87.75	87.28	87.65	86.10	85.07
Mass Pull	%	–	–	–	–	–	–	5.49	5.64	5.32	4.92	4.76	4.88	4.40	4.11

Table 22.11 Platreef 2017 FS Concentrate Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Concentrate Produced	kt	5,568	–	–	–	–	–	75	200	206	196	189	970	1,748	1,984
Concentrator Grade															
Platinum	g/t	38.24	–	–	–	–	–	40.10	37.62	37.41	37.40	37.99	38.21	38.13	38.54
Palladium	g/t	39.07	–	–	–	–	–	42.30	40.56	40.23	40.07	39.45	39.35	39.16	38.34
Gold	g/t	5.32	–	–	–	–	–	4.43	4.48	4.85	5.07	5.11	4.93	5.32	5.73
Rhodium	g/t	2.43	–	–	–	–	–	2.70	2.57	2.52	2.46	2.45	2.51	2.40	2.39
Copper	% Cu	3.27	–	–	–	–	–	2.44	2.68	3.07	3.20	3.14	2.93	3.40	3.47
Nickel	% Ni	5.46	–	–	–	–	–	4.07	4.46	5.13	5.42	5.33	5.03	5.68	5.68
Sulphur	% S	13.12	–	–	–	–	–	9.05	10.62	12.71	13.11	13.17	11.85	13.48	13.87
3PE + Au	g/t	85.07	–	–	–	–	–	89.53	85.23	85.00	85.00	85.00	85.00	85.00	85.00

Table 22.12 Platreef 2017 FS Metal Production

	Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
	Year To												10	20	LOM
Recovered Metal															
Platinum	koz	6,846	–	–	–	–	–	97	242	247	236	231	1,192	2,143	2,458
Palladium	koz	6,994	–	–	–	–	–	102	260	266	253	240	1,227	2,201	2,445
Gold	koz	952	–	–	–	–	–	11	29	32	32	31	154	299	365
Rhodium	koz	436	–	–	–	–	–	7	17	17	16	15	78	135	153
Copper	klb	401,728	–	–	–	–	–	4,045	11,779	13,923	13,841	13,095	62,585	130,899	151,561
Nickel	klb	669,993	–	–	–	–	–	6,733	19,656	3,273	23,425	22,212	107,512	218,807	248,375
Sulphur	klb	1,610,633	–	–	–	–	–	14,976	46,770	57,630	56,650	54,933	253,490	519,564	606,621
3PE + Au	koz	15,228	–	–	–	–	–	216	547	562	536	517	2,651	4,777	5,421

Table 22.13 Platreef 2017 FS Cash Flow

Cash Flow Statement (US$M)		Year												
Year Number	Total	-5	-4	-3	-2	-1	1	2	3	4	5	6	11	21
Year To												10	20	LOM
Gross Revenue	22,298	–	–	–	–	–	289	755	806	780	750	3,785	7,073	8,060
Realisation Costs	5,174	–	–	–	–	–	54	142	151	146	141	897	1,703	1,940
Net Sales Revenue	17,124	–	–	–	–	–	235	614	655	633	610	2,889	5,370	6,119
Less: Site Operating Costs														
Mining	4,150	–	–	–	–	–	100	219	180	152	125	612	1,311	1,451
Processing & Tailings	1,303	–	–	–	–	–	18	38	42	42	42	207	413	500
Infrastructure	295	–	–	–	–	–	9	10	10	10	10	47	93	107
Site Cost	56	–	–	–	–	–	–	–	3	2	2	9	18	22
General & Administration	279	–	–	–	–	–	5	5	9	9	9	46	89	106
Subtotal	6,083	–	–	–	–	–	132	273	243	216	188	920	1,925	2,186
Operating Surplus / (Deficit)	11,041	–	–	–	–	–	103	341	411	418	421	1,969	3,445	3,934
Management Fee	182	–	–	–	–	–	4	8	7	6	6	28	58	66
Capital Costs														
Initial Capital Expenditure	1,544	116	202	247	368	429	181	–	–	–	–	–	–	–
Sustaining Capital Expenditure	418	–	–	–	–	–	–	41	13	25	18	71	122	128
Working Capital	–	–	–	–	–	–	39	69	11	-2	-3	-8	-9	-98
Subtotal	1,962	116	202	247	368	429	221	111	24	22	15	63	112	31
Net Cashflow Before Tax	8,897	-116	-202	-247	-368	-429	-122	222	380	389	401	1,878	3,275	3,837
Depreciation	1,962	–	–	–	–	–	1,544	41	13	25	18	71	122	128
Income Tax Expense	2,425	–	–	–	–	–	–	–	–	–	–	461	914	1,050
Net Cashflow After Tax	6,471	-116	-202	-247	-368	-429	-122	222	380	389	401	1,417	2,361	2,787

22.7 Price Sensitivity Analysis

A price sensitivity to metal prices analysis of the NPV8% were performed on the financial model and the results are shown in Table 22.14 to Table 22.25 and Figure 22.5 to Figure 22.16. Platinum and nickel are the major contributors to project revenue.

Table 22.14 After Tax NPV Platinum and Nickel Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/oz						
Nickel Price - US$/lb	600	800	1,000	1,250	1,400	1,600	1,800
3.00	-128	69	261	501	644	836	1,027
4.50	11	205	396	635	780	971	1,164
6.00	148	340	531	772	915	1,108	1,298
7.60	292	484	676	916	1,059	1,251	1,440
10.00	509	701	893	1,134	1,276	1,465	1,655
12.50	736	927	1,119	1,357	1,499	1,689	1,881
15.00	961	1,154	1,343	1,580	1,724	1,914	2,102

Table 22.15 After Tax NPV Sensitivity to Platinum and Nickel Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/oz						
Nickel Price - US$/lb	600	800	1,000	1,250	1,400	1,600	1,800
3.00	-114%	-93%	-71%	-45%	-30%	-9%	12%
4.50	-99%	-78%	-57%	-31%	-15%	6%	27%
6.00	-84%	-63%	-42%	-16%	-0%	21%	42%
7.60	-68%	-47%	-26%	–	16%	37%	57%
10.00	-44%	-23%	-2%	24%	39%	60%	81%
12.50	-20%	1%	22%	48%	64%	85%	105%
15.00	5%	26%	47%	73%	88%	109%	130%

Figure 22.5 Platinum and Nickel Price Sensitivity

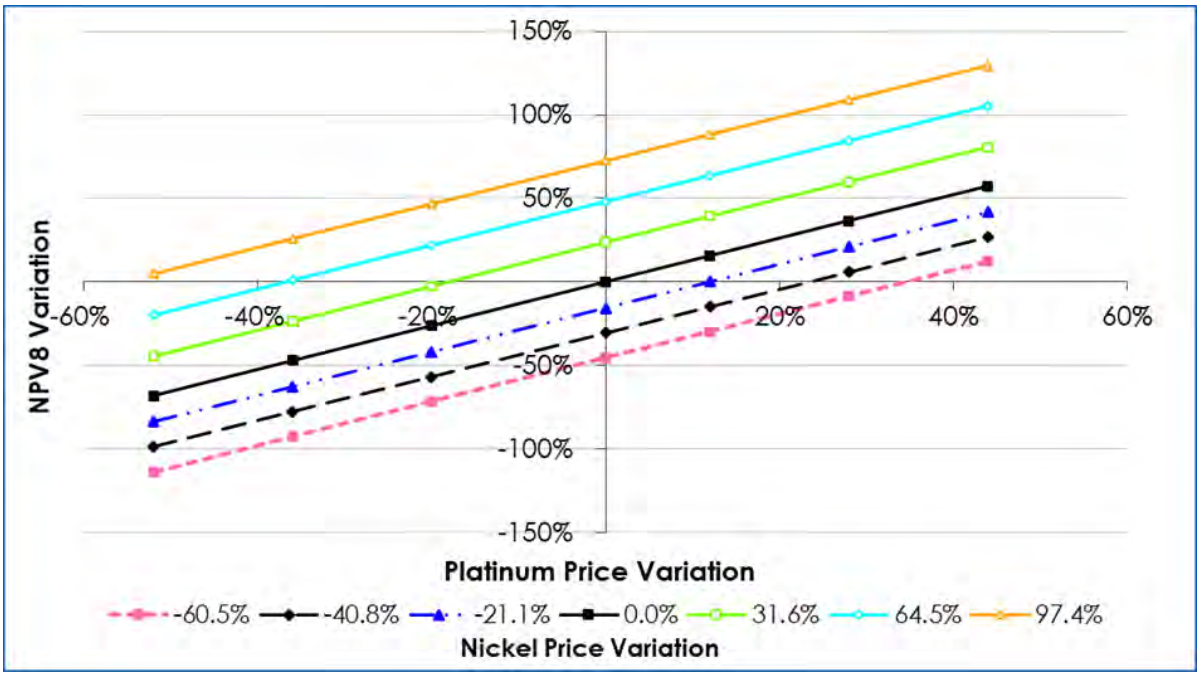


Figure by OreWin, 2017.

Figure 22.6 Nickel and Platinum Price Sensitivity

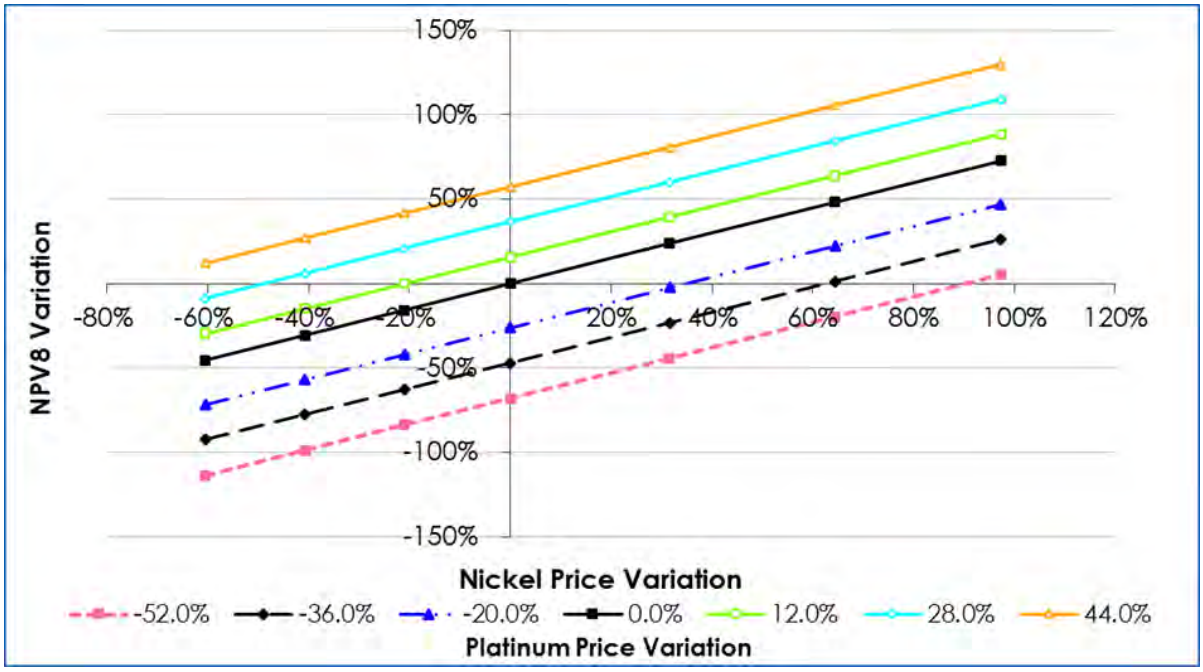


Figure by OreWin, 2017.

Table 22.16 After Tax NPV Platinum and Palladium Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/oz						
Palladium Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
300	-243	-42	153	392	536	730	921
475	-60	135	327	567	712	903	1,095
650	117	310	501	742	885	1,077	1,269
825	292	484	676	916	1,059	1,251	1,440
950	416	608	801	1,039	1,185	1,374	1,563
1,100	566	759	949	1,190	1,332	1,521	1,713
1,200	665	858	1,048	1,288	1,430	1,620	1,812

Table 22.17 After Tax NPV Sensitivity to Platinum and Palladium Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/oz						
Palladium Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
300	-127%	-105%	-83%	-57%	-41%	-20%	1%
475	-107%	-85%	-64%	-38%	-22%	-1%	20%
650	-87%	-66%	-45%	-19%	-3%	18%	39%
825	-68%	-47%	-26%	–	16%	37%	57%
950	-55%	-34%	-13%	14%	29%	50%	71%
1,100	-38%	-17%	4%	30%	45%	66%	87%
1,200	-27%	-6%	14%	41%	56%	77%	98%

Figure 22.7 Platinum and Palladium Price Sensitivity

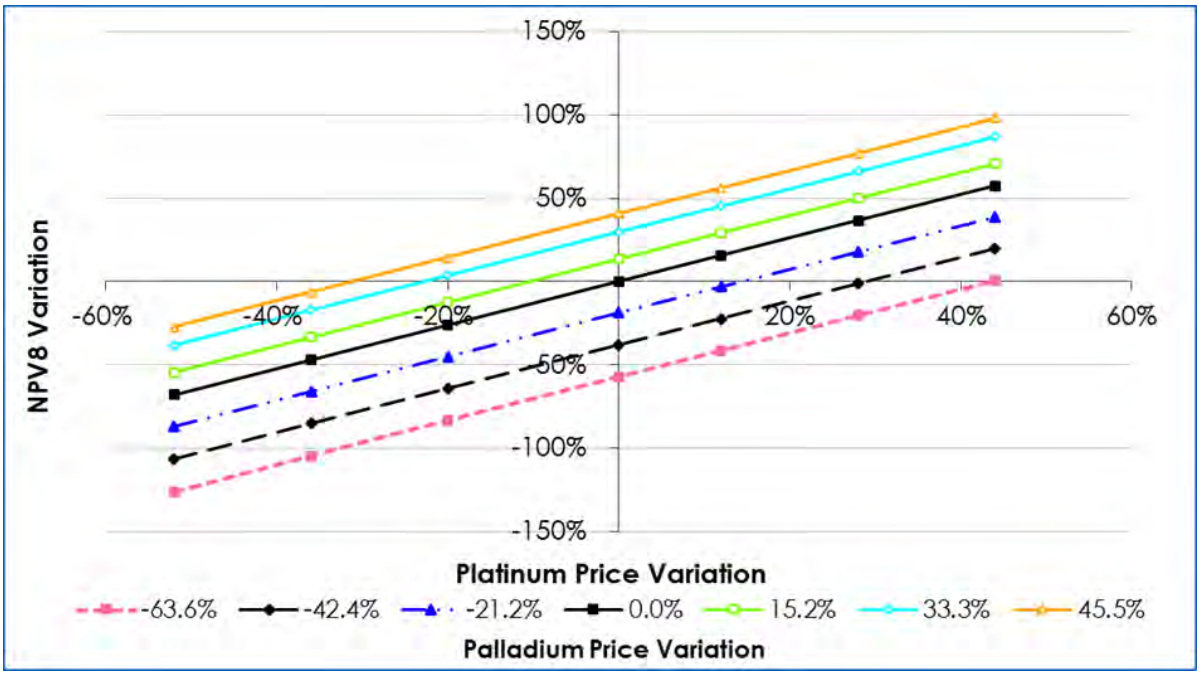


Figure by OreWin, 2017.

Figure 22.8 Palladium and Platinum Price Sensitivity

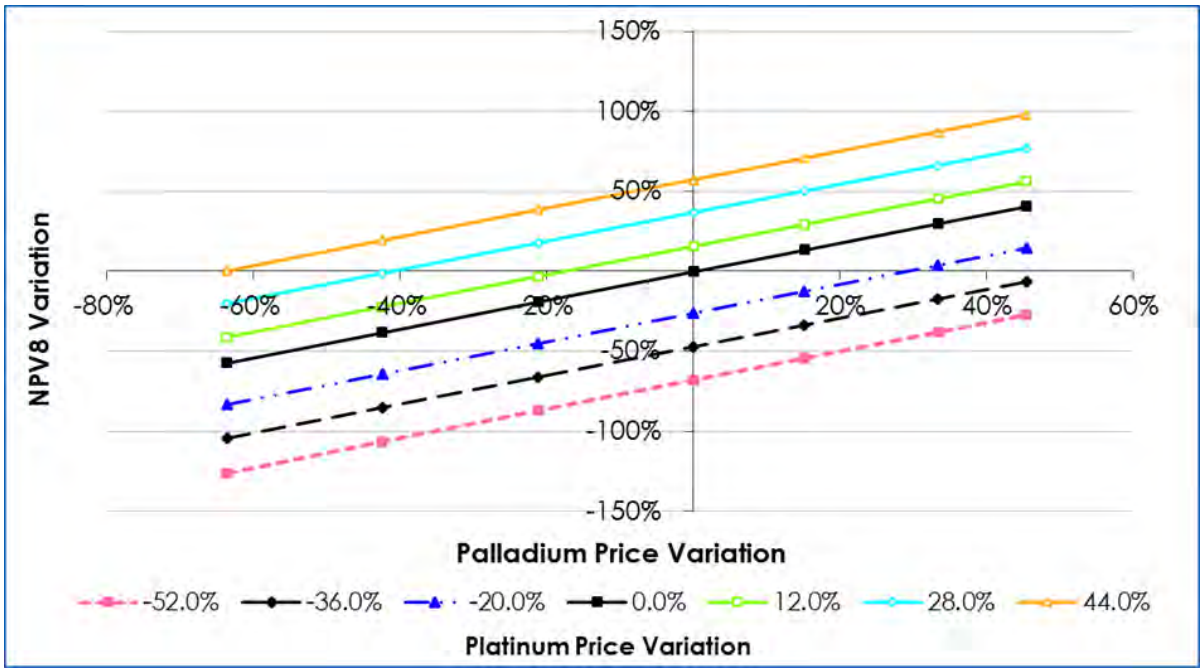


Figure by OreWin, 2017.

Table 22.18 After Tax NPV Platinum and Gold Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/oz						
Gold Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
600	203	395	586	827	970	1,163	1,352
800	229	420	612	852	995	1,188	1,377
1,000	254	446	637	878	1,021	1,213	1,402
1,300	292	484	676	916	1,059	1,251	1,440
1,450	311	503	695	935	1,078	1,270	1,459
1,650	337	528	721	960	1,105	1,295	1,484
1,800	356	547	741	979	1,124	1,314	1,503

Table 22.19 After Tax NPV Sensitivity to Platinum and Gold Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/oz						
Gold Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
600	-78%	-57%	-36%	-10%	6%	27%	48%
800	-75%	-54%	-33%	-7%	9%	30%	50%
1,000	-72%	-51%	-30%	-4%	11%	33%	53%
1,300	-68%	-47%	-26%	–	16%	37%	57%
1,450	-66%	-45%	-24%	2%	18%	39%	59%
1,650	-63%	-42%	-21%	5%	21%	41%	62%
1,800	-61%	-40%	-19%	7%	23%	44%	64%

Figure 22.9 Platinum and Gold Price Sensitivity

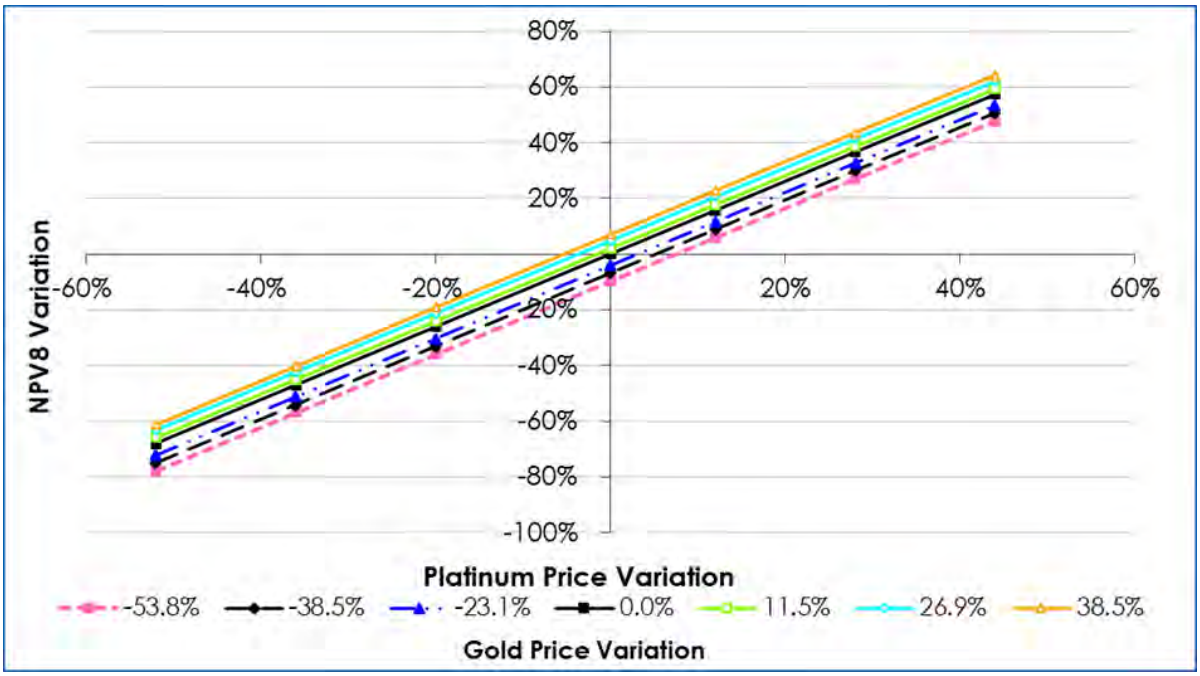


Figure by OreWin, 2017.

Figure 22.10 Gold and Platinum Price Sensitivity

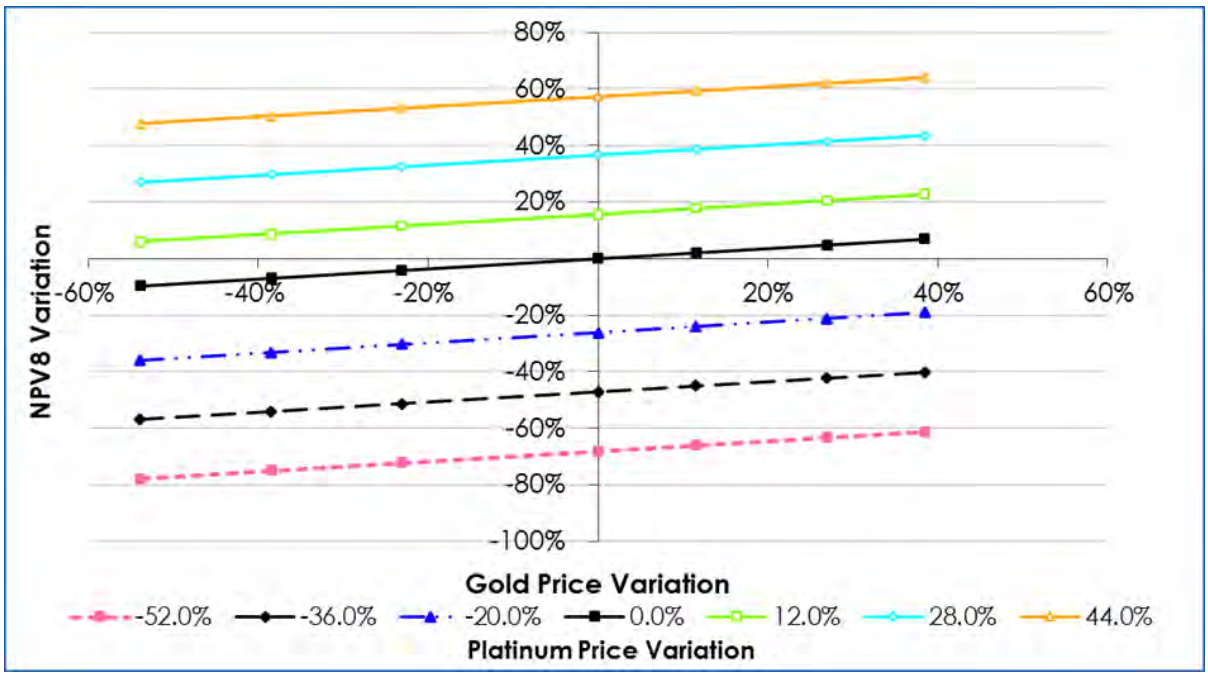


Figure by OreWin, 2017.

Table 22.20 After Tax NPV Platinum and Rhodium Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/oz						
Rhodium Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
550	264	456	648	888	1,031	1,223	1,413
700	274	465	657	897	1,040	1,233	1,422
850	283	475	666	906	1,049	1,242	1,431
1,000	292	484	676	916	1,059	1,251	1,440
1,500	323	515	708	947	1,091	1,282	1,471
2,500	385	577	770	1,009	1,154	1,343	1,532
3,000	416	608	801	1,039	1,185	1,374	1,563

Table 22.21 After Tax NPV Sensitivity to Platinum and Rhodium Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/oz						
Rhodium Price - US$/oz	600	800	1,000	1,250	1,400	1,600	1,800
550	-71%	-50%	-29%	-3%	13%	34%	54%
700	-70%	-49%	-28%	-2%	14%	35%	55%
850	-69%	-48%	-27%	-1%	15%	36%	56%
1,000	-68%	-47%	-26%	–	16%	37%	57%
1,500	-65%	-44%	-23%	3%	19%	40%	61%
2,500	-58%	-37%	-16%	10%	26%	47%	67%
3,000	-55%	-34%	-13%	14%	29%	50%	71%

Figure 22.11 Platinum and Rhodium Price Sensitivity

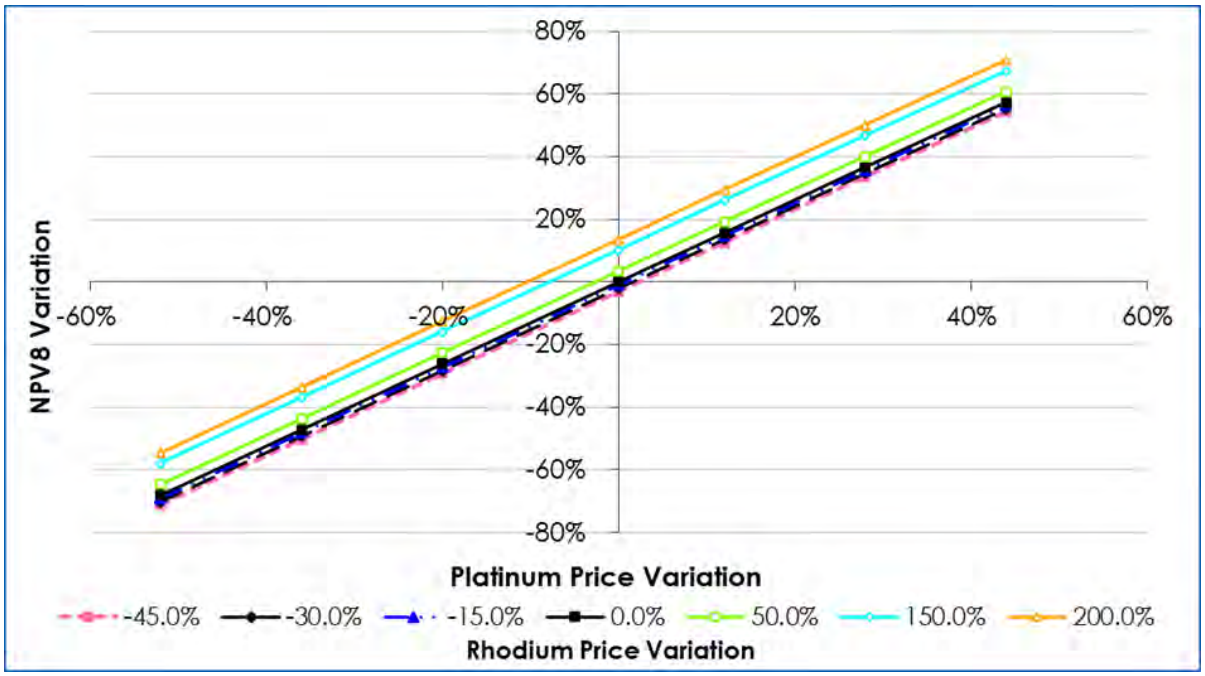


Figure by OreWin 2017.

Figure 22.12 Rhodium and Platinum Price Sensitivity

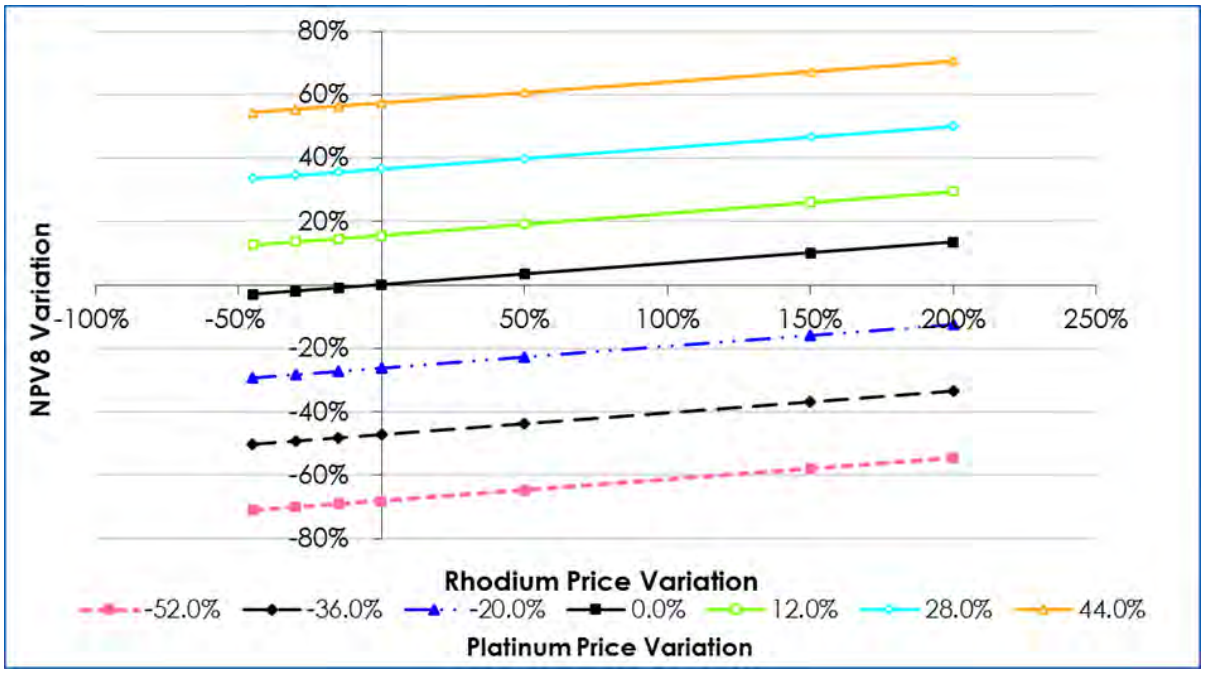


Figure by OreWin 2017.

Table 22.22 After Tax NPV Platinum and Copper Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/oz						
Copper Price - US$/lb	600	800	1,000	1,250	1,400	1,600	1,800
1.50	212	403	595	836	979	1,172	1,361
2.00	239	430	622	862	1,005	1,198	1,387
2.50	266	457	649	889	1,032	1,225	1,414
3.00	292	484	676	916	1,059	1,251	1,440
4.00	346	537	730	969	1,114	1,304	1,493
5.00	399	591	784	1,022	1,168	1,357	1,546
6.00	453	645	837	1,076	1,221	1,410	1,599

Table 22.23 After Tax NPV Sensitivity to Platinum and Copper Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/oz						
Copper Price - US$/lb	600	800	1,000	1,250	1,400	1,600	1,800
1.50	-77%	-56%	-35%	-9%	7%	28%	49%
2.00	-74%	-53%	-32%	-6%	10%	31%	52%
2.50	-71%	-50%	-29%	-3%	13%	34%	54%
3.00	-68%	-47%	-26%	–	16%	37%	57%
4.00	-62%	-41%	-20%	6%	22%	42%	63%
5.00	-56%	-35%	-14%	12%	28%	48%	69%
6.00	-51%	-30%	-9%	18%	33%	54%	75%

Figure 22.13 Platinum and Copper Price Sensitivity

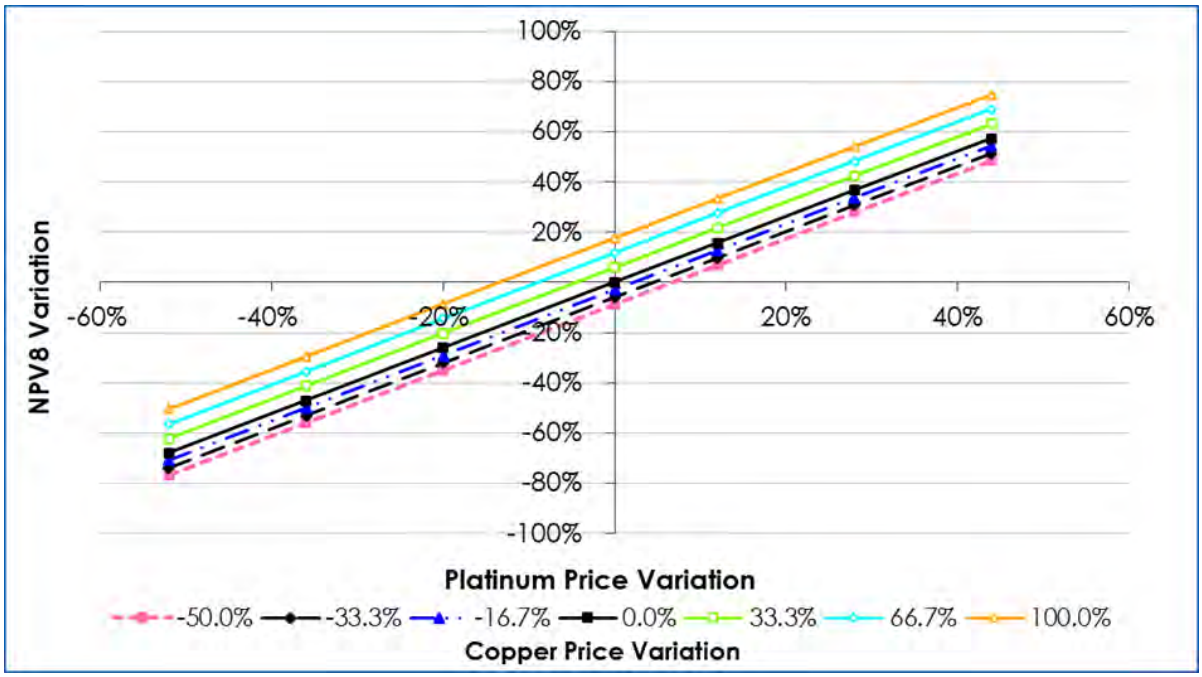


Figure by OreWin, 2017.

Figure 22.14 Copper and Platinum Price Sensitivity

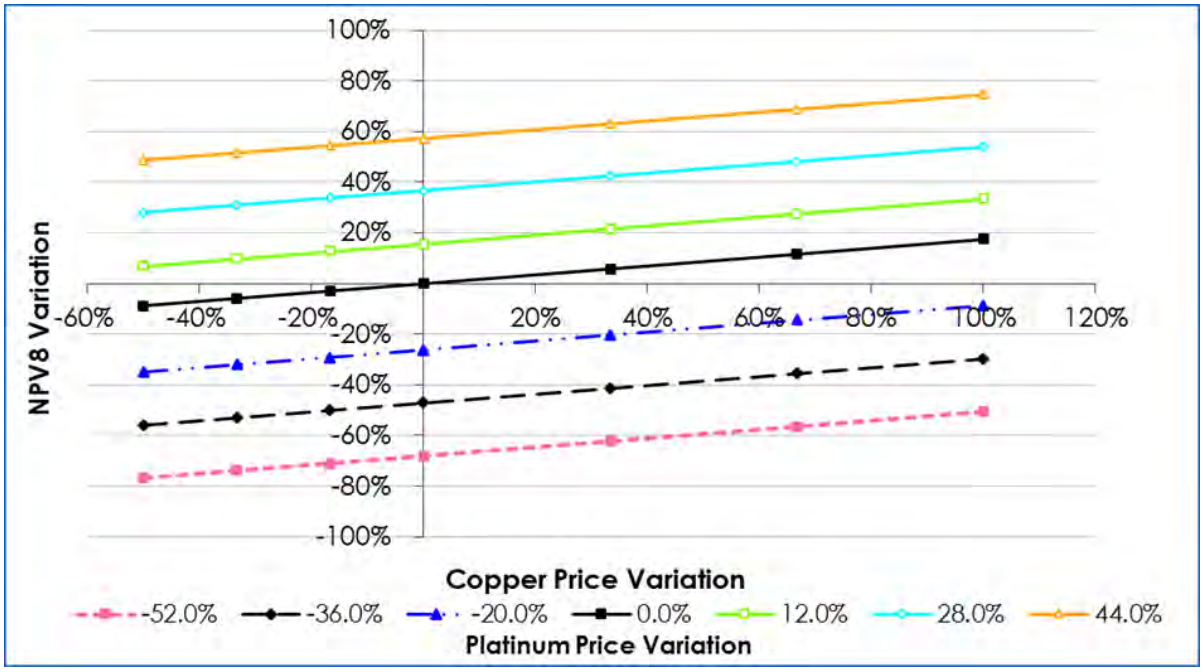


Figure by OreWin, 2017.

Table 22.24 After Tax NPV Nickel and Copper Price Sensitivity

After Tax NPV8 US$M	Nickel Price - US$/lb						
Copper Price - US$/lb	3.00	4.50	6.00	7.60	10.00	12.50	15.00
1.50	421	555	691	836	1,051	1,277	1,500
2.00	448	582	718	862	1,078	1,304	1,527
2.50	474	609	745	889	1,106	1,330	1,553
3.00	501	635	772	916	1,134	1,357	1,580
4.00	554	690	825	969	1,187	1,410	1,633
5.00	607	744	879	1,022	1,240	1,463	1,687
6.00	661	797	932	1,076	1,293	1,516	1,741

Table 22.25 After Tax NPV Sensitivity to Nickel and Copper Price Proportional Change

After Tax NPV8 US$M	Nickel Price - US$/lb						
Copper Price - US$/lb	3.00	4.50	6.00	7.60	10.00	12.50	15.00
1.50	-54%	-39%	-25%	-9%	15%	40%	64%
2.00	-51%	-36%	-22%	-6%	18%	42%	67%
2.50	-48%	-34%	-19%	-3%	21%	45%	70%
3.00	-45%	-31%	-16%	–	24%	48%	73%
4.00	-40%	-25%	-10%	6%	30%	54%	78%
5.00	-34%	-19%	-4%	12%	35%	60%	84%
6.00	-28%	-13%	2%	18%	41%	66%	90%

Figure 22.15 Nickel and Copper Price Sensitivity

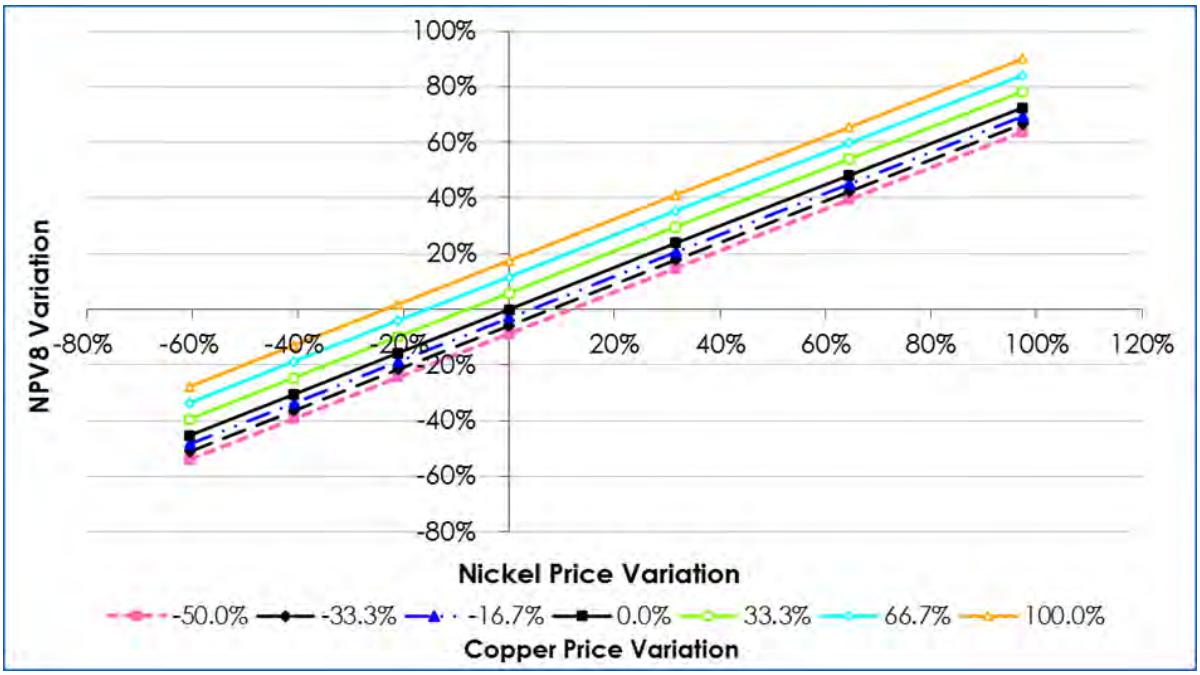


Figure by OreWin, 2017.

Figure 22.16 Copper and Nickel Price Sensitivity

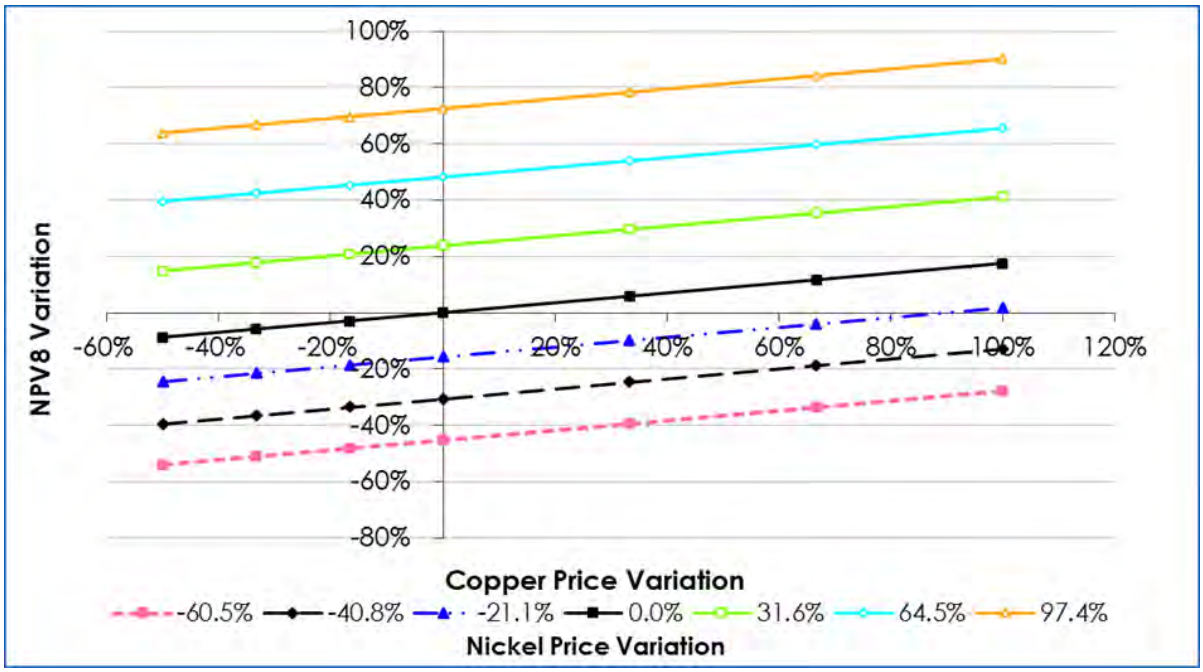


Figure by OreWin, 2017.

22.8 Refining Charges Sensitivity Analysis

A price sensitivity to metal refining charges analysis of the NPV8% was performed on the financial model with the results shown in Table 22.24 and Table 22.25 also, Figure 22.17and Figure 22.18. Results indicate that an increase in metal refining charges from the assumed 18% will lead to a decrease in After Tax NPV8.

Table 22.26 After Tax NPV Refining Charge Sensitivity

After Tax NPV8 US$M	Nickel and Copper Refining Charges						
PGE Metals Refining Charges	10%	15%	18%	20%	25%	30%	35%
10%	1,218	1,167	1,136	1,115	1,062	1,010	959
15%	1,080	1,028	997	977	925	874	823
18%	998	946	916	895	844	792	741
20%	943	892	861	841	789	738	685
25%	807	756	725	704	651	599	548
30%	669	617	586	566	514	463	411
35%	532	481	450	428	377	326	274

Table 22.27 After Tax NPV Refining Charge Sensitivity Proportional Change

After Tax NPV8 US$M	Nickel and Copper Refining Charges						
PGE Metals Refining Charges	10%	15%	18%	20%	25%	30%	35%
10%	33%	27%	24%	22%	16%	10%	5%
15%	18%	12%	9%	7%	1%	-5%	-10%
18%	9%	3%	–	-2%	-8%	-13%	-19%
20%	3%	-3%	-6%	-8%	-14%	-19%	-25%
25%	-12%	-17%	-21%	-23%	-29%	-35%	-40%
30%	-27%	-33%	-36%	-38%	-44%	-49%	-55%
35%	-42%	-48%	-51%	-53%	-59%	-64%	-70%

Figure 22.17 Nickel and Copper Refining Charge Sensitivity

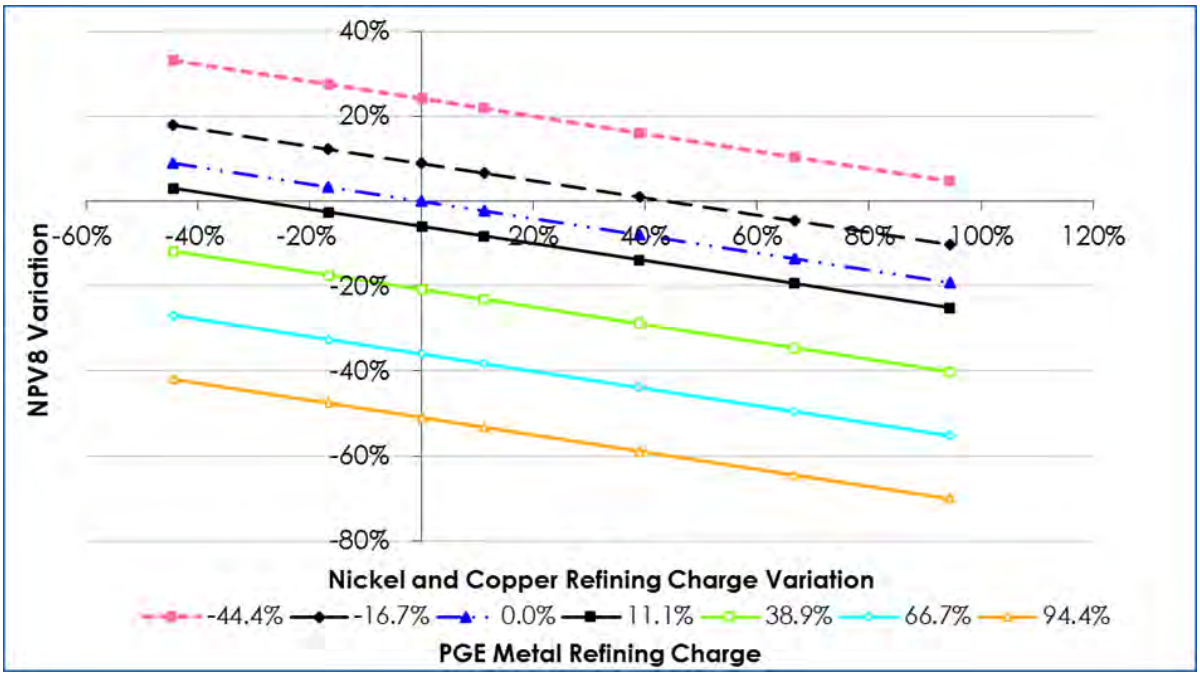


Figure by OreWin, 2017.

Figure 22.18 PGE Refining Charge Sensitivity

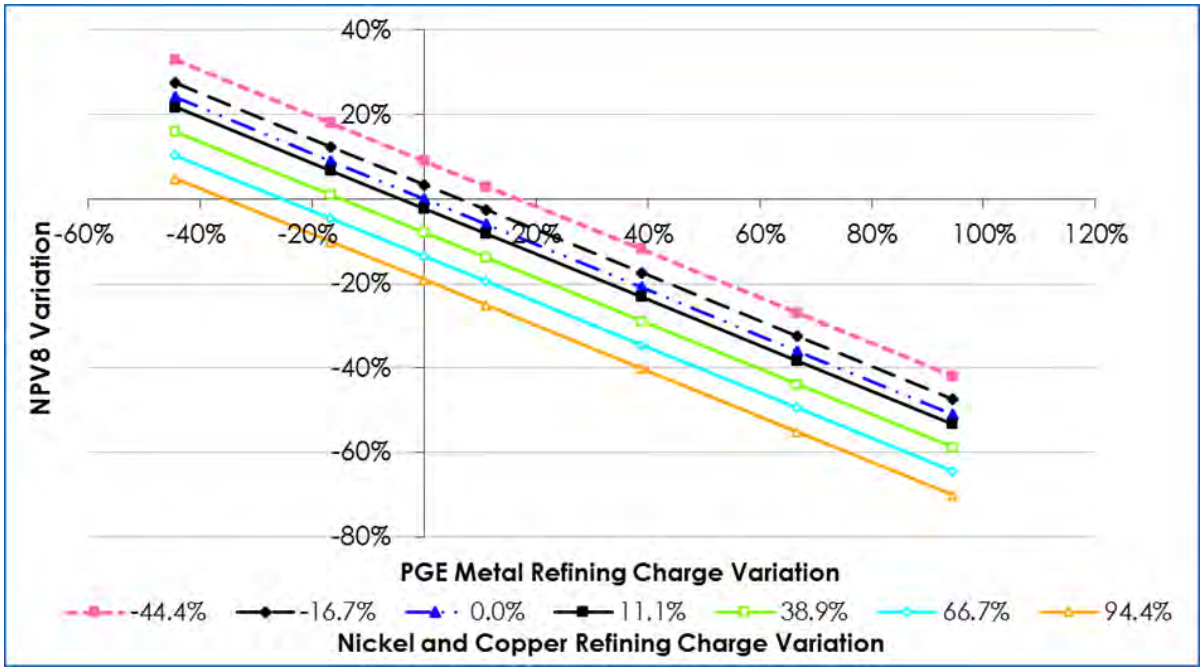


Figure by OreWin, 2017.

22.9 Exchange Rate Sensitivity Analysis

The base currency for the Platreef Project is in United States Dollars. Costs estimated in South African Rand were converted with an exchange rate of R13 to US$1. A currency exchange rate sensitivity analysis was performed on the financial model and the results are shown in Table 22.28 and Figure 22.19. The analysis indicates that an increase in the exchange rate to 16 Rand/US$ results in a 38% increase in the After Tax NPV8.

Table 22.28 Exchange Rate Sensitivity

South African Rand to US$	7.00	9.00	11.00	13.00	16.00	19.00	21.00
After Tax NPV8% US$M	-711	94	580	916	1,259	1,493	1,609
Difference US$M	-1,627	-822	-336	–	343	577	693
Difference %	-178%	-90%	-37%	–	38%	63%	76%

Figure 22.19 Exchange Rate Sensitivity

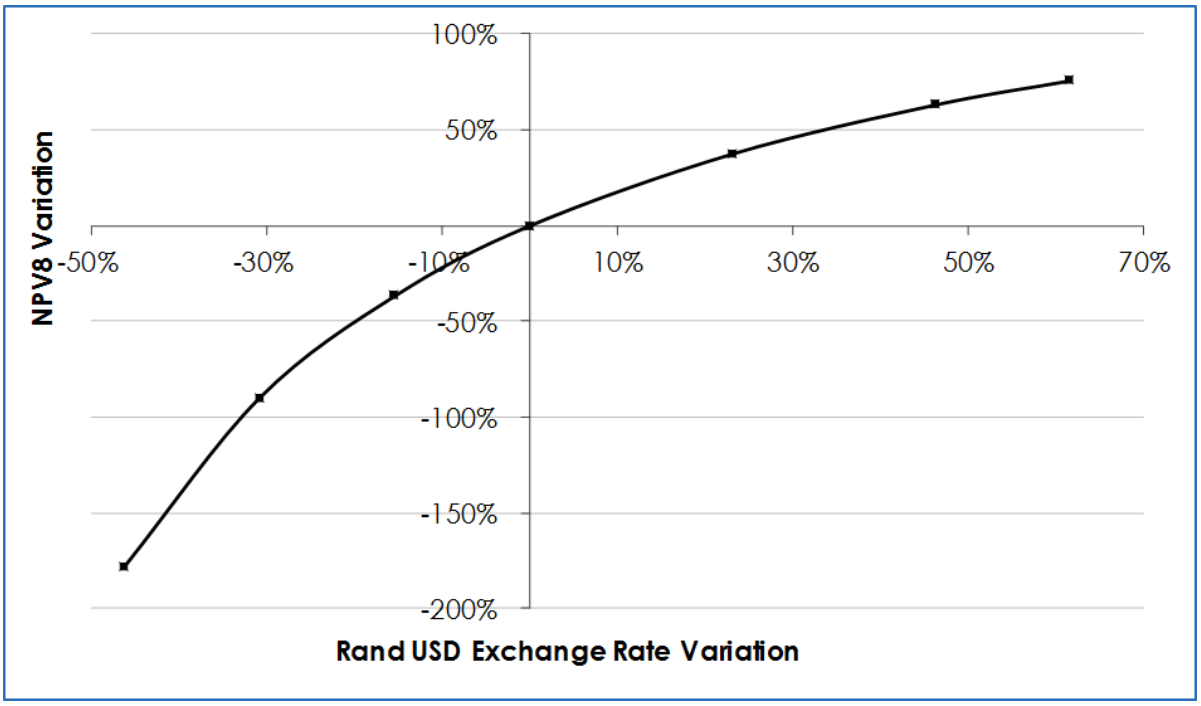


Figure by OreWin, 2017.

22.10 Cost Sensitivity Analysis

Cost sensitivity analyses of the After Tax NPV8% were examined for the pre-production capital, sustaining capital, and operating costs.

The pre-production capital cost sensitivity analysis shows the After Tax NPV8% of US$1,142 million at a 25% decrease in the pre-production capital cost and the After Tax NPV8% of US$686 million at a 25% increase in the pre-production capital cost. The results of the pre-production capital cost sensitivity analysis are shown in Table 22.29 and Figure 22.20.

Table 22.29 Pre-Production Capital Cost Sensitivity

Pre-production Capital Cost	-25%	-20%	-10%	0%	10%	20%	25%
After Tax NPV8% US$M	1,142	1,098	1,007	916	825	732	686
Difference US$M	226	182	91	–	-91	-184	-230
Difference %	25%	20%	10%	0%	-10%	-20%	-25%

Figure 22.20 Pre-Production Capital Cost Sensitivity

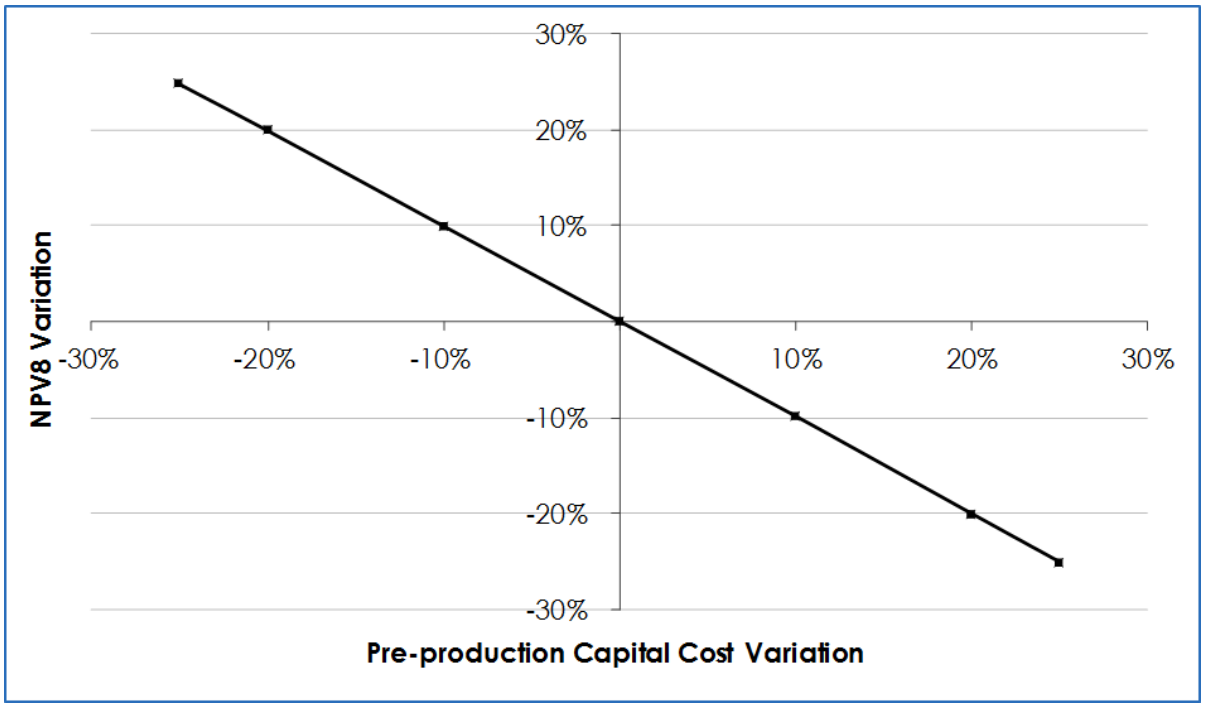


Figure by OreWin, 2017.

The sustaining capital cost sensitivity analysis shows the After Tax NPV8% of US$933 million at a 25% decrease in the sustaining capital cost and the After Tax NPV8% of US$898 million 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 22.21.

Table 22.30 Sustaining Capital Cost Sensitivity

Sustaining Capital Cost	-25%	-20%	-10%	0%	10%	20%	25%
After Tax NPV8% US$M	933	930	923	916	909	902	898
Difference US$M	17	14	7	–	-7	-14	-17
Difference %	1.9%	1.5%	0.8%	0.0%	-0.8%	-1.5%	-1.9%

Figure 22.21 Sustaining Capital Cost Sensitivity

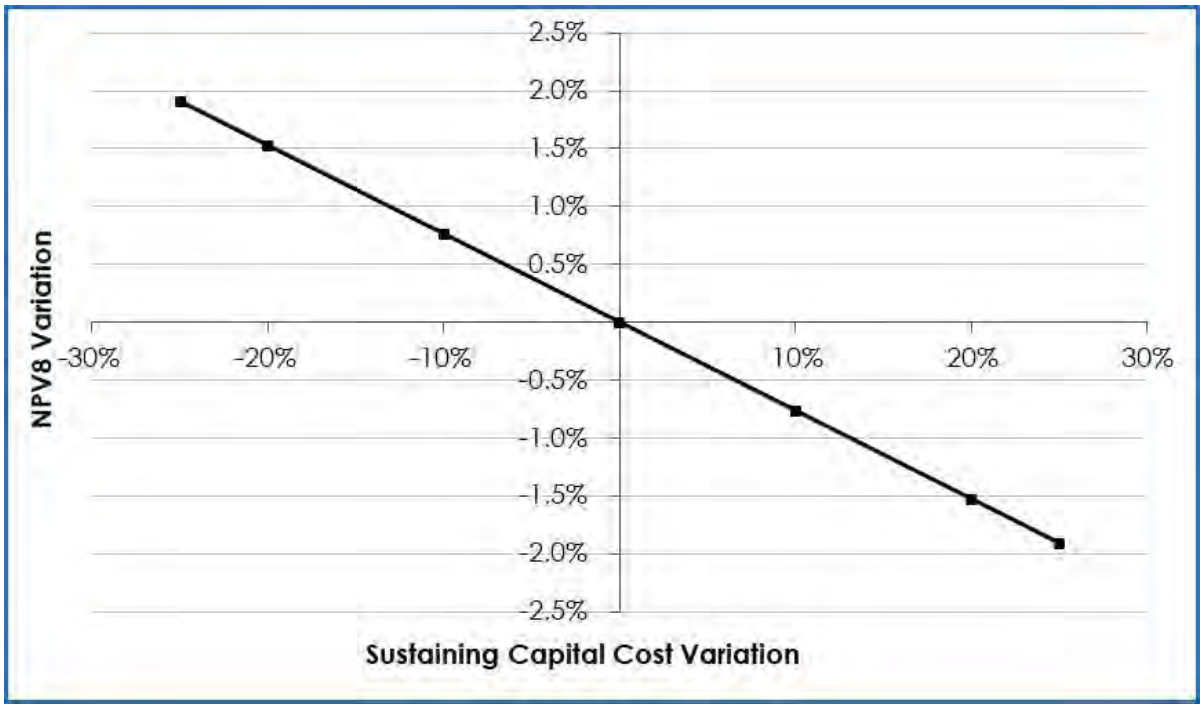


Figure by OreWin, 2017.

The operating cost sensitivity analysis shows the After Tax NPV8% of US$1,221 million at a 25% decrease in the operating cost and the After Tax NPV8% of US$617 million at a 25% increase in the operating cost. The results of the operating cost sensitivity analysis are shown in Table 22.31 and Figure 22.22.

Table 22.31 Operating Cost Sensitivity

Operating Cost Sensitivity	-25%	-20%	-10%	0%	10%	20%	25%
After Tax NPV8% US$M	1,221	1,161	1,037	916	795	675	617
Difference US$M	305	245	121	–	-121	-240	-299
Difference %	33%	27%	13%	0%	-13%	-26%	-33%

Figure 22.22 Operating Cost Sensitivity

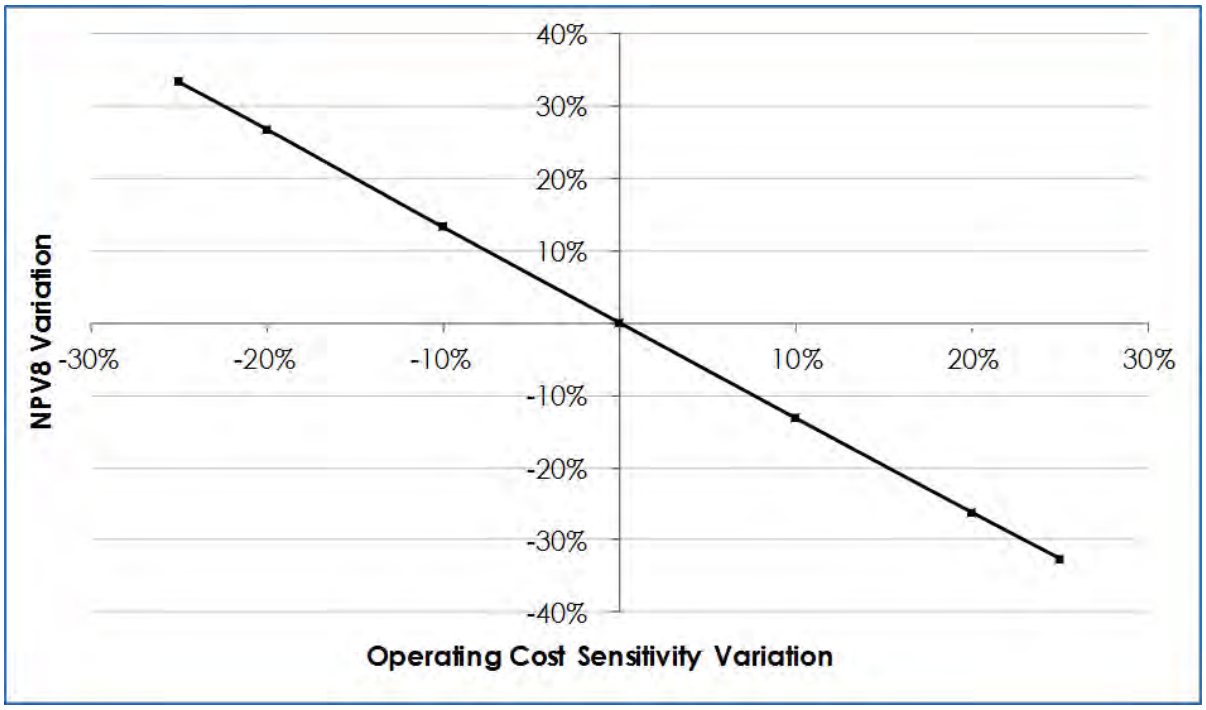


Figure by OreWin, 2017.

22.11 Comparison to Platreef 2015 PFS

A comparison of the Platreef 2017 FS and Platreef 2015 PFS financial models was carried out. The Platreef 2017 FS estimates of cash flows were prepared on a real basis as at 1 January 2017 to calculate NPV and the Platreef 2015 PFS estimates cash flows were prepared on a real basis as at 1 January 2015. There is some shifting in the relative start dates schedule between the two studies and the production, mine and plant designs and costs have been increased to FS accuracy. Although the comparison between the two studies in not an exact match, the analysis indicates that the Platreef 2017 FS has a greater NPV when compared using the same metal prices. The after tax financial results in 2017 FS and 2015 PFS is shown in Table 22.32.

The mine plan in the Platreef 2017 FS has been improved over the Platreef 2015 PFS by targeting higher-grade zones. This has increased the overall LOM grade and the grade profile in the early production years. It has also modified the top cut development approach to dramatically reduce the amount of ore development on the critical path, and the ratio of ore development to Longhole Stoping. This improvement in the grade profile has meant that the results of the economic analysis in the Platreef 2017 FS are similar to those of the Platreef 2015 PFS even though there has been a reduction in the long-term metal prices and changes in costs.

Table 22.32 After Tax Financial Results Comparison

	Discount Rate	2017 FS	2015 PFS
Net Present Value ($US M's)	Undiscounted	6,471	6,981
	5.0%	1,961	2,113
	8.0%	916	972
	10.0%	500	519
	12.0%	217	210
	15.0%	-57	-86
	20.0%	-291	-336
Internal Rate of Return		14.2%	13.4%
Project Payback Period (Years)		5.3	6.8

The 2017 FS and 2015 PFS metal prices are based on a review of consensus price forecasts from a financial institutions and similar studies. A comparison of the base and precious metal prices in 2017 FS and 2015 PFS is presented in Table 22.33.

Table 22.33 Metal Prices Comparison

Parameter	Unit	2017 FS	2015 PFS
Platinum	US$/oz	1,250	1,630
Palladium	US$/oz	825	815
Gold	US$/oz	1,300	1,300
Rhodium	US$/oz	1,000	2,000
Copper	US$/lb	3.00	3.00
Nickel	US$/lb	7.60	8.90

The key production summary comparison is shown in Table 22.34. A comparison of the Pre-Production Capital Costs, Sustaining Capital Costs and total Capital Costs in 2017 FS and 2015 PFS are shown in

Table 22.35 to Table 22.37. A comparison of the mine site cash costs in 2017 FS and 2015 PFS is summarised in Table 22.38.

Table 22.34 Production Summary Comparison

Item	Unit	2017 FS	2015 PFS
Mined and Processed	Mt	125	120
Platinum	g/t	1.95	1.76
Palladium	g/t	2.01	1.87
Gold	g/t	0.30	0.26
Rhodium	g/t	0.14	0.13
3PE + Au	g/t	4.40	4.02
Copper	% Cu	0.17	0.15
Nickel	% Ni	0.34	0.32
Concentrator Recoveries			
Platinum	%	87.4	87.2
Palladium	%	86.9	86.9
Gold	%	78.6	76.7
Rhodium	%	80.5	92.0
Copper	%	87.9	87.7
Nickel	%	71.9	68.8
Concentrate Produced	kt (dry)	5,568	4,915
Platinum	g/t	38.24	37.51
Palladium	g/t	39.07	39.84
Gold	g/t	5.32	4.81
Rhodium	g/t	2.43	2.83
3PE + Au	g/t	85.07	85.00
Copper	% Cu	3.27	3.30
Nickel	% Ni	5.46	5.42
Recovered Metal			
Platinum	koz	6,846	5,927
Palladium	koz	6,994	6,295
Gold	koz	952	761
Rhodium	koz	436	448
3PE + Au	koz	15,228	13,431
Copper	Mlb	402	358
Nickel	Mlb	670	588

Table 22.35 Pre-Production Capital Costs Comparison

Item	Unit	2017 FS	2015 PFS
Mining			
Geology	US$M	12	3
Mining	US$M	779	626
Capitalised Operating Costs	US$M	63	35
Subtotal	US$M	854	664
Concentrator & Tailings			
Concentrator	US$M	240	123
Capitalised Operating Costs	US$M	0.05	–
Subtotal	US$M	240	123
Infrastructure			
Infrastructure	US$M	230	115
Site Costs	US$M	9	8
Capitalised Operating Costs	US$M	36	–
Subtotal	US$M	275	123
Owners Cost			
Owners Cost	US$M	44	144
Closure	US$M	1	–
Subtotal	US$M	45	144
Capex Before Contingency	US$M	1,413	1,054
Contingency	US$M	131	114
Capex After Contingency	US$M	1,544	1,168

Table 22.36 Sustaining Capital Costs Comparison

Item	Unit	2017 FS	2015 PFS
Mining			
Geology	US$M	9	0.4
Mining	US$M	349	970
Capitalised Operating Costs	US$M	–	–
Subtotal	US$M	358	970
Concentrator & Tailings			
Concentrator	US$M	6	211
Capitalised Operating Costs	US$M	–	–
Subtotal	US$M	6	211
Infrastructure			
Infrastructure	US$M	23	63
Site Costs	US$M	3	1
Capitalised Operating Costs	US$M	–	–
Subtotal	US$M	26	64
Owners Cost			
Owners Cost	US$M	8	27
Closure	US$M	16	18
Subtotal	US$M	24	45
Capex Before Contingency	US$M	414	1,291
Contingency	US$M	3	110
Capex After Contingency	US$M	418	1,401

Note: Waste development was considered as sustaining capital in Platreef 2015 PFS but considered as an operating cost in the Platreef 2017 FS.

Table 22.37 Total Capital Costs Comparison

Item	Unit	2017 FS	2015 PFS
Mining			
Geology	US$M	20	4
Mining	US$M	1,129	1,596
Capitalised Operating Costs	US$M	63	35
Subtotal	US$M	1,213	1,634
Concentrator & Tailings			
Concentrator	US$M	246	334
Capitalised Operating Costs	US$M	0.05	–
Subtotal	US$M	246	334
Infrastructure			
Infrastructure	US$M	253	178
Site Costs	US$M	11	9
Capitalised Operating Costs	US$M	36	–
Subtotal	US$M	300	187
Owners Cost			
Owners Cost	US$M	52	171
Closure	US$M	17	18
Subtotal	US$M	69	190
Capex Before Contingency	US$M	1,827	2,345
Contingency	US$M	135	224
Capex After Contingency	US$M	1,962	2,569

Table 22.38 Cash Costs Comparison

	Unit	LOM AVERAGE	
		2017 FS	2015 PFS
Mine Site Cash Cost	US$/oz Recovered 3PE + Au	399	401
Realisation	US$/oz Recovered 3PE + Au	340	390
Total Cash Costs Before Credits	US$/oz Recovered 3PE + Au	739	792
Nickel Credits	US$/oz Recovered 3PE + Au	334	389
Copper Credits	US$/oz Recovered 3PE + Au	79	80
Total Cash Costs After Credits	US$/oz Recovered 3PE + Au	326	322

23 ADJACENT PROPERTIES

This Section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This Section not used.

25 INTERPRETATION AND CONCLUSIONS

25.1 Platreef 2017 FS

The Platreef 2017 FS 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 2017 FS will provide the basis for detailed planning of execution activities and to update the long-term development plans for Platreef.

The mine plan in the Platreef 2017 FS has been improved over the Platreef 2015 PFS by targeting higher-grade zones. This has increased the overall LOM grade and the grade profile in the early production years. It has also modified the top cut development approach to dramatically reduce the amount of ore development on the critical path, and the ratio of ore development to Longhole Stoping. This improvement in the grade profile has meant that the results of the economic analysis in the Platreef 2017 FS are similar to those of the Platreef 2015 PFS even though there has been a reduction in the long-term metal prices and changes in costs.

25.2 Geology and Mineral Resources

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.

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 sulphides in the TCU lithologies. Platinum group minerals are mainly present as PGE–sulphides, PGE–BiTe and PGE–As alloys, that are fine-grained (< 10 µm) and may occur within base metal sulphides, on their rims, or encapsulated in silicates.

Dr Parker and Mr Kuhl are of the opinion that knowledge of the deposit settings, lithologies, mineralisation style and setting, and structural and alteration controls on mineralisation within the AMK, ATS, and UMT deposits are sufficient to support Mineral Resource estimation. For the Platreef 2017 FS only the underground-mineable UMT deposit is considered. The mineralisation delineated at the Turfspruit 241 KR, Macalacaskop 243 KR, and Rietfontein 2 KS farms is typical of Platreef-style mineralisation within the Northern Limb of the Bushveld Complex. Exploration programmes developed using the Merensky-reef analogue are appropriate to the deposit style.

The Mineral Resources are limited to areas that have been sufficiently drilled to support geological interpretation and grade estimation. There is approximately 9.4 km^2 of exploration targets and an additional 48 km^2 of prospective ground on the Turfspruit and Macalacaskop farms to the southwest of the Mineral Resources.

25.2.1 Drilling, Sampling and Data Verification

Drilling on the Platreef Project has been undertaken in two major phases. The database supporting Mineral Resource estimation, which was closed as at 26 October 2012, contains 954 core drillholes; this included 555 holes (194,591 m) from the open pit programme and 399 holes (429,657 m) from the underground programme. From 26 October 2012 to 2 October 2014, a total of 90 drillholes (72,382 m) and 59 deflections (12,269 m) have been completed at Platreef. The additional drillholes have been completed for geotechnical data, metallurgical samples, and geology/resource drilling. During December 2014, an additional six core holes (5,850 m) were drilled.

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/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.

Dr Parker and Mr Kuhl are 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

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

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

- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual economic extraction including:
  - Long-term commodity price assumptions.
  - Long-term exchange rate assumptions.
  - Assumed mining method.
  - Availability of water and power.
  - Operating and capital cost assumptions.
  - Metal recovery assumptions.
  - Concentrate grade and smelting/refining terms.
- Additional mini pilot plant test work is planned. This work may result in changes to the metallurgical recoveries and smelter payables assumptions used to evaluate reasonable prospects of eventual economic extraction.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. The current practice of using grade shells in the area drilled in detail may underestimate the variability of the grades within and in the vicinity of the T1MZ and the T2MZ, and any stope boundaries that are laid out along the 2PE+Au grade shell surface will likely not, in practice, be able to follow the exact actual surface. The consequence would be that the effects of contact dilution and ore loss could be more than is currently projected. The data on positions of grade shell boundaries should be examined to the extent possible to estimate their short-scale variability; the likely accuracies of down-hole surveys should be taken into account, and it is recognised a definitive answer may have to await exposures in underground workings.

25.3 Mineral Reserve Estimate

The mineral reserve estimate for Platreef was based on proven methods, mining practices, and modelling techniques applied to a well-defined resource block model. The cost assumptions assigned are reasonable and support the conservative cut-off grades developed by Stantec for use in defining the reserve model and supporting mine plan. Based on this assessment, the Platreef Probable Mineral Reserve will support a 30-year mine life at a production rate of 4 Mtpa.

25.4 Mining Risks and Opportunities

25.4.1 Risks

The following is a list of potential risks for Platreef:

- There could be a lack of available work force with sufficient skills to meet the specified performance rates at the stated labour rates.
- Production and schedule constraints due to shaft pillar designs.
- Possible requirement of replacement ore and waste passes; the design provides for locations but they are not currently in the design.
- Uncertainty of raiseboring to depths of 750 m and deeper in this location.
- The amount of lining required in the ventilation raises once they are reamed.
- Mining through the Tshukudu Fault.
- Timeliness of the definition drilling during the preproduction and early production stages.
- Cooling capacity during the summer peaks. This can be managed by a work rest regime and supplemental underground air conditioning units.
- Mining underneath cemented paste fill or rock fill in the sill pillars (although less than 1% of the tonnage is mined under a sill pillar).
- The amount of additional S3 support that may be required in the later years of the mine life if ground conditions are worse and mining impacts are more severe.
- Difficulty handling the low-grade material during development.
- The stability of the accesses in the secondary stopes during and after the mining of the primary stopes.

25.4.2 Opportunities and Recommendations

The following is a list of potential opportunities Platreef may experience:

- In the current design, Shaft 2 has a 6 Mtpa capacity. This, combined with the conversion of Shaft 1 to a production shaft with a 2.5 Mtpa capacity, may present an opportunity to achieve the anticipated production rate of 8 Mtpa in Phase 2 as described in the Platreef 2014 PEA (March 2014).
- Reducing the angle of the footwall and hanging wall in production stopes once the definition drilling program better defines the block model.

- Automation of production LHDs and trucks.
- Remote operation of fixed rock breakers at ore and waste truck dumps.
- Potential to incorporate electric-powered mobile equipment in an effort to reduce ventilation and associated refrigeration requirements.
- Further analysis of equipment utilisation to reduce fleet size.
- Reducing the number of individuals supporting the development crews.
- Opportunity to use production ore passes as waste passes during the development phases.
- Storing and hoisting development ore during the Shaft 1 hoisting phase.
- Optimisation of the mine air cooling system.
- Further definition and delineation of FW Mineral Resources.

25.4.3 Mining Conclusions

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

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

25.5 Metallurgy

It is the opinion of the qualified person responsible for the metallurgical aspects of the Platreef 2017 FS, Mr Val Coetzee, that an acceptable metallurgical test work 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 test work 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 test work programmes have been conducted by parties well versed in the processing of ores from the Bushveld Igneous Complex. The necessary checks and balances have been applied to ensure that the test work and chemical analysis has been conducted with the necessary diligence and accuracy.

The proposed circuit is considered to be the preferred option for the concentrator. The use of a multi-stage crusher circuit followed by a single stage milling circuit is considered to be the option of least risk to the project and is recommended for this stage of the study.

The proposed flotation circuit is based on interpretation of the results obtained from the bench-scale flotation test work. The design and specification of the various flotation stages is considered adequate for this the level study and provides sufficient flexibility required during commercial production to optimise outputs.

Observations and conclusions from the test work are accurately interpreted and included in the proposed design. The mini pilot plant test work programme that will be undertaken as part of the project implementation phase will test the robustness of the design, further optimise the various stages in the process and provide additional design parameters required for project implementation.

25.6 Infrastructure

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

The Platreef Project team has addressed the supply-demand requirements of bulk power and water to a sufficient level of detail for this study. Bulk water availability seems to be sufficient based on the level of accuracy of the study performed, however the timing when water will be available is of concern, and the delivery of the various alternative bulk water supply options must not be taken for granted. On the other hand, the bulk power supply application made to ESKOM in 2011 is not sufficient, and Ivanplats urgently needs to revise the current application to a NMD of 100 MVA.

It can be concluded that the availability of skilled labour resources, for both construction and operational phases, is limited and that the training and skills development programme will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Platreef Project.

25.6.1 Infrastructure Risks

The availability and licencing of sufficient water for the project start-up poses a risk to the Platreef Project, in that it can slow down construction and delay the start of operations. Continuous engagement with stakeholders, the appointment of dedicated consultants and active investigation into alternative sources are some of the mitigation methods being implemented by Ivanplats.

The possibility of Eskom not being able to increase the current 70 MVA power supply applied for to the required 100 MVA power demand is a risk. Engagement and negotiations with Eskom should commence as soon as possible to understand power availability in the area.

The environmental impact of the proposed water projects on water in the Olifants river system must not be underestimated. The Olifants River, a major source of water to the Kruger National Park, has in recent years been adversely affected by mining and irrigation projects in its catchment area.

25.6.2 Infrastructure Opportunities

Being on the forefront of mining development in the Bushveld Complex, the Platreef Project should investigate methods on how to use the new infrastructure upgrades in the area to the Platreef Project's advantage. One such opportunity being pursued is to include suitable changes to the N11 (off-ramps etc.) early and as part of the N11 upgrade project being developed by SANRAL.

Identification of other mining projects in the area and collaboration with such companies can mitigate risk and minimise costs for common infrastructure projects to be undertaken.

26 RECOMMENDATIONS

26.1 Platreef 2017 FS

The Platreef 2017 FS will provide the technical basis for Ivanplats to commence 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.

Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of Phases 2 and 3 should be reevaluated using the findings of the Platreef 2017 FS.

26.2 Drill Programme

Ivanhoe is planning a long-term delineation drilling program in support of detailed mine access and stope designs for the following reasons:

- Positioning of footwall access drives is dependent on the footwall contact of the mineralisation; thus, this information is based on the results of the delineation drilling and needs to be available prior to development.
- If these footwall access drives are developed too soon, they could be in the incorrect position (causing ore loss in the stopes)
- Numerous drillholes are planned from individual drill sites. The last of these drillholes must be completed prior to the footwall access drives being developed

Ivanhoe has noted that many of these considerations have not been fully incorporated into the Feasibility Study mine plan. A revised plan will be necessary prior to implementation to address concerns related to sequencing of development and time necessary to process drillhole information.

The planned delineation drilling is separated by Ivanhoe into two periods:

- A 25 m drill grid is planned for the first 10 years (2022 to 2031). 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 2032 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 mineralization, and localised variability in grade, will be better understood following the tighter drill grid and mining in previous years.

Ivanhoe has budgeted higher amounts of drilling up-front, as numerous mining areas are developed in quick succession. The drilling rate is assumed to stabilise as steady-state production is reached. From 2032, mining is planned to continue for 25 years but the delineation drilling is estimated over only 15 years as in Ivanhoe's view, there should not be a need for delineation drilling in the final years of production.

It is anticipated that approximately 24,000 m of drilling per year on average will be necessary for years 2022 to 2031, 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 M to $50 M ($127/m to $152/m) over the life of the mine and has been included in the costs of the Platreef 2017 FS.

Amec Foster Wheeler has reviewed the estimates and given the assumptions in the proposed program, considers the costs to be reasonable.

Figure 26.1 Proposed Delineation Drilling

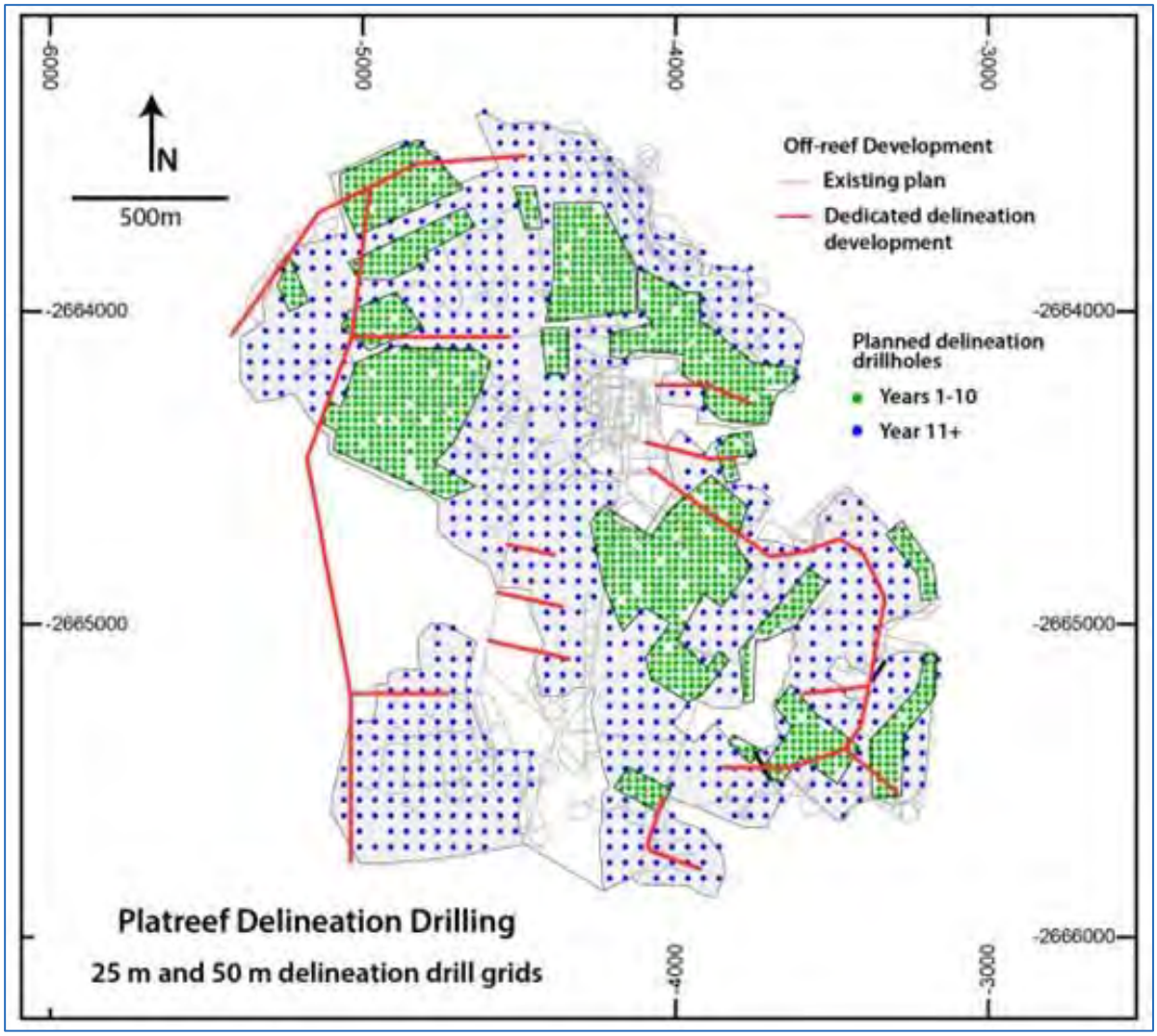


Figure courtesy Ivanhoe 2017

Figure 26.2 Perspective View for Delineation Drilling

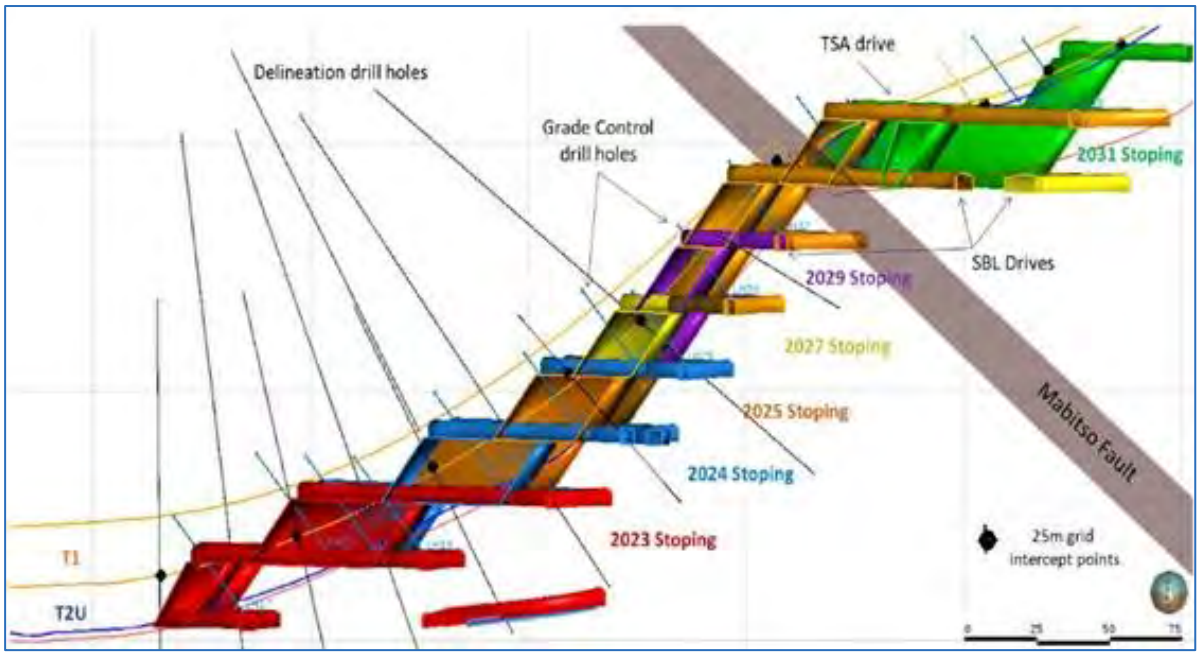


Figure courtesy Ivanhoe 2017

26.3 Resource Estimate

Amec Foster Wheeler recommends the assay tables for the ATS and, AMK drillhole campaigns in the AcQuire database be validated against supporting documents. This is required before the AMK and ATS assay data can be used for Mineral Resource estimation. This is estimated depending whether this is done internally or by a consultant to cost approximately $15,000 to $25,000.

26.4 Mineral Reserve Recommendations

The results and conclusions from the feasibility mining study are positive; as a result, Stantec recommends the advancement of the Platreef study to the execution stage. As the definition drilling data is available it should be used to update the resource model and be reviewed against the current mine design.

26.5 Mining Recommendations

Following are Stantec's recommendations regarding for additional work and modifications to the current mine plans during the project phase:

- Optimise the definition drilling program 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 optimize 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 program for ore pass monitoring to ensure the longevity of the passes.
- Set up a ground control observation program to proactively recondition the ground support as needed during the mine life.

26.6 Metallurgical Recommendations

The metallurgical test work programme has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes. To further evaluate optimisation opportunities and confirm additional detail design parameters, a mini pilot plant test work programme is proposed and will be undertaken as part of the project implementation phase. The mini plant test work programme will be conducted at Mintek in 2017 using an 8 t composite sample comprised of the remaining drill core sample from the variability drilling programme. The mini plant sample-selection process will aim to target the expected life of mine ore blend and feed grade. The mini pilot plant testing should focus on establishing and/or confirming the following:

- The effect of rougher flotation feed PSD for the pilot scale primary rod mill and secondary ball mill circuit as compared to the bench scale flotation performance using a laboratory rod mill
- Determination of the optimal high chrome grinding media specification
- Evaluate pilot-scale flotation mass pull as compared to bench scale testing
- Measuring the steady state flotation concentrate solids concentrations and confirm the expected cleaner circuit operating density
- Evaluate the impact of scavenger cleaner tails recycle
- Quantify the benefit of column flotation and establish the optimal location within the flowsheet
- Evaluate the impact of reagent build up in recycle streams and optimize reagent addition requirements
- Evaluate the effect of operating with process water sourced from Mokopane
- Make reactor scouting tests to evaluate potential opportunities for improved fines recovery
- Generate tailings sample for confirmatory thickening test work on a classified feed sample, to confirm the design parameters
- Generate sufficient cleaner concentrate for thickening and filtration test work, to confirm the design parameters

26.7 Infrastructure

Moving forward, regular interfacing and liaison with the assigned project teams representing Eskom, the Joint Water Forum and SANRAL is required 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. Memorandums of understanding are to be negotiated with regard to already identified alternative water sources.

Negotiations with Eskom are to commence as soon as possible to secure the additional power demand for a predicted average NMD of 100 MVA required to operate the Platreef mine.

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;
- Integrated Water Use License Application (IWULA) in compliance with the National Water Act (NWA); and
- Integrated Waste Management License in compliance with the National Environmental Management Waste Act (NEMWA).

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27.2 Glossary of Symbols and Units

Table 27.1 Table Of Symbols and Units

Meaning	Symbol / Unit
Minute (plane angle)	'
Second (plane angle)	"
Percent	%
Less than	<
Greater than	>
Degrees Celsius	°C
Micrometre (micron)	µm
Annum (year)	a
Billion	b
Billion pounds	blb
Billion tonnes	bt
Centimetre	cm
Square centimetre	cm^2
Cubic centimetre	cm^3

Meaning	Symbol / Unit
Day	d
Days per week	d/wk
Dry metric tonne	dmt
Gram	g
Grams per tonne	g/t
Hour (not hr)	h
Hectare (10,000 m^2)	ha
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per tonne	kg/t
Thousand hours	kh
Kilometre	km
Kilometre per hour	km/h
Kilopascal Gauge	kPa(g)
Square kilometre	km^2
Thousand Troy ounces	koz
Kilopascal	kPa
Thousand tonnes	kt
Thousand tonnes per day	kt/d
Thousand tonnes per hour	kt/h
Litre	L
Pound	lb
Metre	m
Million	M
Square metre	m^2
Cubic metre	m^3
Metres per second	m/s
Million years	Ma
Metres above (mean) sea level	masl
Milligram	mg
Million pounds	Mlb
Millimetre	mm
Millimetres per annum	mm/a
Millimetres per hour	mm/h
Million ounces	Moz
Megapascal	MPa
Million tonnes	Mt
Millivolts	mV
Ounce	oz
Parts per billion	ppb

Meaning	Symbol / Unit
Parts per million	ppm
Metric tonne (1,000 kg)	t
Tonnes per annum	t/a
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per cubic metre	t/m^3
Watts per m^2	W/m^2
Week (seven days)	wk
Wet metric tonnes	wmt

27.3 Glossary of Abbreviations and Terms

Table 27.2 Table Of Abbreviations and Terms

Description	Abbreviation / Term
Atomic Absorption Spectroscopy	AAS
Annual Information Form	AIF
Ammonium nitrate fuel oil	ANFO
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Diamond drillhole	DDH
Environmental Impact Assessment	EIA
Engineering, Procurement, and Construction Management	EPCM
Environmental and Social Impact Assessment	ESIA
Front end loader	FEL
Feasibility study	FS
General and administration	G&A
Gigabit (10^9 bits)	Gbit
Geographic information system	GIS
Internal Rate of Return	IRR
Local Area Network	LAN
Load-haul-dump	LHD
Life-of-mine	LOM
Material take-off quantity (m^3)	MTO
Non-acid forming	NAF
Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects	NI 43-101
Net Present Value	NPV
Net Present Value at 8% Discount Rate	NPV8%
Net Smelter Return	NSR
Ordinary kriging estimation method	OK
Potentially acid forming	PAF
Prefeasibility study	PFS

Description	Abbreviation / Term
Quality assurance and quality control	QA/QC
Qualified Person	QP
Reverse circulation	RC
Chinese Renminbi (also called Yuan)	RMB
Rock Mass Rating	RMR
Run of mine	ROM
Rock quality designation	RQD
Semi-autogenous grinding	SAG
Social Impact Assessment	SIA
SAG Performance Index	SPI
SAG Power Index	SPI
North seeking Gyro	SRG
Standard Reference Material	SRM
Short-wave infrared	SWIR
Time Domain Reflectometers	TDR
Total Dissolved Solids	TDS
Technical Report	TR
Tailings storage facility	TSF
Unconfined Compressive Strength	UCS
Unconfined Tensile Strength	UTS