IVANHOE MINES LTD.

Platreef Project

2016 Resource Technical Report

June 2016

IMPORTANT NOTICE

This notice is an integral component of the Platreef 2016 Resource Technical Report and should be read in its entirety and must accompany every copy made of the Platreef 2016 Resource Technical Report. The Platreef 2016 Resource Technical Report has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Platreef 2016 Resource Technical Report has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by OreWin Pty Ltd (OreWin), Amec Foster Wheeler E&C Services Inc (Amec Foster Wheeler), SRK Consulting Inc (SRK), Stantec Consulting International LLC (Stantec), DRA Projects (Pty) Ltd (DRA) as the Report Contributors. The Platreef 2016 Resource Technical Report is based on information and data supplied to the Report Contributors by Ivanhoe and other parties. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of the Report Contributors, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. Each portion of the report is intended for use by Ivanhoe subject to the terms and conditions of its contract with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the report, by any third party, is at that party's sole risk. The Platreef 2016 Resource Technical Report is intended to be used by Ivanhoe, subject to the terms and conditions of its respective contracts with the Report Contributors. Recognising that Ivanhoe has legal and regulatory obligations, the Report Contributors have consented to the filing of the Platreef 2016 Resource Technical Report with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR).

The results of the Platreef 2016 Resource Technical Report represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Platreef 2016 Resource Technical Report. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

The conclusions and estimates stated in the Platreef 2016 Resource Technical Report are to the accuracy stated in the Platreef 2016 Resource Technical Report only and rely on assumptions stated in the Platreef 2016 Resource Technical Report. The results of further work may indicate that the conclusions, estimates, and assumptions in the Platreef 2016 Resource Technical Report need to be revised or reviewed.

The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Platreef 2016 Resource Technical Report. Where the Report Contributors have made those estimates and approximations, they are subject to qualifications and assumptions, and it should also be noted that all estimates and approximations contained in the Platreef 2016 Resource Technical Report will be prone to fluctuations with time and changing industry circumstances.

The Platreef 2016 Resource Technical Report should be construed in light of the methods, procedures, and techniques used to prepare the Platreef 2016 Resource Technical Report. Sections or parts of the Platreef 2016 Resource Technical Report should not be read in isolation of or removed from their original context.

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Title Page

Project Name: Platreef Project

Title: Platreef 2016 Resource Technical Report

Location: Limpopo Province.
Republic of South Africa

Effective Date of Technical Report: 22 April 2016

Effective Date of Mineral Resources:

Mineral Resource Amenable To Selective Underground Mining Methods:	22 April 2016.
Bikkuri Mineral Resource Amenable To Selective Underground Mining Methods:	22 April 2016.
Supply of Ongoing Drill Programme Information:	24 July 2015.
Mineral Resource estimation update	22 April 2016.

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Signature Page

Project Name:	Platreef Project
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Location:	Limpopo Province. Republic of South Africa
Effective Date of Technical Report:	22 April 2016

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1 SUMMARY

1.1 Introduction

The Platreef 2016 Resource Technical Report has an effective date of 22 April 2016 and has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). The Platreef 2016 Resource Technical Report is an Independent NI 43-101 Technical Report prepared 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 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: (i) the Kamoa Project; (ii) the Platreef Project, and (iii) the Kipushi Project.

Ivanhoe has undertaken further mineral resource studies following the Platreef 2014 PFS on the Platreef Project that has formed the basis of the Platreef 2016 Resource Technical Report, which summarises the current Ivanhoe development strategy for the Platreef Project. The Platreef 2016 Resource Technical Report provides an update of the Platreef Project Mineral Resource, with the Mineral Reserve from the Platreef 2014 Prefeasibility Study (Platreef 2014 PFS) remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. The Platreef 2016 Resource Technical Report should be read in this context.

Figure 1.1 Platreef Project Location

Figure supplied by Ivanhoe, 2014.

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); ITC Platinum (ITC) an affiliate of 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 Prospecting Right (PR) LP30/5/111/2/740PR. Together, these two rights form the Platreef Project. 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.

A Mining Right (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 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 key features of the mineral resource updates are summarised in Sections 1.2 to 1.10.

1.2 Mineral Tenure and Surface Rights

1.2.1 Macalacaskop 243 KR and Turfspruit 241 KR

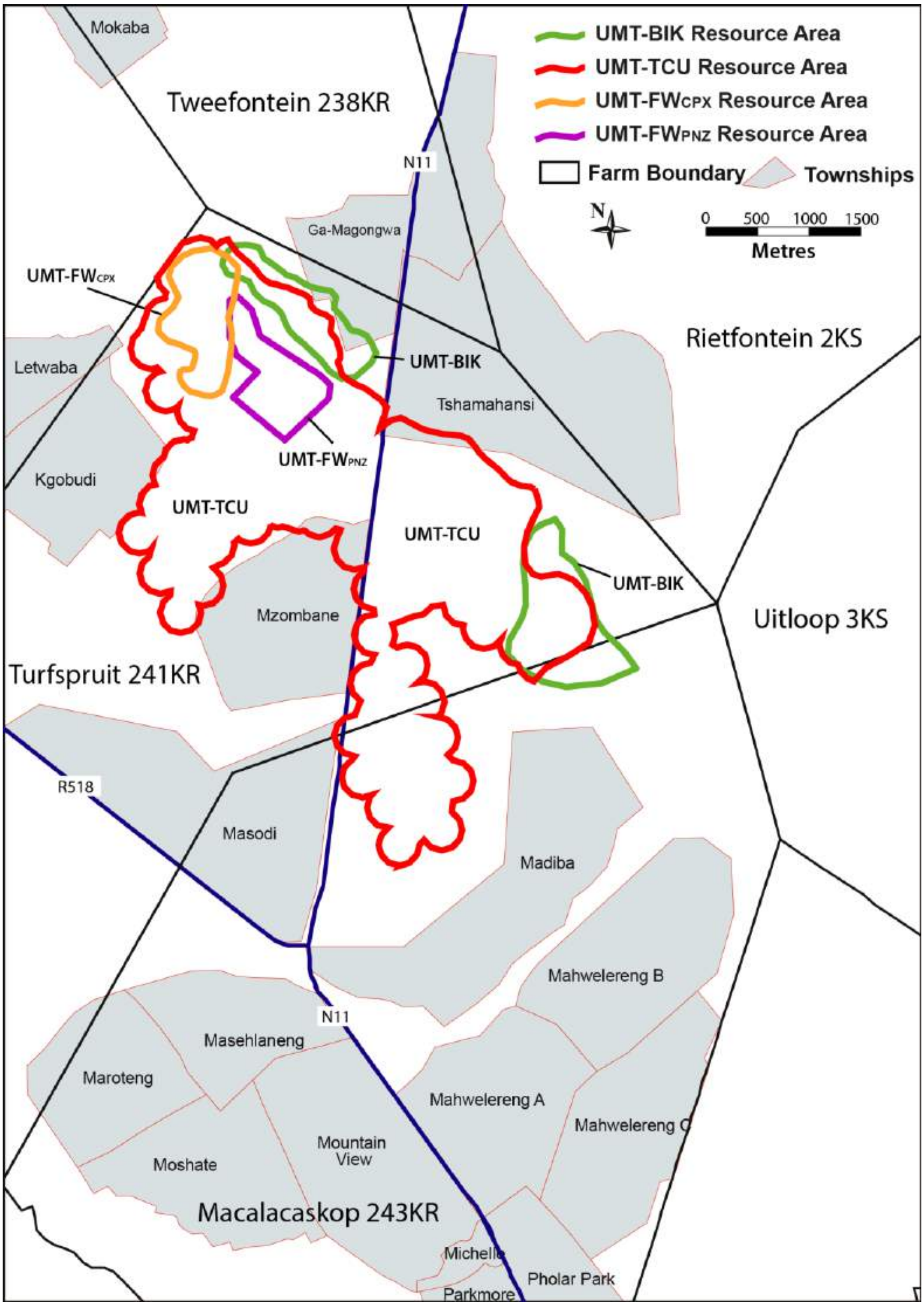
On 6 June 2013, Ivanhoe applied for a Mining Right in respect of platinum group elements (PGEs) and all associated metals and minerals mined out of necessity and convenience together the with the PGEs, over the Macalacaskop 243 KR and Turfspruit 241 KR farms. The Mining Right application was accepted by the DMR on 17 July 2013, and granted on 30 May 2014 on the eve of the expiry of the Prospecting Right which was registered in the South African Mineral and Petroleum Titles Registration Office on 9 February 2006 under registration number 55/2006 PR. On 4 November 2014, the Mining Right was notarially executed and thereby commenced.

Ivanplats (Pty) Limited, registration number 1988/000334/07, now holds the sole and exclusive right to mine and recover the minerals in, on, or under the mining area being the Turfspruit 241 KR and Macalacaskop 243 KR farms, comprising a combined 7,841 ha in extent, under a Mining Right.

1.2.2 Rietfontein

Atlatsa Resources Corporation (Atlatsa; formerly Anooraq Resources Corporation) through its South African Subsidiary Plateau Resources Limited, holds exclusive prospecting rights to prospect 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 Prospecting Right MPT 76/2007 PR. The Prospecting Right 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 prospecting right 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 mining right. To date, Ivanhoe has advised that to thebest of its knowledge, a mining right has not been issued over Rietfontein. The JV is valid until the expiry of the prospecting right. 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 Major Township and Farm Locations



The main road indicated on this plan is the N11 highway. The UMT and Bikkuri areas are considered to be amenable to underground mining methods; the AMK and ATS areas are considered to be amenable to open pit mining methods. The boundary of ATS is constrained by the Turfspruit 241 KR farm (mineral tenure) boundary, and the north-eastern boundary of UMT. Figure generated by Amec Foster Wheeler 2015, information from Ivanhoe.

1.2.3 Surface Rights

Surface rights over the Macalacaskop 243 KR, Turfspruit 241 KR and Rietfontein 2KS 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. Ivanhoe has advised that it undertook extensive consultation with the communities who are the lawful occupiers of the Macalacaskop 243 KR and Turfspruit 241 KR farms, and surface use agreements and co-operation agreements, regulating, among other things, the compensation for losses and damages, were entered into with four local communities during 2010. These agreements were extended in Q3'2014 and will remain in force until the conclusion of the long-term surface lease agreements.

Long-term surface lease agreements will have to be concluded when the mining phase commences in order to cater for the required surface mining and plant infrastructure. Ivanhoe is currently in the process of consulting with the affected communities and the government in order to further negotiations for a long-term surface lease.

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 2016 Resource Technical Report.

Ivanhoe acquired a prospecting licence 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 testwork, and conceptual mining studies that considered open pit scenarios.

In 2007, Ivanhoe commenced a deep drilling programme to investigate the continuity and grade in an area targeted as having underground mining potential. This resulted in multiple mineral resource estimates assuming underground mining methods between September 2010 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 testwork, geotechnical and hydrological investigations, seismic survey, social and environmental impact assessments, mineralogical studies, mineral resource estimation, 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. 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. The complete copy of the 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.

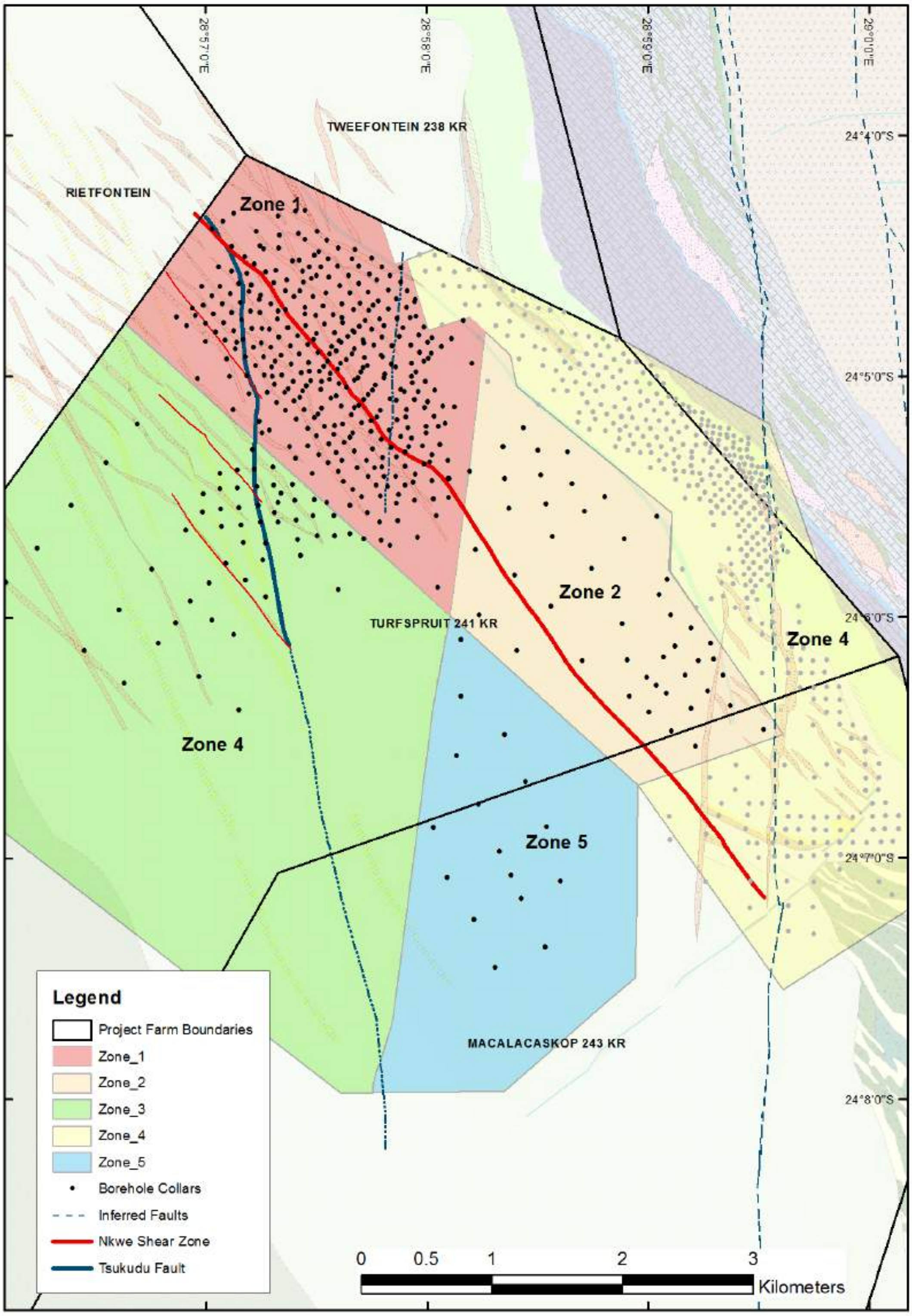
Within the TCU, high-grade PGE–Ni–Cu mineralisation is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, bound by chromitite stringers and containing coarse-grained sulphides; this is known as T2, with the mineralised portion referred to as T2MZ. The 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 recognized 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 drill holes (PUM001 and PUT001) drilled in 2012 which are located in Zone 4. These drill holes are grouped with the open-pit drill holes.

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) totaling 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 defection and do not include the mother or pilot hole portion. This includes 33 drillholes and deflections (9,181 m) completed for geotechnical purposes and 62 drillholes and deflections (23,001 m) completed for metallurgical sampling purposes.

Standardised geological 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 1409 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 memo from site (Ivanplats, 2015) discussing a review of the downhole surveys states that EMS downhole surveys were selected over gyro survey results for 70 drill holes.

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 have been 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, bench-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 have been added to the database during 2014-2015 and permit the grade shells to be constructed using 3PE+Au cutoffs. 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 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. A portion of the Indicated Mineral Resources has been used to support Mineral Reserve estimation in the Platreef 2014 PFS. 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	4.7	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	1431	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 drill holes 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 drill holes 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 drill holes 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 drill holes 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 Geotechnical Investigations

Geotechnical investigations were conducted using 64 representative boreholes collated from programmes conducted prior to and during the PFS. A total of 413 laboratory tests, including uniaxial and triaxial compressive strength and Brazilian tensile strength tests, were used to determine the rock mass properties in the Platreef area. The analysis and interpretation of the geotechnical borehole data and laboratory test results was used to classify the rock mass and to determine rock properties. The interpretation of the structural data was incorporated in a structural domain model. Conceptual numerical analyses were conducted to investigate the induced stress in the stope walls and then stope dimensions were determined empirically. Backfill and stope support requirements have been determined. Guidelines for access development and support have been provided. A geotechnical risk investigation was conducted. A summary of the findings of the geotechnical risk investigation is presented below:

- There is very little variability in the quality of the rock mass, which is fair to good. There are small weak zones within the hanging wall, orebody and footwall, which will have a structural effect and requires further investigation during the next phase of study.
- The mean hanging wall, orebody and footwall strengths are 191 Mpa, 166 Mpa, and 185 Mpa, which is relatively strong rock.
- The stope hydraulic radii for design of the walls and backs (supported) are 8 m and 6 m and the stope backs must be supported with split sets, mesh and 6 m long cable anchors.
- Five structural domains were determined and stopes have been optimally orientated within each domain. Stope stability could be affected by adverse structures, but this can be managed through visual stope monitoring during mining and early placement of backfill if conditions deteriorate excessively. These disruptions could influence productivity and will require further investigation in the next phase of study.
- The geological structural model will need to be improved during the next phase of study to optimise mining layouts and minimise geotechnical risk. This is particularly relevant to the access development.
- The stress state is currently unknown and a sensitivity analysis was therefore conducted using a range of horizontal to vertical stress (K) ratios. The K ratio is expected to be around 1, but could be up to 2 and may result in significantly higher stresses in the stope backs and tunnels, but the proposed support caters for this hazard. Stope walls do not experience as large an increase in stress, but since these are not supported, they are more vulnerable to increased stress. Damage to stope walls could affect stope productivity if they are unfavourably oriented with respect to the stress.
- Rockbursts are not expected to be a major concern at this depth unless the horizontal stress is very high, but this will require further investigation during the feasibility study.
- Stress measurements should be undertaken during the next phase of study to determine the K ratio and principal stress orientations, to verify the assumptions and improve the stress damage and rockburst risk investigations.

1.12 Metallurgical Testwork Overview

There have been a number of metallurgical testwork campaigns and conceptual flow sheet designs carried out for the treatment of Platreef samples since 2001. Metallurgical testwork focused on maximising recovery of PGEs and base metals, mainly nickel, while producing an acceptably high-grade concentrate suitable for further processing and/or sale to a third party.

In 2008, with the advent of the deep drilling exploratory programme, testwork was performed on high-grade composite samples. The high-grade testwork results were promising and indicated that there was a strong possibility of increasing concentrate grade and recovery.

A flotation testwork programme on drill core samples was completed at the SGS laboratories in Johannesburg. The results have indicated that a potentially saleable concentrate can be produced. Following the SGS work, a test programme was undertaken at Xstrata Process Support Canada (XPS) laboratories. The XPS work did not materially add to the results from SGS Johannesburg.

In 2012, the resource was geologically re-assessed, and samples of three new geo-metallurgical units were supplied to Mintek. These units were designated T1, T2U, and T2L.

Comminution tests have indicated that the plant feed is classified as hard to very hard and thus not suitable for Semi-Autogenous Grinding (SAG) milling. A crusher and ball mill circuit will be the preferred option, with lowest associated technical risk. The flotation testwork has shown that the plant feed is amenable to treatment by conventional flotation without the need for re-grinding. Flotation losses from the circuit are due to a non-floating PGE population locked in gangue at sizes of 10 µm or finer and amounting to approximately 10%–15% of the contained PGEs.

Comminution and flotation work indicated that the optimum mill grind would be 80% passing 75 µm. Batch open circuit flotation work was performed as well as locked cycle flotation testwork. The inclusion of post mill conditioning of solids prior to flotation resulted in improved PGE recovery and upgrade in the cleaner circuit. 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 improved PGE, Cu and Ni recovery. All of the three geo-metallurgical units and the composites produced acceptable smelter-grade final concentrates of approximately 85 g/t PGE (Pt, Pd, Rh + Au) at acceptable recoveries.

Stainless steel grinding media and high chrome grinding media proved to be advantageous over that of carbon steel media in terms of metal recovery and concentrate grade.

The processing plant consists of a relatively standard flotation concentrator targeted at producing a saleable concentrate, based on specifications provided by others.

The design approach currently entails concentrators able to accommodate Phases 1–3 beginning at a concentrator feed rate of 4 Mtpa followed by expansions to 8 Mtpa (Phase 2) and 12 Mtpa (Phase 3).

1.13 Platreef Development Scenarios

Ivanhoe has identified significant Mineral Resources on the Platreef Project. The Platreef 2014 PFS was applied to part of the selectively-mineable Mineral Resources within and adjacent to Turfspruit Cyclic Unit (TCU) mineralised zones. The Platreef 2014 PFS analysed Phase 1, the 4 Mtpa Concentrator Case.

There remain substantial additional Mineral Resources on the Platreef Project that will be the subject of further study.

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. 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 1.4. 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 by 2020. Phase 2 includes an additional ramp-up to a plant capacity of 8 Mtpa by 2024. Phase 3 envisages a further ramp-up to a plant capacity of 12 Mtpa by 2028. All production is sourced from underground mining.

Phase 1 has recently commenced with the sinking of an exploration shaft (now called Shaft No. 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. A Bulk Sample Application was lodged with the Department of Mineral Resources (DMR) in Polokwane in September 2012, and permission was granted in late 2013.

In the current design, Shaft No. 2 has a 6 Mtpa capacity. This, combined with the conversion of Shaft No. 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 1.4 Platreef Project Development Scenarios

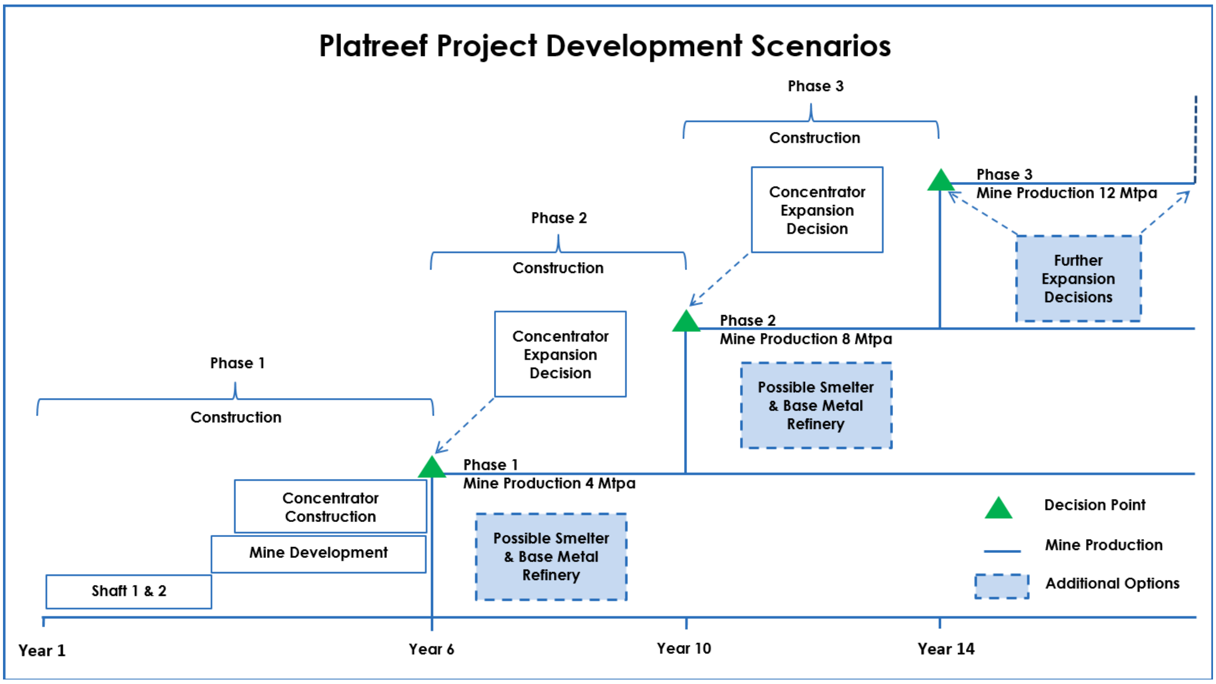


Figure by OreWin, 2014.

1.14 Mining

Stantec assisted with preparation of the underground mine plan and ore reserve estimate. The mine planning responsibilities included mine layout design, mining method selection, production and development planning and scheduling, capital and operating cost estimates, equipment selection, and related infrastructure design.

Mining zones in the current Platreef mine plan occur at depths ranging from approximately 700 m to 1,600 m. Access to the mine will be via four vertical shafts. Shaft No. 2 is the main men and material handling system, while Shaft Nos. 1, 3 and 4 are designated ventilation shafts. Shaft No. 1 is under development and will be used for initial access and development.

Mining will be performed using highly productive mechanised methods and paste backfill will be used to fill open stopes. The ore will be trammed from the active mining areas to the bottom of Shaft No. 2; after being crushed, it will be hoisted to the surface. Figure 1.5 shows an overview of the underground mine workings (looking north-east). Layouts, schedules, and cost estimates were completed for mining Platreef at a rate of 4 Mt of ore per year over a 30-year mine life.

Figure 1.5 Elevated View of Mining Areas by Method

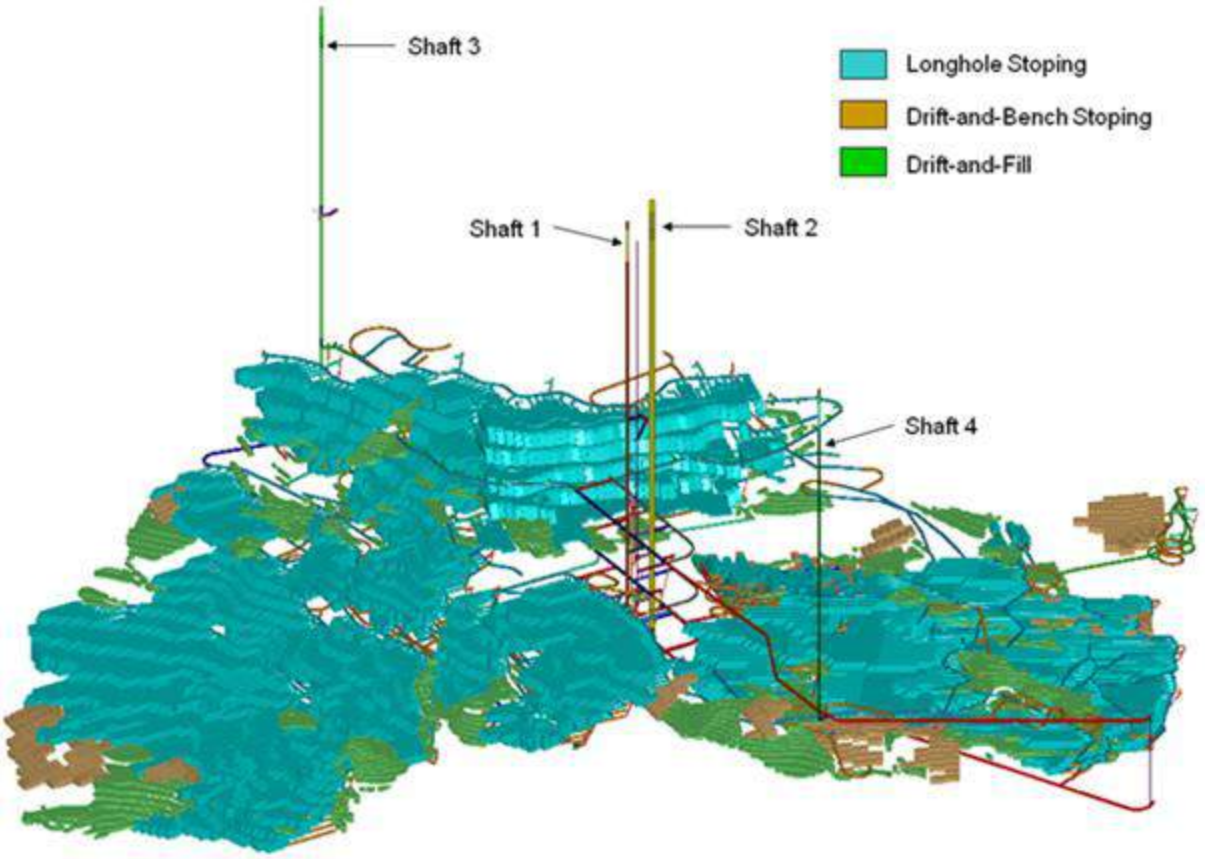


Figure by Stantec 2014.

1.15 Production Summary

Mine production is shown in Figure 1.6 and the key average annual production results over the 31 year mine life are shown in Table 1.2.

Figure 1.6 Mining Production

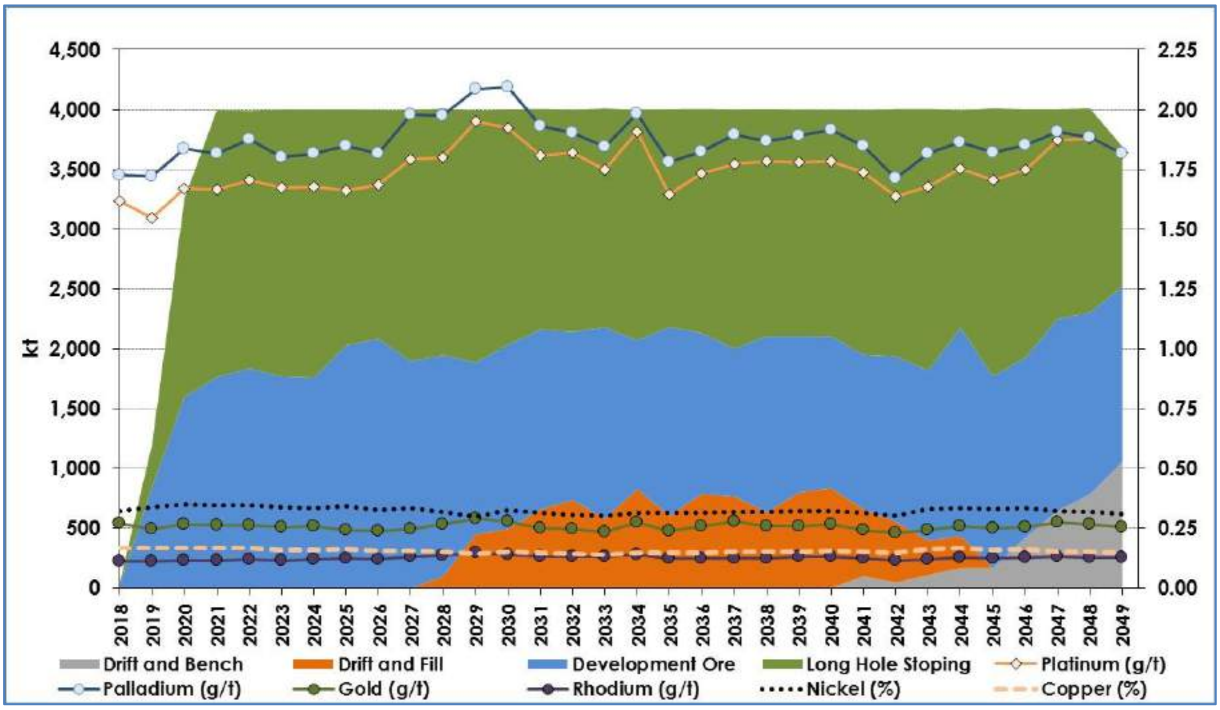


Figure by OreWin 2014.

Table 1.2 Production Summary

Item	Units	Total
Mined and Processed	Mtpa	4
Platinum	g/t	1.76
Palladium	g/t	1.87
Gold	g/t	0.26
Rhodium	g/t	0.13
3PE+Au	g/t	4.02
Copper	%	0.15
Nickel	%	0.32
Recoveries		
Platinum	%	87.2
Palladium	%	86.9
Gold	%	76.7
Rhodium	%	92.0
Copper	%	87.7
Nickel	%	68.8
Concentrate Produced		
Concentrate	ktpa	159
Platinum	g/t	37.5
Palladium	g/t	39.8
Gold	g/t	4.8
Rhodium	g/t	2.8
3PE + Au	g/t	85
Copper	%	3.3
Nickel	%	5.4
Recovered Metal		
Platinum	kozpa	191
Palladium	kozpa	203
Gold	kozpa	25
Rhodium	kozpa	14
3PE + Au	kozpa	433
Copper	Mlbpa	12
Nickel	Mlbpa	19

1. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.
2. Production over 31 years

1.16 Mineral Reserves

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in the Platreef 2014 PFS. 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. This same Mineral Resource block model was used for the Platreef 2014 PEA.

Mineral Reserves were calculated from the block model using the combination of stope optimizer and generated grade based on the economic NSR cut-off values. Three stoping methods (longhole, drift-and-bench, 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–$58.53 per tonne, excluding capital recovery and profit margin. For the production schedule and mineral reserve, a declining cut-off was chosen. A $100 NSR cut-off value was used in defining these reserves in order to increase the initial mill head grade and to shorten the payback period. An $80 NSR cut-off value was used later in the mine life, as the higher grade reserves deplete and mining progresses further from the production shaft. 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.

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–$58.53 per tonne) and production schedule NSR cut-offs ($100 and $80) will provide protection from future negative impacts of these factors.

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

Table 1.3 Platreef Probable Mineral Reserve – Tonnage and Grades as at 8 January 2015

Method	Mt	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Longhole Stopes	106	133.5	1.73	1.86	0.25	0.12	3.97	0.16	0.32
Drift-and-Fill	10	144.3	1.99	1.95	0.29	0.13	4.36	0.14	0.30
Drift-and-Bench	5	146.4	1.95	2.01	0.28	0.14	4.38	0.15	0.32
Total	120	**134.9**	**1.76**	**1.87**	**0.26**	**0.13**	**4.01**	**0.15**	**0.32**

1. Metal prices used in the reserve estimate are as follows: Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $1,250/oz, Cu: 2.73/lb, and Ni: $8.81/lb.
2. Tonnage and grade estimates include dilution and recovery allowances.
3. A declining NSR cut-off of $100–$80 was used in the mineral reserve estimates.
4. Total may not add due to the rounding.
5. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)

Table 1.4 Platreef Probable Mineral Reserve – Contained Metal as at 8 January 2015

Method	Mt	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Longhole Stopes	106	5.88	6.33	0.86	0.42	13.49	362	758
Drift-and-Fill	10	0.63	0.62	0.09	0.04	1.39	30	65
Drift-and-Bench	5	0.28	0.29	0.04	0.02	0.64	15	32
Total	**120**	**6.80**	**7.24**	**0.99**	**0.49**	**15.51**	**408**	**855**

1. Metal prices used in the reserve estimate are as follows: Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $1,250/oz, Cu: 2.73/lb, and Ni: $8.81/lb.
2. Tonnage and grade estimates include dilution and recovery allowances.
3. A declining NSR cut-off of $100–$80 was used in the mineral reserve estimates.
4. Total may not add due to the rounding.
5. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)

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.

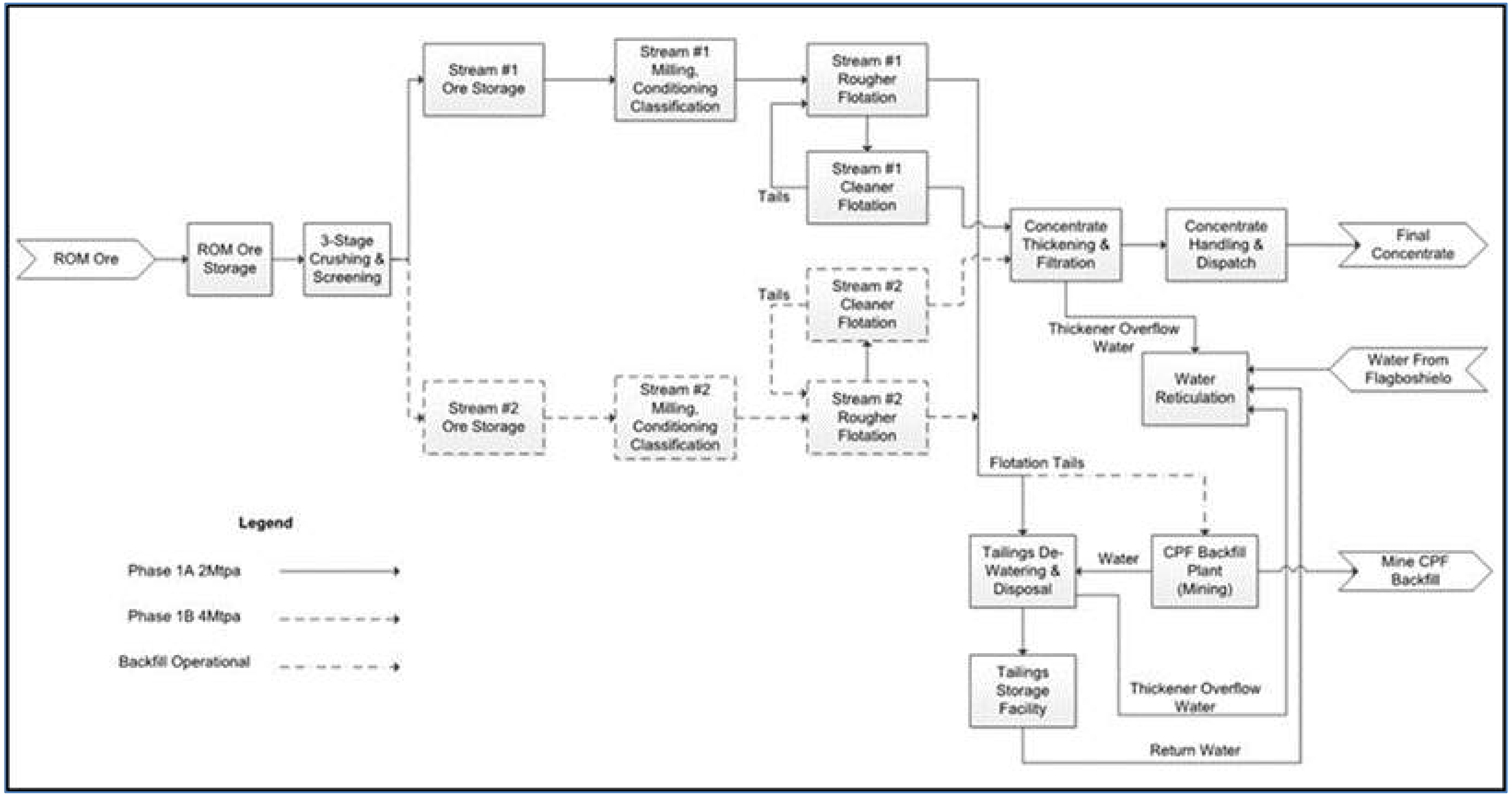
1.17 Recovery Methods

Based on the latest flotation testwork results a concentrator flow sheet was developed for the treatment of T1, T2U, and T2L zones.

Phase 1 includes the production from a 4 Mtpa concentrator and other associated infrastructure in 2019, in two modules of 2 Mtpa.

A two-phased production approach was used for the Phase 1 flow sheet development and design. The selected flowsheet comprises a common three-stage crushing circuit, feeding crushed material to two milling-flotation modules, each with a capacity of 2 Mtpa. Milling is achieved in a ball mill with classification and rougher flotation in a split high, medium and low-grade circuit. Each concentrate is cleaned in a dedicated cleaner circuit with varying stages and recycles. Flotation is followed by common tailings handling and concentrate thickening filtration and storage. The process description is presented in Figure 1.7.

Figure 1.7 Concentrator Flowsheet



Capacity of Phase 1A and 1B are not cumulative. Figure by DRA.

1.18 Infrastructure

1.18.1 Bulk Water

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

The Olifants River Water Resource Development Project (ORWRDP) is designed to deliver water to the Eastern Limb and Northern Limb of the Bushveld Igneous Complex (BIC) of South Africa. The ORWRDP consists of the new De Hoop Dam, the raising of the wall of the Flag Boshielo Dam, and related pipeline infrastructure, which will ultimately deliver water to Pruissen, located to the south-east of Mokopane, and to the Platreef Project. From this point, the Pruissen Pipeline Project will be developed to deliver water to the communities and mining projects on the Northern Limb.

Ivanplats is a member of the Joint Water Forum (JWF) (part of the ORWRDP) and the Pruissen Water Forum. These forums have been established to facilitate and coordinate discussions with the various participants in the ORWRDP scheme within the Eastern Limb and Northern Limb of the BIC. Other major participants in these forums are Anglo Platinum, Lonmin and the Mogalakwena Municipality. Ivanplats is committed to working with the JWF to develop the ORWRDP as the primary source of bulk water to service the needs of the Platreef Project.

During the prefeasibility study, the bulk water supply and demand volumes were determined. The water balance model simulations showed that the average bulk water supply over the life of the mine is 5.5 ML/d, but the maximum daily bulk water supply volume that is required during dry years, assuming limited groundwater ingress to the workings, is 10.3 ML/d.

Ivanplats is pursuing alternative bulk water sources for the Platreef Project to fill the shortfall for the interim period during sinking and construction phases until the ORWRDP is operational. To date, treated sewage effluent and local groundwater have been identified as sources of water to meet the 6 ML/d shortfall for the period between years 2015–2019. All the options are pursued until agreement is reached for one or more of the sources and further investigations into other water sources will continue.

1.18.2 Project Power Supply

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 SA has not experienced any load shedding.

A new main transmission substation (MTS), called the Borutho MTS (400 kV/132 kV/22 kV) is sized at 500 MVA (extendable to 1,000 MVA) and will be commissioned during the H1'15. The Borutho substation is located approximately 26 km from the site and will provide the main feed to the Platreef Project as shown in the diagram in Figure 1.8.

Figure 1.8 Proposed Power Transmission

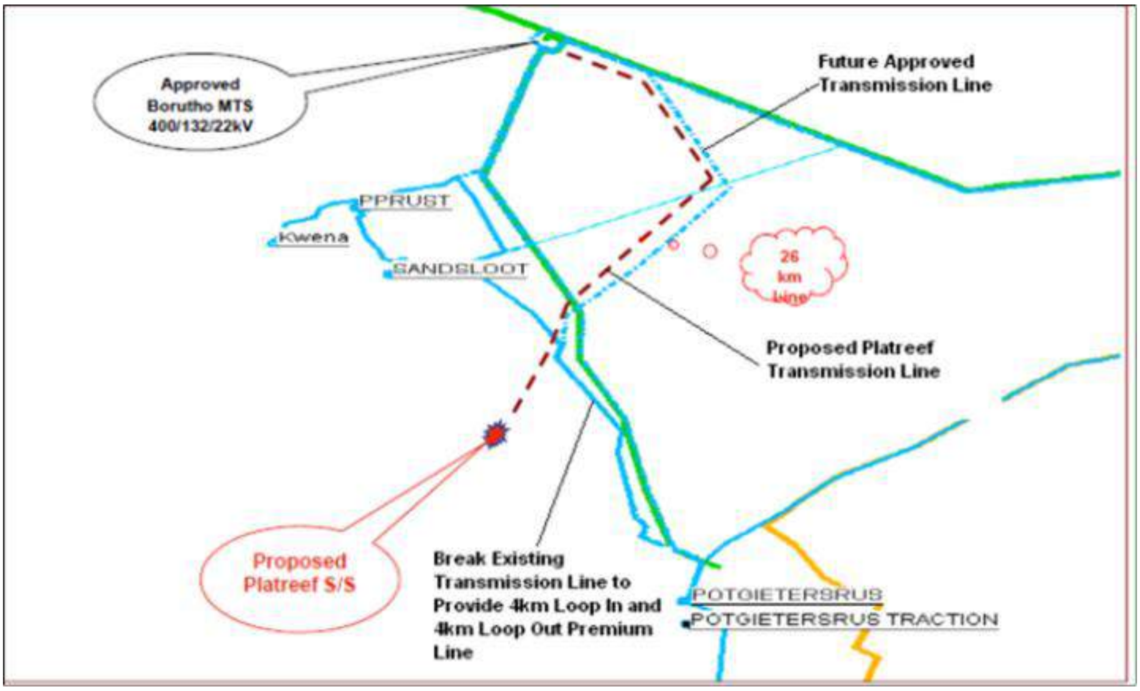


Figure supplied by Ivanhoe 2013.

In 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 for the Platreef Project was completed and Ivanplats then requested that Eskom complete the budget quote study that considers the premium supply option.

The Environmental Impact Assessment (EIA) process is complete and the design fees for the 70 MWA has been paid. The latest forecast energisation date of the Platreef Eskom incoming substation is H2'18.

Eskom has completed the relevant land and rights as well as EIA processes. The Eskom self-build option Budget Quote (BQ) has been accepted and paid by Ivanplats and the detail design package has been completed by Eskom. The latest forecast energisation date of the Platreef Eskom incoming substation is H2'17.

Based on the prefeasibility study design work prepared by Stantec and DRA, the Platreef Project power requirement for a 4 Mtpa underground mine have been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats has notified of and requested from Eskom the 30 MVA additional power demand required 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. The latest forecast energisation date for the 5 MVA construction power supply is Q3'16.

1.18.3 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 commenced the new N11 intersection construction works to the mine site in Q1'16.

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

- Option 1 is to upgrade the existing road through Mokopane, to cater for the increased traffic volumes.
- Option 2 is to build a re-route of the N11 highway, exiting the N1 north of Mokopane and entering the existing N11 approximately 5 km north of the Platreef Project area. The realignment route will bypass the Turfspruit 241 KR and Macalacaskop 243 KR farms, but will bisect the Rietfontein 2 KS farm, and has therefore been considered in the Rietfontein 2 KS Tailings Storage Facility (TSF) footprint.

1.19 Market Studies and Contracts

South Africa has a number of smaller PGE mining companies. Toll smelting and refining contracts and purchase agreements have therefore become more prevalent in South Africa than in the past. The major PGE mining companies have some internal purchase contracts with their own mining/concentrating operations and external and arm's-length purchasing or toll contracts with independent or JV companies. Within the industry and along the value chain there are various possibilities for metal sales contracts: concentrates, furnace and converter mattes, and PGE residues or concentrates have all been sold or toll-treated in the past. The conclusions of the marketing studies have been used as the basis for the realisation and other marketing assumptions in the Platreef 2014 PFS.

Potential purchasers must consider competitive, cost and capacity pressures while weighing up impurity tolerances with changing feed mixes and process efficiency improvements. Final terms will be significantly influenced by contract term and escalation for both treatment and or refining charges and exchange rate movements. The estimates made for the Platreef 2014 PFS have been made based on knowledge of concentrate sales contracts that have been agreed in South Africa with local purchasers for historical and current concentrates. Actual terms may vary and will be dependent on the negotiations at the time the contracts are agreed.

Metallurgical work carried out to date on Platreef concentrate has so far indicated that the historically documented range of 80-84% return would equally apply, taking into account the variables and conditions above. The midpoint of 82% has been applied at this stage to the

Platreef 2014 PFS analysis. The contractual terms for Platreef and comparable products are highly variable and are subject to change as conditions for any one local or offshore purchaser alter.

It appears that current smelter, converting and acid plant capacity as well as nickel refining capacity are the constraining factors that may even limit the mining rate for Platreef. Sufficient furnace capacity is probably currently available but converting and sulphur removal capacity are probably constrained by equipment and environmental issues. It is believed that there is sufficient nickel refining capacity to accommodate Phase 1 of the Platreef project.

It does appear that there is some upside capacity for increased trading in concentrates. For instance, Zimbabwe may be successful in the government's goal of providing refining capacity in Zimbabwe for all Zimbabwean PGE's and base minerals. This will impact upon the available smelting, converting, acid capture and nickel refining in South Africa. Beneficiation options in Zimbabwe are currently being explored by local producers.

The PGE mining industry in South Africa is currently in a state of flux. Labour unrest, closures of unprofitable shafts and the threat of an export ban in Zimbabwe are all factors which could free up smelting and refining capacity in South Africa. Expansion plans, in particular at Anglo Platinum's Mogalakwena mine and the reopening of closed shafts will have the opposite effect. At this stage this is difficult to predict and it has been assumed that there will be sufficient smelting and refining capacity in South Africa to accommodate the first phase of the Platreef project by 2020. It has been concluded that the Platreef project is a clear demonstration of the evolution of the South African PGE mining industry. As a highly mechanised, low cost and high grade operation Platreef is expected to be well placed to supply into the PGE market.

1.20 Environmental Studies, Permitting, Social, and Community Impact

The Platreef Project site lies in a north-westerly direction, approximately 8 km from the town of Mokopane (previously known as Potgietersrus). The Platreef Project is situated in the magisterial district of the Mogalakwena Local Municipality and within the Waterberg District Municipality.

There are several communities within the proposed project area which are affected by the Platreef Project.

Baseline studies have been undertaken within the Platreef Project area, in support of an Environmental and Social Impact Assessment (ESIA), which is part of the MRA that was submitted on 6 June 2013. These ESIAs were conducted to comply with local legislation as well as international requirements and consisted of the following:

- Topography assessment
- Heritage and archaeology
- Aquatic ecology and wetlands
- Fauna and flora
- Dust monitoring (air quality)
- Noise assessment

- Soils and land capability
- Visual assessment
- Socio-economic assessment
- Resettlement action plan framework

The ESIA summarises relevant results of the interim environmental and social baseline of the Platreef Project area. Further baseline studies would be required to be conducted during the completion of ESIA, to ensure compliance with local and international requirements.

Ivanplats has a programme of work in place to comply with the necessary environmental, social and community requirements. Key work should include:

- ESIA in accordance with the MPRD Act, the National Environmental Management Act (NEM Act) as well as the Equator Principles (EP) and International Finance Corporation's (IFC) Performance Standards in Mining;
- 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).

1.21 Capital and Operating Cost Summary

The total pre-production and sustaining capital costs required, including contingency, from the Platreef 2014 PFS are shown in Table 1.5.

Table 1.5 Total Projection Capital Cost

US$M	Pre-Production	Sustaining	Total
Mining			
Underground	542	956	1,498
Surface Infrastructure	64	-	64
Backfill Plant	21	14	34
Capitalised Operating Costs	35	-	35
Subtotal	**661**	**970**	**1,631**
Processing & Tailings			
Concentrator	93	181	274
Rietfontein TSF	30	30	59
Subtotal	**123**	**211**	**334**
Infrastructure			
General Infrastructure	115	63	178
Site Pre-Production	8	1	9
Closure Costs	-	18	18
Subtotal	**123**	**83**	**206**
Indirects			
Exploration & Geology	3	0.4	4
Engineering Procurement Contract Management (EPCM)	59	17	75
Capitalised G&A & Other Costs	15	6	21
Subtotal	**77**	**23**	**100**
Owners Cost	71	4	75
Capex Before Contingency	**1,054**	**1,291**	**2,345**
Contingency	114	110	224
Capex After Contingency	**1,168**	**1,401**	**2,569**

1. Sustaining capital expenditure also includes 2019 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 1.8. Among the current and future Northern Limb producers, Platreef's estimated cash cost of US$322 per 3PE+Au ounce, net of copper and nickel by-product credits, ranks at the bottom of the cash-cost curve.

Table 1.6 Cash Costs After Credits

	US$/oz Payable 3PE+Au		
	Life-of-Mine Average	5-Year Average	10-Year Average
Mine Site Cash Cost	401	454	429
Realisation	390	384	390
Total Cash Costs Before Credits	**792**	**838**	**819**
Nickel Credits	389	438	419
Copper Credits	80	90	86
Total Cash Costs After Credits	**322**	**310**	**314**

Totals vary due to rounding.

Figure 1.9 Cash Cost Comparison Platreef and 2013 Producers

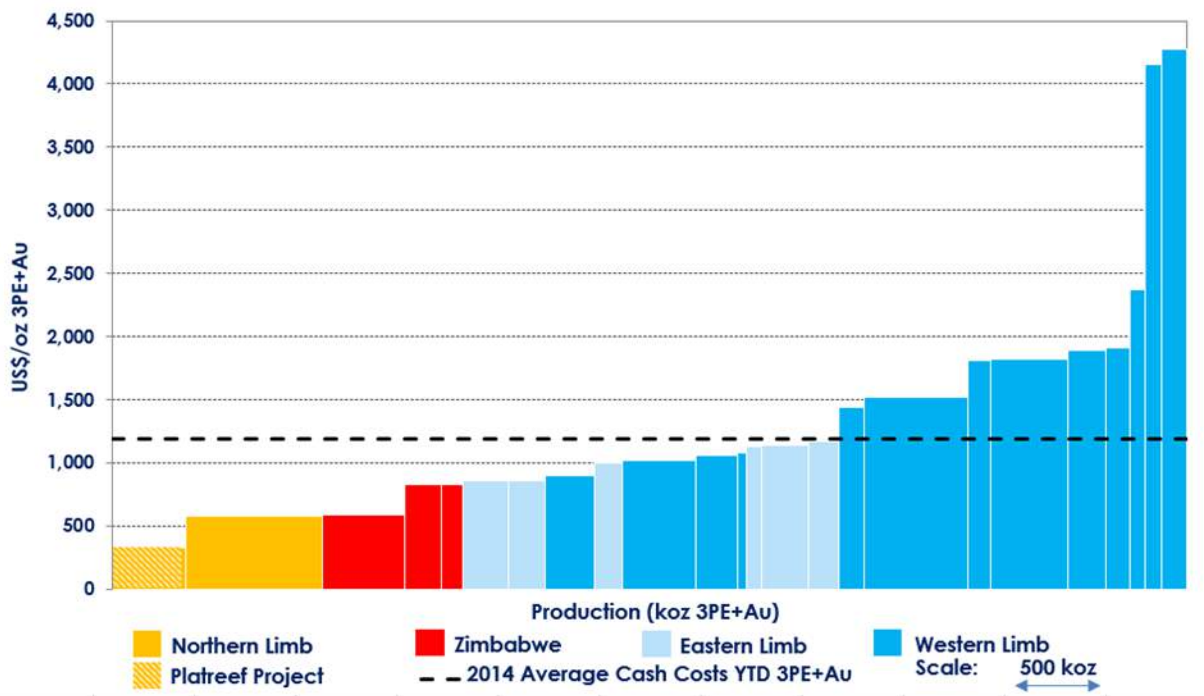


Figure by OreWin Producer cost data. Source: SFA (Oxford).

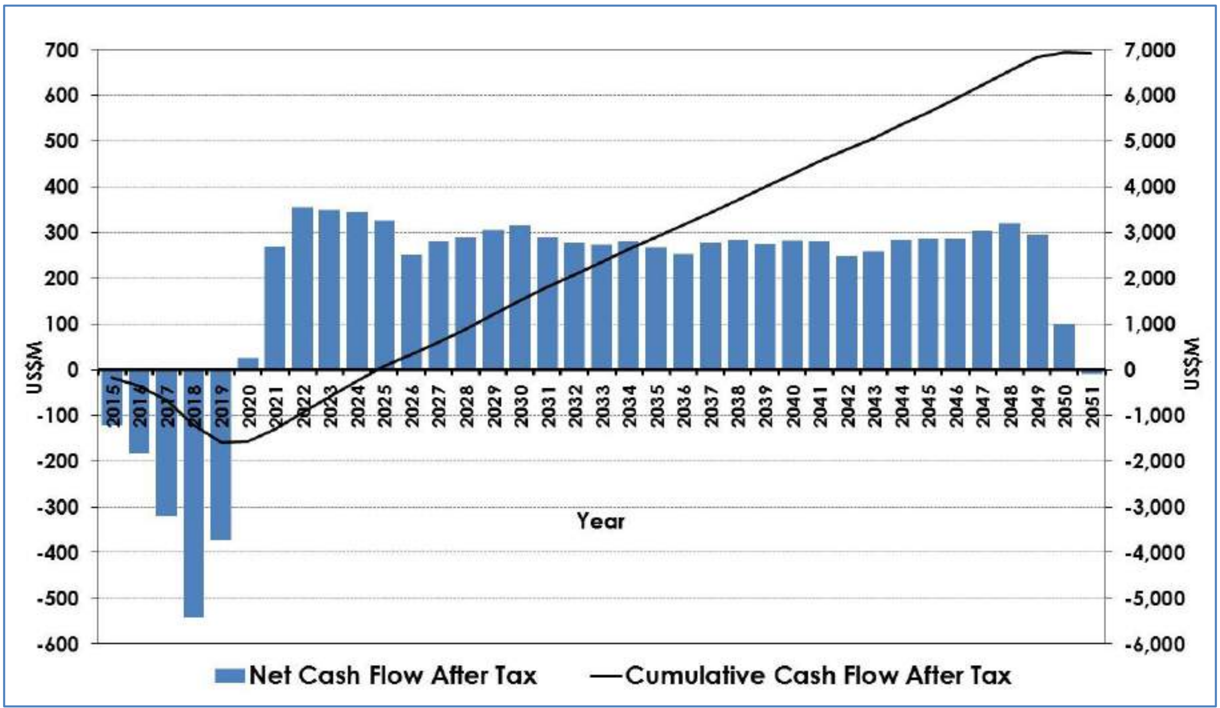
Table 1.7 Operating Costs and Revenues

	Life of Mine TOTAL US$M	US$/t milled		
		5-Year Average	10-Year Average	Life of Mine Average
Gross Sales Revenue	22,981	188.18	189.54	191.19
Less: Realisation Costs				
Transport Costs	195	1.55	1.59	1.63
Treatment & Refining Charges	4,137	33.87	34.12	34.41
Royalties	908	5.61	6.82	7.55
Total Realisation Costs	**5,240**	**41.03**	**42.52**	**43.59**
Net Sales Revenue	**17,742**	**147.15**	**147.02**	**147.60**
Site Operating Costs				
Mining	3,585	32.71	31.44	29.83
Processing & Tailings	1,372	11.88	11.58	11.42
General & Administration	434	3.86	3.70	3.61
Total	**5,392**	**48.45**	**46.72**	**44.86**
Operating Margin	**12,349**	**98.70**	**100.30**	**102.74**
Operating Margin (%)	**54%**	**52%**	**53%**	**54%**

Totals vary due to rounding.

The Net Cash Flow after Tax and the Cumulative Cash Flow after Tax is shown in Figure 1.10.

Figure 1.10 Cumulative Cash Flow After Tax



1.22 Summary of Financial Results

The economic analysis uses price assumptions of US$1,630/oz Pt, US$815/oz Pd, US$1,300/oz Au, US$2,000/oz Rh, US$8.90/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 11 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$/oz	1,630
Palladium	US$/oz	815
Gold	US$/oz	1,300
Rhodium	US$/oz	2,000
Copper	US$/lb	3.00
Nickel	US$/lb	8.90
Base Metals Refining Charge	% Gross Sales	18%
Precious Metals Refining Charge	% Gross Sales	18%

The results of the financial analysis show an After Tax NPV8 of US$972M. The case exhibits an after tax IRR of around 13% and a payback period of around seven years. The estimates of cash flows have been prepared on a real basis as at 1 January 2015 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	9,619	6,981
	5.0%	3,024	2,113
	8.0%	**1,491**	**972**
	10.0%	885	519
	12.0%	473	210
	15.0%	80	-86
	20.0%	-254	-336
Internal Rate of Return		15%	13%
Project Payback Period (Years)		7	7

1.23 Summary of Platreef 2014 PFS

The key features of the Platreef 2014 PFS included:

- Development of a large, mechanised, underground mine is planned at an initial 4 Mtpa throughput scenario.
- Planned average annual production rate of 433 koz of platinum, palladium, rhodium, and gold (3PE+Au).
- Estimated pre-production capital requirement of approximately US$1.2 billion, including US$114 million in contingencies.
- After-tax Net Present Value (NPV) of US$972 million, at an 8% discount rate.
- After-tax Internal Rate of Return (IRR) of 13%.
- The Platreef 2014 PFS 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 Platreef 2014 PFS Results

Item	Units	Total
Mined and Processed		
Mineral Reserve	Mt	120
Platinum	g/t	1.76
Palladium	g/t	1.87
Gold	g/t	0.26
Rhodium	g/t	0.13
3PE+Au	g/t	4.02
Copper	%	0.15
Nickel	%	0.32
Concentrate Produced		
Concentrate	kt	4,915
Platinum	g/t	37.5
Palladium	g/t	39.8
Gold	g/t	4.8
Rhodium	g/t	2.8
3PE+Au	g/t	85
Copper	%	3.3
Nickel	%	5.4
Recovered Metal		
Platinum	koz	5,927
Palladium	koz	6,295
Gold	koz	761
Rhodium	koz	448
3PE+Au	koz	13,431
Copper	Mlb	358
Nickel	Mlb	588
Key financial results		
Life of Mine	years	31
Pre-Production Capital	US$M	1,168
Mine Site Cash Cost	US$/oz 3PE+Au	401
Total Cash Costs After Credits	US$/oz 3PE+Au	322
Site Operating Costs	US$/t Milled	44.86
After Tax NPV8	US$M	972
After Tax IRR	%	13
Project Payback Period	years	7

1. The economic analysis is based on Mineral Reserves only.
2. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)
3. Metal prices used in the reserve estimate are as follows.US$1,699/oz Pt, US$667/oz Pd, US$1,315/oz Au, US$1,250/oz Rh, US$8.81/lb Ni, US$2.73/lb Cu
4. A declining Net Smelter Return (NSR) cut-off of US$100/t–$80/t was used in the mineral reserve estimates.
5. Metal price assumptions used for the base case economic analysis are: US$1,630/oz Pt, US$815/oz Pd, US$1,300/oz Au, US$2,000/oz Rh, US$8.90/lb Ni, US$3.00/lb Cu.
6. Tonnage and grade estimates include dilution and recovery allowances.

Totals may not add due to the rounding.

1.24 Conclusions

1.24.1 Platreef 2016 Resource Technical Report

The Platreef 2016 Resource Technical Report presents the Mineral Reserve for Phase 1 of the Platreef development, as determined in the Platreef 2014 PFS. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. The subsequent completion of these studies is necessary to bring Phase 1 to a feasibility study level. Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of Phases 2 and 3 will be evaluated at a later date and the decision to expand can be deferred or brought forward as markets dictate and funding permits.

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 mineralization has been modelled based on limited drill data, as not all of the UMT drill holes extended into the FW. For this reason, estimation of Mineral Resources has been restricted to the northwestern area of the Platreef Project where drill spacing is in the order of 100 metres to 200 metres. Similar mineralization has been seen in drill holes 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 cutoff. Discontinuous pods of mineralization at a 2.0 g/t 3PE+Au cutoff 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 mineralization that may permit mass mining methods at a lower cutoff (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 2014 PFS 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 mechanized underground mine at a prefeasibility-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

For the purposes of Platreef 2014 PFS the testwork was considered to be adequate. Detailed mineralogical analysis of the ores, the concentrates and the tailing samples have contributed to the understanding of the mode of occurrence and liberation characteristics of the valuable minerals.

The testwork programmes have been conducted by parties well versed in the processing of ores from the Bushveld Igneous Complex. The necessary checks and balances have been applied to ensure that the testwork and chemical analysis has been conducted with the necessary diligence and accuracy.

The selection of samples, done in conjunction with the mining and geological teams, submitted for the metallurgical testwork for the purposes of the Platreef 2014 PFS are deemed to be sufficient. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. 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 recommended for this stage of the study.

The proposed flotation circuit is based on interpretation of the results obtained from the flotation testwork. The sizing of the various flotation stages is adequate for this level of pre-feasibility study and will be refined during the future variability testwork campaigns.

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 various 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 2Mtpa.

The availability of skilled labour resources for both construction and operational phases of the Platreef Project is limited and the training program will have to be closely monitored to ensure that the correct skills development is done at the correct time, depending 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 2016 Resource Technical Report

The Platreef 2016 Resource Technical Report provides an update of the Mineral Resource with the Mineral Reserve from the Platreef 2014 PFS remaining the same. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

The results of the Platreef 2014 Resource Technical Report support the recommendations made in the Platreef 2014 PFS, to progress studies to a feasibility study (FS) level. It recommends that Ivanhoe continue to optimise the FS scope of work and execution plan. The FS should be based on Phase 1 and evaluation of Phase 2 and Phase 3 scenarios should continue at scoping study level. The options for a smelter and BMR should be further evaluated and incorporated into the overall project studies. The costs of these studies are included in the cost analysis of the Platreef 2016 Resource Technical Report.

Ivanhoe has retained Whittle Consulting (Whittle) of Melbourne, Australia to conduct an optimisation study using as a base the Platreef 2014 PFS production schedules, revenues and costs. The work has been completed and will be used in optimising the FS mine plan . The recommendations identified in the study are as follows:

- Net value per bottleneck unit to identify the location of the most profitable ore;
- Bring forward revenue and cash flow using enhanced scheduling techniques;

- Examination of initial smaller scale operation to bring forward the start date of operations and generate earlier cash flow;
- Analysis of dynamic grind to increase processing rates and dynamic mass pull to flotation concentrate to increase recovery to increase early cash flow;
- Increased cut-off grade in the initial years to increase revenue and cash flow;
- Alternative use of the planned mining infrastructure.

1.25.2 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 mineralization 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 recognized a definitative answer may have to await exposures in underground workings.

1.25.3 Mining Recommendations

As the results and conclusions from the Platreef 2014 PFS were positive, Stantec has recommended the advancement of the Platreef study to FS level. Stantec recommended the following additional work and modifications to the current mine plans during the feasibility study:

- Mine layouts, ore and waste pass system, and designs should be refined and optimised to the extent possible in the feasibility study to enable more accurate scheduling and cost estimates.
- Stoping layouts should be prepared in greater detail with top cuts and bottom cuts as part of the optimization process.
- Shaft sinking and other development rates should be reviewed and modified as necessary in order to ensure that the preproduction development targets (milestones) are reasonable in the feasibility study.
- Alternative types and sources for backfill should be evaluated for the time period between production start-up and commissioning of the paste backfill plant.

This work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

1.25.4 Metallurgical

The metallurgical testwork conducted to date has been based on industry-accepted procedures, and is considered to be adequate to meet the mineral processing requirements of the pre-feasibility study. It has, however, confirmed a number of potential risks and opportunities that need to be explored and addressed in the future phases of project development. These include:

- The mineralised material is considered to be very hard and competent. The feasibility study phase should aim to delineate material hardness variability to domain and spatial location;
- The oxalic acid and thiourea reagent suite will require extended conditioning time, during full scale operation, in order to achieve laboratory recovery and concentrate specifications. Alternative reagent suites should be further explored during the feasibility study phase to simplify the process and reduce capital and operating costs;
- Concentrate specifications and optimisation are required in subsequent study phases to optimise the concentrate specifications in line with off-take options. Testwork should aim to maximise 3PE+Au grade and metal recovery, whilst minimising the sulphide gangue component reporting to the concentrate;
- The future study phase should aim to delineate variability of recovery and operating costs to grade, domain and special location in the deposit.

Further testwork is recommended on domain and variability samples to address these risks and explore the opportunities in the early stages of the feasibility studies. Variability testwork in relation to the mining plan should be considered during the feasibility study phase.

The presence of floatable gangue species and the effect of these minerals on the grade-recovery profile is sufficient motivation for the commissioning of a mini or pilot plant campaign to understand the interaction and potential build-up of floatable contaminants in the flotation circuit.

Once a better understanding of the processes is achieved a more accurate estimate of the process capital and operating costs can be developed in the feasibility study phase.

The metallurgical testwork and the proposed concentrator design are at a suitable stage to justify that the project should progress to a feasibility stage from a metallurgical processing viewpoint.

Alternative comminution circuits need to be considered once further comminution results become available in the feasibility phase. The application of alternative comminution technology can also be considered in light of the successes achieved on other operations in the vicinity of Platreef.

1.25.5 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 to notify has notified of and requested from Eskom the 30 MVA additional power demand required for the Platreef Project. Alternative power sources need to be further investigated during the FS.

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); ITC Platinum Development Ltd. (ITC) an affiliate of 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 2016 Resource Technical Report, 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 2016 Resource Technical Report 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 2016 Resource Technical Report provides an update of the Platreef Project Mineral Resource, with the Mineral Reserve from the Platreef 2014 Prefeasibility Study (Platreef 2014 PFS) remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. The Platreef 2016 Resource Technical Report should be read in this context.

The Platreef 2016 Resource Technical Report has an effective date of 22 April 2016 and has been prepared using the Canadian National Instrument (NI) 43-101 Standards of Disclosure for Mineral Projects.

The following companies have undertaken work in preparation of the Platreef 2016 Resource Technical Report (and/or the Platreef 2014 PFS):

- 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 Inc. (Stantec): Underground mine plan.
- Geo Tail Pty Ltd (Geo Tail): Tailing Storage Facility (TSF).
- DRA Projects SA (Pty) Ltd (DRA): Process engineering and infrastructure.

The Platreef 2016 Resource Technical Report uses metric units of measure. The currency used is 2014 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 1.1–1.2, 1.3.2, 1.13, 1.19–1.23, 1.24.1, 1.25.1; Sections 2–4; Section 6; Section 19; Section 20; Sections 21.1–21.3 and 21.14; Section 22; Section 23; Section 24; Section 25.1; Sections 26.1.1, 26.7; and Section 27.
- Dr Harry Parker, SME Registered Member (2460450), employed by Amec Foster Wheeler E&C Services Inc as a Consulting Geologist and Geostatistician, was responsible for: Sections 1.3.1, 1.4 to 1.10, 1.24.2, and 1.25.2; Sections 2.2 to 2.5; 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 26.1.2 to 26.1.3; and Section 27.
- Timothy Kuhl, SME Registered Member (1802300), employed by Amec Foster Wheeler E&C Services Inc as a Principal Geologist, was responsible for: Sections 1.3.1, 1.4 to 1.10, 1.24.2, and 1.25.2; Sections 2.2 to 2.5; 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 26.1.2 to 26.1.3; and Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting Inc. as Principal Consultant, was responsible for: Section 1.11; Section 2; Section 3; and Section 16.1.
- Mel Lawson, B. Eng. (Mining), SME Registered Member (1859650), employed by Stantec Consulting International LLC as Mining Principal, was responsible for: Sections 1.14-1.16; 1.24.3, 1.25.3; Section 2; Section 3; Section 15; Section 16.2; Sections 21.4, 21.8, 21.9; Sections 25.3, 25.4; Sections 26.3, 26.4; Section 27.
- Val Coetzee, B.Eng (Chemical), M.Eng (Mineral Economics), Process Manager, DRA Projects (Pty) Ltd, was responsible for: Sections 1.12, 1.17, 1.24.4, 1.25.4, Section 2, Section 3, Sections 10.7, 10.10.2; Section 13, Section 17, Sections 21.10 to 21.13, Section 25.5 Section 26.4 and Section 27.

- Graham Smith, B.Sc Civil Engineering, Managing Director – Infrastructure Division, DRA Projects (Pty) Ltd was responsible for: Sections 1.18, 1.24.5, 1.25.5, Section 2, Section 3, Section 5, Section 18, Sections 21.5, to 21.7, Section 25.6; Section 26.5; 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 and on 9 October 2014. 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 Mel Lawson participated in the Platreef Project kick-off 7–9 November 2012 in South Africa, which included meetings in Ivanhoe's Sandton offices with Ivanhoe staff and project consultants. A site visit on 8 November 2012 included briefings from Ivanhoe geology and exploration personnel, discussion of the status of the ongoing geotechnical programme, examination of representative drill core, and inspection of potential areas for infrastructure and shaft siting. On 04–05 April 2014, Mr. Lawson attended a PFS 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 testwork is underway.
- Mr. Graham Smith has not visited the Project site, however he was Lead Civil Engineer on the nearby Mogalakwena Platinum 600 ktpm Concentrator Project for 2 years and visited the area often during the design and construction phase of that project.

2.4 Effective Dates

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

- Date of the Mineral Resource estimate that is amenable to open pit mining methods: 31 March 2011.
- Date of the Mineral Resource estimate that is amenable to underground mass mining methods: 13 March 2013.
- Date of the Mineral Resource estimate that is amenable to selective underground mining methods: 22 April 2016.
- Date of the Bikkuri Mineral Resource estimates that are amenable to selective underground mining methods: 22 April 2016.
- Date of the supply of the last drillhole information used in the UMT models: 24 July 2015.
- The base date for the capital and operation cost estimates; 31 January 2014.
- Date of the Mineral Resource estimation update; 22 April 2016.

2.5 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of the Platreef 2016 Resource Technical Report 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 2016 Resource Technical Report 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 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 2016 Resource Technical Report and in support of the Mineral Resource estimate in Section 14.

3.2 Royalties and Taxes

The assumptions for royalties and taxes have been provided by Ivanhoe and are based on the letter dated 13 November 2014 from KPMG to Ivanhoe on the subject of Platreef Resources: Updated commentary on specific tax consequences applicable to an operating mine in the Republic of South Africa. 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.

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

3.4 Environmental

Bernard Peters, the QP for the legal, political, and 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 relied upon and disclaim responsibility for this information in support of the resource estimate in Section 14.

4 PROPERTY DESCRIPTION AND LOCATION

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

4.1 Location

The Platreef Project centroid is located at approximately 24°05'S and 28°59'E. The Platreef Project is located on three farms: Turfspruit 241 KR (3,561 ha), Macalacaskop 243 KR (4,281 ha) and Rietfontein 2 KS (2,878 ha), in the Limpopo Province of the Republic of South Africa (Figure 4.1).

4.2 Property and Title in South Africa

Information in this sub-section was based on public domain sources, and the country review prepared by Ivanhoe in support of their 2012 Initial Public Offering (Ivanhoe, 2012). 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.

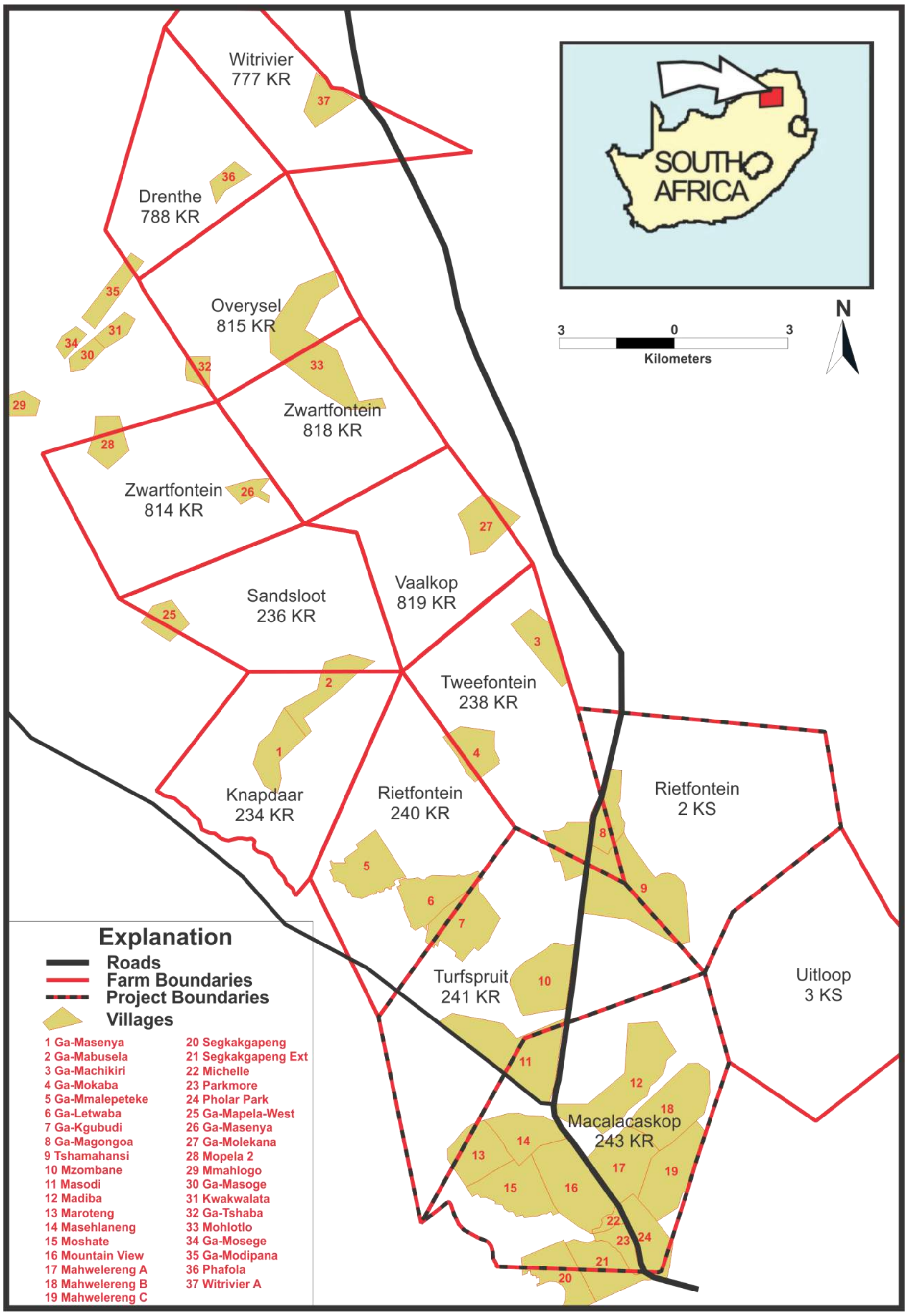
Under the South African Mining Charter of 2004 (the Charter), companies are required to divest a portion of their investments to Historically Disadvantaged South Africans (HDSAs), as a condition of the conversion of old-order mining rights to new-order mining rights. In the Charter, mining company ownership 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.

Figure 4.1 Project Location and Farm Boundaries



Khaki areas on the plan are the main settlements and townships. Figure by Amec Foster Wheeler, 2013; data from Ivanhoe.

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.

4.2.2 Surface Rights Title

Under a common-law position previously in force in South Africa, which was supported by the 1991 Act, a land owner was the owner of the whole of the land, including the air space above the surface and everything below it. The MPRD Act replaced this common-law position, and the 1991 Act was repealed by the MPRD Act.

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.

4.2.3 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 phase 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.

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

Ivanhoe Mines Ltd. effectively (directly and indirectly) holds 64% of Ivanplats (Pty) Ltd, through an interest in Ivanplats Holding SARL (formerly Beales SARL). The minority interests held in the Platreef Project and in Ivanplats Holding SARL are held by Itochu Corporation (Itochu) and ITC Platinum Development Ltd. (ITC), a consortium of Itochu, the Japanese state-owned Japan Oil, Gas and Metals National Corporation (JOGMEC) and JGC Corporation (JGC). The B-BBEE Partners, comprising 20 local communities, local entrepreneurs, and employees, hold the remaining 26% in the Platreef Project. Figure 4.2 and Figure 4.3 present the Platreef Projects' ownership structure before and after the proposed B-BBEE transaction, respectively. Ivanhoe is the operator of the Platreef Project.

Figure 4.2 Ownership Structure for the Platreef Project Before the B-BBEE Transaction

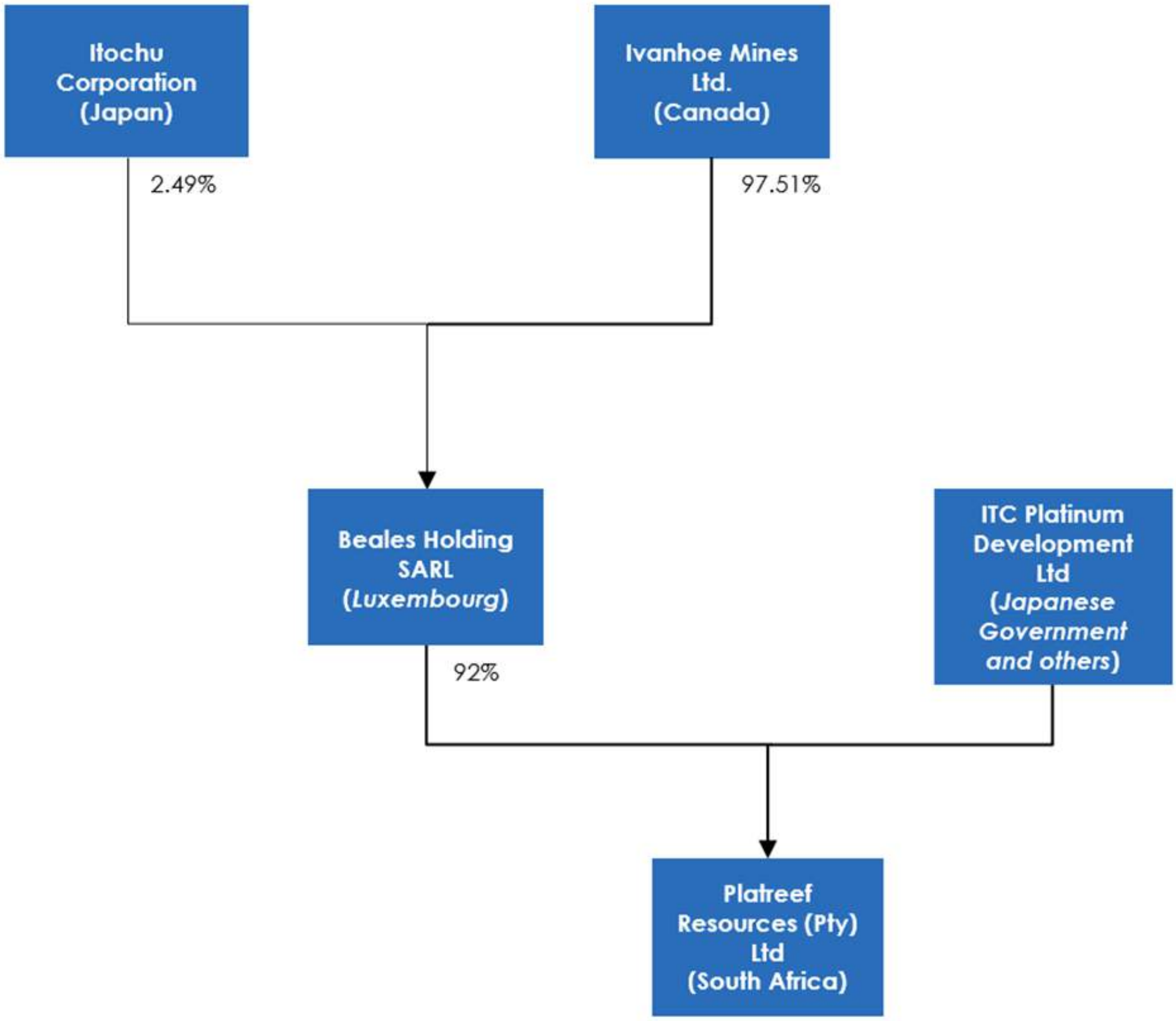


Figure supplied by Ivanhoe, 2014.

Figure 4.3 Ownership Structure for the Platreef Project After the B-BBEE Transaction

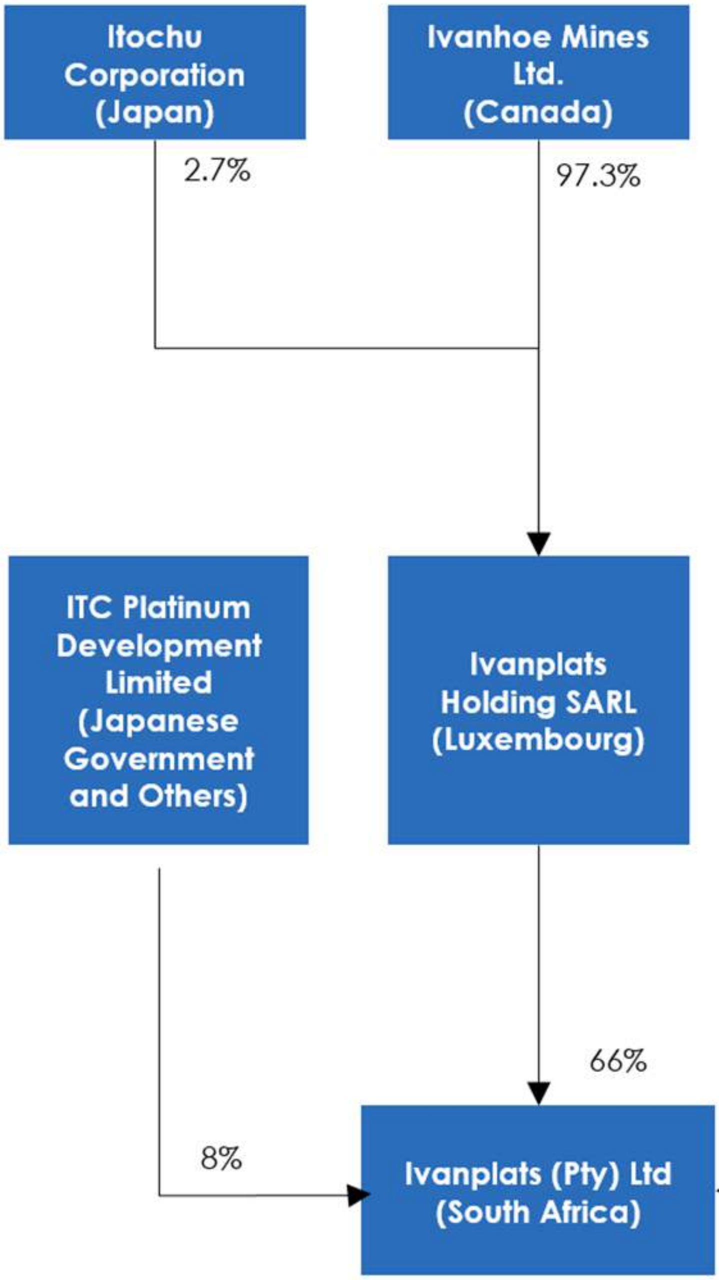


Figure supplied by Ivanhoe, 2014.

In October 2010, Itochu acquired a 2% interest in PR LP30/5/1/1/2/872PR from Ivanhoe for $10 million (840 million Japanese Yen). On 26 May 2011, Itochu announced the acquisition of an 8% direct interest in the Prospecting Right from Ivanhoe through Itochu's affiliate ITC for an additional $280 million (22.4 billion Japanese Yen), and has concluded a Joint Operation and Investment Agreement with Ivanhoe (Itochu, 2011). Consequently, Itochu and ITC (collectively the Itochu Consortium) holds an aggregated interest of 10% in PR LP30/5/1/1/2/872PR; (now MR LP30/5/2/2/1/10067MR); Ivanhoe owns 64%. The Itochu Consortium's cash contribution will be applied to exploration and development activities.

Itochu established a 100%-owned subsidiary, Itochu Mineral Resources Development Corporation, with intentions to undertake exploration and development projects in the mineral resources sector. Itochu's Platreef Project participation is one of the projects that will be promoted by and between Itochu and this newly-established subsidiary. Additional information on the Itochu Agreement is included in Section 4.8.2.

Ivanhoe has entered into a series of agreements in order to comply with and give meaningful effect to Section 2(d) of the MPRD Act, No. 28 of 2002.

Ivanhoe holds a converted old-order prospecting right in respect of the Platreef Project in South Africa: in order to proceed with the development of the Platreef Project, Ivanhoe has applied for a mining right, which has been approved on 30 May 2014 and activated on 4 November 2014.

4.5 Mineral Tenure

Location plans of the farms and PRs discussed in the next sub-sections are provided in Figure 4.1. The PRs, and therefore the prospecting licence boundaries, are the same as the farm perimeter boundaries in the plan. Boundaries of MPT 55/2006 PR (LP30/5/1/1/2/872PR) and MR (LP30/5/2/2/1/10067MR) correspond to the perimeter boundaries of the Macalacaskop 243 KR and Turfspruit 241 KR farms. The boundaries for MPT 76/2007 PR (LP30/5/111/2/740PR) correspond to the perimeter of the Rietfontein 2 KS farm.

Figure 4.4 shows the locations of the townships that have developed within the farming areas, including on farms that are outside the Platreef Project area.

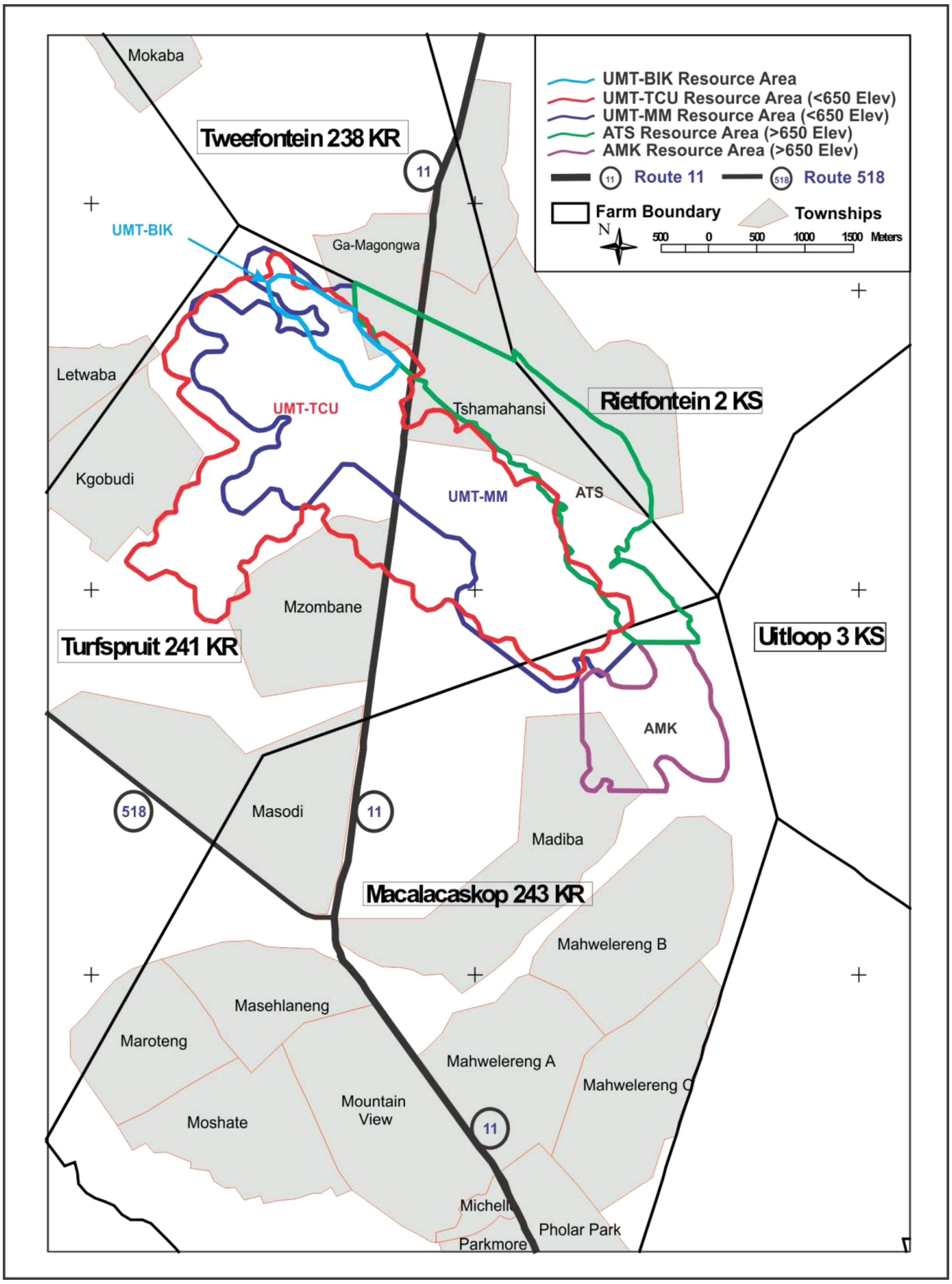
4.5.1 Prospecting Right MPT 55/2006 PR (LP30/5/1/1/2/872PR)

Ivanhoe provided legal opinions and annexes dated 20 November 2006, 21 November 2006, 12 November 2009, 7 September 2012, and 25 March 2014, which reviewed the legal status of the Mineral Lease K2921/2001 on the Turfspruit 241 KR and Macalacaskop 243 KR farms.

These documents support that Ivanplats (Pty) Limited, registration number 1988/000334/07, a subsidiary of Ivanhoe Mines Ltd., holds exclusive prospecting rights to prospect for base minerals and precious metals on the Turfspruit 241 KR and Macalacaskop 243 KR farms. At the outset, these rights were granted in accordance with the 1991 Act. The MR became legally effective in October 2002.

The old-order PR was lodged for conversion in terms of Schedule II of the MPRD Act, No. 28 of 2002, on 3 March 2005 prior to the expiry of the PR, which expired on 6 March 2005. The conversion was granted, and a new-order PR was executed on 2 February 2006, in favour of Ivanhoe and in respect of base minerals and precious metals over the Turfspruit 241 KR and Macalacaskop 243 KR farms, under Mineral Prospecting Right MPT 55/2006, PR LP30/5/1/1/2/872PR, which was to expire on 1 February 2011.

Figure 4.4 Major Township and Farm Locations



The main road indicated on this plan is the N11 highway. The UMT and Bikkuri areas are considered to be amenable to underground mining methods; the AMK and ATS areas are considered to be amenable to open pit mining methods. The boundary of ATS is constrained by the Turfspruit 241 KR farm (mineral tenure) boundary, and the north-eastern boundary of UMT. Figure generated by Amec Foster Wheeler 2013, information from Ivanhoe.

In terms of Section 18(4) of the MPRD Act, the Platreef Project PR may be renewed once for a period not exceeding three years. Ivanplats made application to renew the PR for a three-year extension of term prior to the expiry date. Ivanplats was notified by the DMR on 4 May 2011 that the PR had been renewed for a further three-year term, and the relevant Notarial Deed of Renewal was executed and commenced on 1 June 2011 ending on 31 May 2014, unless cancelled or suspended in terms of section 47 of the MPRD Act. The renewal was registered in the Mineral and Petroleum Titles Registration Office on 4 November 2011.

For the title to have been maintained, Ivanplats was obligated to pay the required annual title fees and to comply with the relevant obligations and work programmes relating to its prospecting activities on the PR.

Ivanplats confirmed that all the required payments have been made and that all reporting in terms of the prospecting activities are up to date as at the effective date of the Report.

4.5.2 Mining Right LP30/5/2/2/1/10067MR

A PR can only be renewed for one three-year period under the MPRD Act. This renewal has occurred in respect of the Platreef PR. To maintain tenure continuity over Mineral Prospecting Right MPT 55/2006, Ivanhoe would need to apply for a MR prior to expiry of the PR. Subject to complying with the provisions of the PR, the holder has the exclusive right, in Section 19(1)(b) of the MPRD Act, to apply for and be granted a MR in respect of the mineral and prospecting area in question.

On 6 June 2013, Ivanhoe electronically lodged, through the South African Mineral Resources Administration System (SAMRAD) portal, an application for a MR in terms of Section 22 of the MPRD Act 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, situated in the Magisterial District of Mokerong, Limpopo Province, for a period of 30 years. Hard copy files of the Mining Right Application were hand-delivered to the DMR on 18 June 2013.

The MRA consisted of a set of documents with the three main components being the following:

- Mining Works Programme (MWP);
- Social and Labour Plan (SLP); and
- Black Economic Empowerment plan (BEE).

On 17 July 2013, the DMR notified Ivanhoe that it had accepted its application for a MR on the Macalacaskop 243 KR and Turfspruit 241 KR farms and requested Ivanhoe to comply with inter alia the following:

- To conduct an EIA and submit seven copies or folds of the environmental management program on or before 13 January 2013 but not earlier than 14 October 2013;
- To submit a scoping report in terms of Regulation 49(2) on or before 16 August 2013;
- To notify and consult with the landowner or lawful occupier and any other affected party; and

- To consult with the Department of Land Affairs if the land is state owned and in the event that the land is subject to land restitution to consult with the office of the Commission of Restitution for Land Rights and submit the result of such consultation to this office on or before 13 January 2013.

Due to several of the dates referred to in the aforementioned letter referring to the incorrect year (an apparent typing error), the DMR issued a revised acceptance letter on 26 August 2013, where in the DMR inter alia confirmed that the Mining Right Application had been accepted and directed Platreef to:

- Conduct an EIA and submit seven copies or folds of the environmental management program on or before 24 February 2014 but not earlier than 25 November 2013;
- Submit a scoping report in terms of Regulation 49(2) on or before 25 September 2013;
- Notify and consult with the landowner or lawful occupier and any other affected party; and
- Consult with the Department of Land Affairs if the land is state owned and in the event that the land is subject to land restitution to consult with the office of the Commission of Restitution for Land Rights and submit the result of such consultation to this office on or before 24 February 2014.

Ivanhoe has complied with the terms of the acceptance letter in that it:

- Submitted a scoping report to the DMR on 16 August 2013 and to the Department of Environmental Affairs and the Limpopo Provincial Department of Economic Development, Environment and Tourism (LEDET) on 23 August 2013;
- Submitted an EIA and Environmental Management Plan to the DMR on 13 January 2014 and to LEDET on 15 January 2014; and
- Submitted a confirmatory letter from the Department of Rural Development and Land Reform (DRDLR) to the DMR on 13 December 2013.

Consultations with communities and DRDLR are on-going.

Ivanhoe advised that the Mining Right Application submitted by Platreef Resources meets all the above requirements.

4.5.3 Prospecting Right MPT 76/2007 PR (LP30/5/1/1/2/740PR)

Atlatsa Resources Corporation (Atlatsa; formerly Anooraq Resources Corporation) through its South African Subsidiary Plateau Resources Limited, holds exclusive prospecting rights to prospect 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 Prospecting Right MPT 76/2007 PR. The Prospecting Right 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 prospecting right 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 mining right. To date, Ivanhoe has advised that to thebest of its knowledge, a mining right has not been issued over Rietfontein. The JV is valid until the expiry of the prospecting right.

4.6 Surface Rights

The land over which the Mineral Prospecting Right MPT No. 55/2006 PR is held, is owned by the State and held in trust for the respective communities. The Madiba, Masodi, Masehlaneng, Maroteng, Moshate, Mahwelereng (A, B, C), Pholar Park, Parkmore, Mountain View, and Michelle communities are the lawful occupiers of the Macalacaskop 243 KR farm, and the Tshamahansi (Hlongwane, Baloyi and Matjeke), Kgobudi, Masodi, and Magongoa communities are the lawful occupiers of the Turfspruit 241 KR farm (see Figure 4.4) Rights to prospect and mine the land are granted by the State.

In terms of Section 5 of the MPRD Act, the holder of a prospecting right is entitled, among other things, to enter the land to which the right relates together with its employees, to bring machinery and equipment onto the land to lay down and erect infrastructure, to prospect and carry out activities incidental to prospecting.

Prior to the Mineral and Petroleum Resources Development Amendment Act 29 of 2008 coming into force on 7 June 2013, it was required, before the holder may commence with prospecting, to notify and consult with the owner or lawful occupier of the land. The owner or lawful occupier of the land is entitled to compensation for losses and damages suffered or likely to be suffered as a result of the proposed prospecting operation.

In the absence of an agreement between the holder and the owner or lawful occupier, compensation for losses and damages must be determined by arbitration or a competent court.

Ivanhoe advised that the company had undertaken extensive consultation with the communities who are the lawful occupiers of the prospecting area, and surface use and co-operation agreements regulating among other things the compensation for losses and damages had been entered into with four local communities during 2010 as follows:

- Kgobudi Community in September 2010.
- Magonoa Community in June 2010.
- Tshamahansi Community in October 2010.
- Madiba Community in April 2010.

Additional community consultations will be required; the legal opinion provided indicates that there are internal differences of opinion within some of the communities as follows:

- Disagreements as to whether the community authorities who entered into the access agreements had the authority and legitimacy of their respective representative bodies to conclude the agreements. Disagreements as to where the access payments are being made.
- Disagreements as to the lack of involvement of the DRDLR in negotiating and signing the agreements.

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 Project.

Long-term surface lease agreements will have to be concluded when the mining phase commences in order to cater for the required surface mining and plant infrastructure. Platreef Resources is currently in the process of consulting with communities in order to start negotiations for a long-term surface lease. This is a lengthy process, governed by the DRDLR State Land Lease and Disposal Policy of 25 July 2013, as these agreements are concluded with the DRDLR and Traditional Authority, but need to be agreed to by the community by way of resolution. The resolution is passed by voting on a show of hands at a public meeting in the affected community, which meeting was convened for that purpose. As one of the first steps in the long-term surface lease process, Ivanhoe has obtained valuations of the land from two independent DRDLR recommended valuators, surveyed the proposed lease area, and appointed an experienced facilitator, selected from the DRDLR database, to assist in and facilitate the process. This surface lease process provides for monetary compensation that will be paid to a trust for the affected communities.

4.6.1 Land Claims

Land claims by HDSAs have been lodged with a government commission over many regions of South Africa. All such South African land claims are to be reviewed by a governmental entity.

Ivanhoe noted it may have to pay some form of compensation to any claimants who are granted land as a consequence of such successful assertions. In the event of a claim succeeding, the claimant is entitled to restoration of the actual land claimed or to “equitable redress”.

The Rietfontein 2 KS farm has been claimed by the Mamashela Community, and the claim has been gazetted.

In a letter from the DRDLR, dated 16 April 2012, the department confirmed that a claim for restitution has been lodged over the Turfspruit 241 KR farm by the Mokopane Tribe. As of the Report effective date, the claim had not been gazetted.

Ivanhoe has requested that Digby Wells, during the environmental studies that Platreef has commissioned, confirm the land status, and assist with resolving any queries regarding potential claims in an equitable manner.

4.6.2 Macalacaskop 243 KR and Turfspruit 241 KR Farms

The Ivanhoe-controlled farms, Macalacaskop 243 KR and Turfspruit 241 KR, are contiguous, sharing a common boundary along the north-west border of Macalacaskop 243 KR and the south-eastern border of Turfspruit 241 KR farm. Macalacaskop 243 KR farm comprises 4,281 ha of land. Turfspruit 241 KR farm comprises 3,561 ha of land. The combined total is 7,842 ha.

The Macalacaskop 243 KR and Turfspruit 241 KR 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 the Northern Province) of South Africa. Macalacaskop 243 KR farm is filed at that location under reference SG 1496/1894. Turfspruit 241 KR is filed at the same location as reference SG A44/1963. Plot surveys and land area calculations were performed by the Surveyor General as indicated on the registered diagrams: SG Diagrams A 44/63 (Turfspruit 241 KR) and A 45/63 (Macalacaskop 243 KR).

4.6.3 Rietfontein 2 KS Farm

Rietfontein 2 KS farm has a contiguous border with Turfspruit 241 KR farm, sharing a common boundary along the south-western border of Rietfontein 2 KS and the north-eastern border of Turfspruit 241 KR. Rietfontein 2 KS farm has an area of 2,878 ha. Surveys and land area calculations were performed by a professional land surveyor. The Rietfontein 2 KS farm has 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 the Northern Province) of South Africa.

Rietfontein forms part of the approved Environmental Management Plan for the Project, and a portion of the farm will form part of the surface lease for the Project. The JV is still in place until next year.

A mining right is not required for a tailings dam. Ivanplats will need to obtain a surface lease. Rietfontein is owned by the State, and held in trust for the community. The community in question, Mamashela Community, have lodged a land claim with a land claims commissioner for the Farm Rietfontein, and the status is currently that their claim has been gazetted (approved), but they have not yet taken full ownership and received the title deed for the farm. Ivanplats is in the process of negotiating a Surface Use & Cooperation Agreement (SUCA) with the Mamashela Community in order to secure access for the tailings dam. Ivanplats will also include Rietfontein in the long-term surface lease process.

4.7 Royalties and Encumbrances

The Turfspruit prospecting licence was subject to an initial royalty agreement in 2001 with the Lebowa Minerals Trust (the Trust). A second agreement, which superseded the first, was later signed with the Trust. Upon conversion of the old-order lease to a new-order lease, under the Transitional Provisions of the MPRD Act, old-order rights, which include such provisions as contained in the second Trust agreement, lapsed.

Although the Transitional Provisions do make an exception for the continuation of payment of royalties to communities, the Trust was dissolved by an Act of Parliament, with the rights of the Trust then vested in the South African government. There are also tax-related provisions for continuation of payments required under old-order rights for removal and disposal of minerals; however, the agreement between the Trust and Ivanhoe provided for prospecting payments, and not for removal of minerals, and also does not apply.

4.8 Property Agreements

4.8.1 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; Atlatsa were considering the offer as of the effective date of the Report.

4.8.2 Itochu Agreement

In October 2010, Ivanhoe entered into an Earn-in Agreement with Itochu Corporation (the Itochu Earn-in Agreement). Under the Itochu Earn-in Agreement, Itochu purchased a 2% interest in Beales SARL (Beales), a currently 90%-owned subsidiary of Ivanhoe. Beales owns the holding company Platreef Resources (Pty) Ltd (Platreef Resources), which holds the Platreef Project.

On 26 May 2011, Itochu acquired, through its affiliate ITC Platinum Development Ltd (ITC), an additional 8% interest in the Platreef Project, indirectly through Beales, through a Joint Operation and Investment Agreement (JOIA). The JOIA includes various adjustment and other clauses relating to the Beales shareholdings such that on enactment of the JOIA on 6 June 2011, the effective participating interests in the underlying Platreef Project became as follows: Ivanhoe – 90%, Itochu – 2% and ITC – 8%.

Under the JOIA, Ivanhoe granted Itochu and ITC (collectively Itochu) a number of rights intended to preserve Itochu's minority interest in the Platreef Project. Such rights include:

- A covenant that prohibits dilution of Itochu's proportional ownership interest in the Platreef Project as a result of a BEE investment.
- A pre-emptive right that permits Itochu to maintain its proportional interest in the Platreef Project as a result of any other issuance of securities at a price equal to the subscription price for those securities.
- A right of first offer to purchase the equity stake held by Ivanhoe in Beales or on a sale by Beales or Platreef Resources of an interest in the Platreef Project holdings.
- A "tag-along" right of Itochu in which it will be entitled to put its interest along with a sale by Ivanhoe of a significant equity stake in the Platreef Project holdings on the same terms and conditions as Ivanhoe receives from such a sale.

The parties have also agreed to establish a technical committee and a management committee in which Itochu will, in each case, be entitled to appoint two of six members so long as it holds no less than a 2% interest in the Platreef Project.

The JOIA provides for cash calls for development funding by the two parties, and dilution to the extent funding is covered by the other party. To the extent that Itochu's interest in the Platreef Project falls below 2%, its interest will be converted into a 1% net smelter return royalty.

The JOIA provides for preferential third-party Japanese participation in the future operations of the Platreef Project. In particular, Itochu has covenanted to assist in securing Project financing for the development of the Platreef Project, while the parties have agreed to provide either Itochu or an Itochu-facilitated financial-assistance entity a right to off-take of production at commercial rates from the Platreef Project.

Finally, on a change of control, certain rights of the parties will be terminated and, to the extent that financial assistance has been provided, the JOIA acknowledges that such financial assistance will be reviewed, and repayment may be accelerated.

4.9 Environmental Studies

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

4.10 Permits

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

4.10.1 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 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.

4.11 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.

4.12 Comments on Section 4

In the opinion of the QP Bernard Peters, the information discussed in this section supports the Platreef 2016 Resource Technical Report. The following key points are of note:

- Information provided by legal experts and Ivanhoe supports Ivanhoe's ownership claims to PR MPT 55/2006PR. Ivanhoe applied for a MR on 6 June 2013, prior to expiry of PR MPT 55/2006PR 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 BEE 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 MPT 76/2007PR.
- 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

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

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 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; and 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 conducted, 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. For a more detailed discussion of the power supply and associated infrastructure for the Platreef Project please refer to Section 18.5.

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 SA 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 has been paid. The latest forecast energisation date of the Platreef Eskom incoming substation is H2'18.

Ivanplats has requested that Eskom complete the budget quote study for 70 MVA that considers a premium supply option. The latest Eskom programme forecasts the completion of the EIA towards the end of 2014, where after the lands and rights, and surveying processes will commence prior to the finalisation of the budget quotation by the middle of 2015. The latest forecast energisation date of the Eskom Platreef Project incoming substation is H2'17.

Based on the prefeasibility study design work prepared by Stantec and DRA, 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

Based on the prefeasibility study design work prepared by Stantec and DRA, the Platreef Project power requirement for a 4 Mtpa underground mine have been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats has notified of and requested from Eskom the 30 MVA additional power demand required 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. The latest forecast energisation date for the 5 MVA construction power supply is Q3'16.

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. For a more detailed discussion of the ongoing initiatives to supply water to the area please refer to Section 18.4.

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 ML/day for domestic use and 78 ML/day for industrial projects i.e. a total of 140 ML/day.

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 have been authorised by the DWA during 2014 (Figure 5.2).

Figure 5.1 Location Plan Flag Boshielo Dam and Proposed Water Pipeline

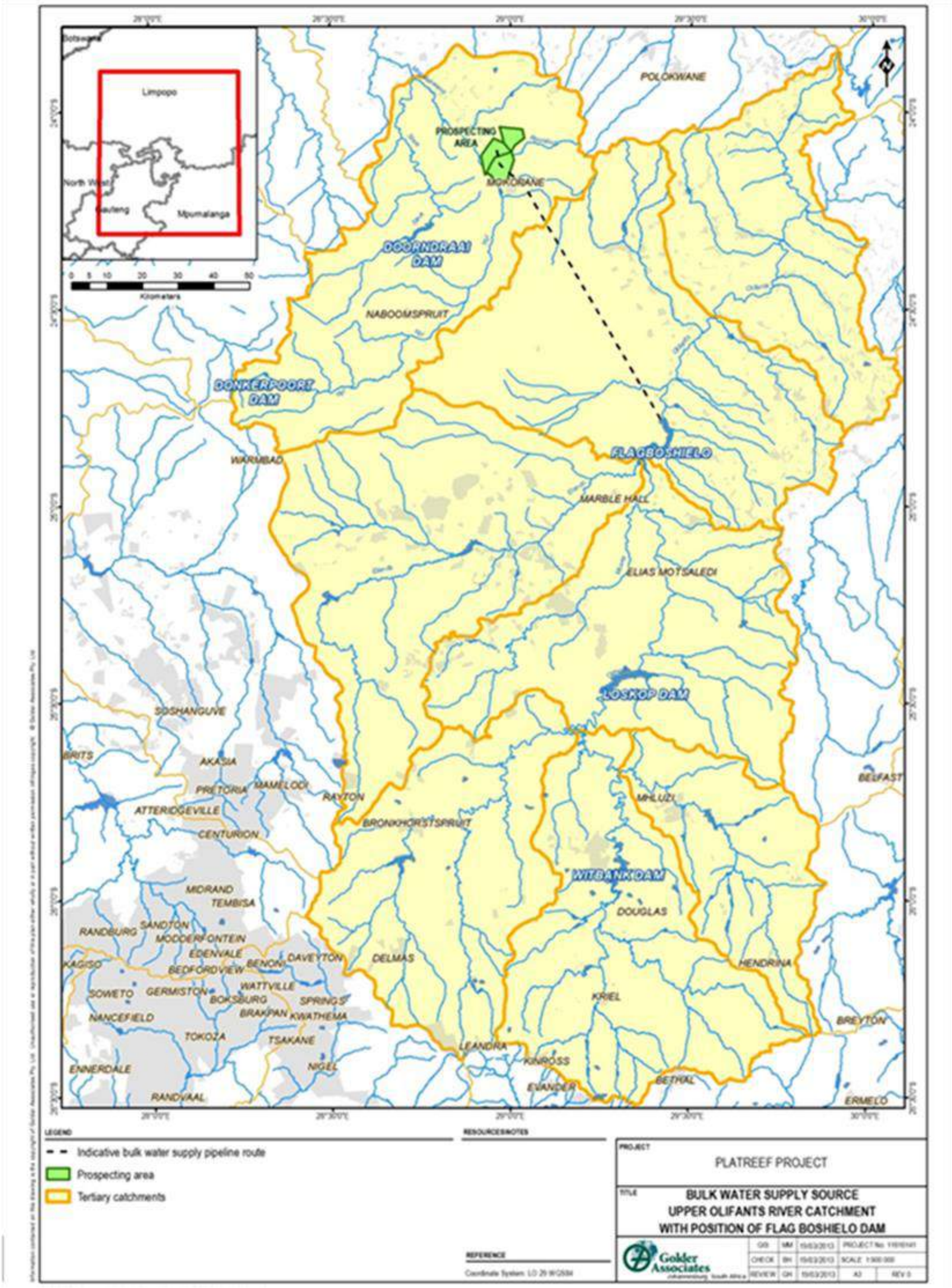


Figure courtesy Ivanhoe, 2013.

Figure 5.2 Water Bore Location Plan

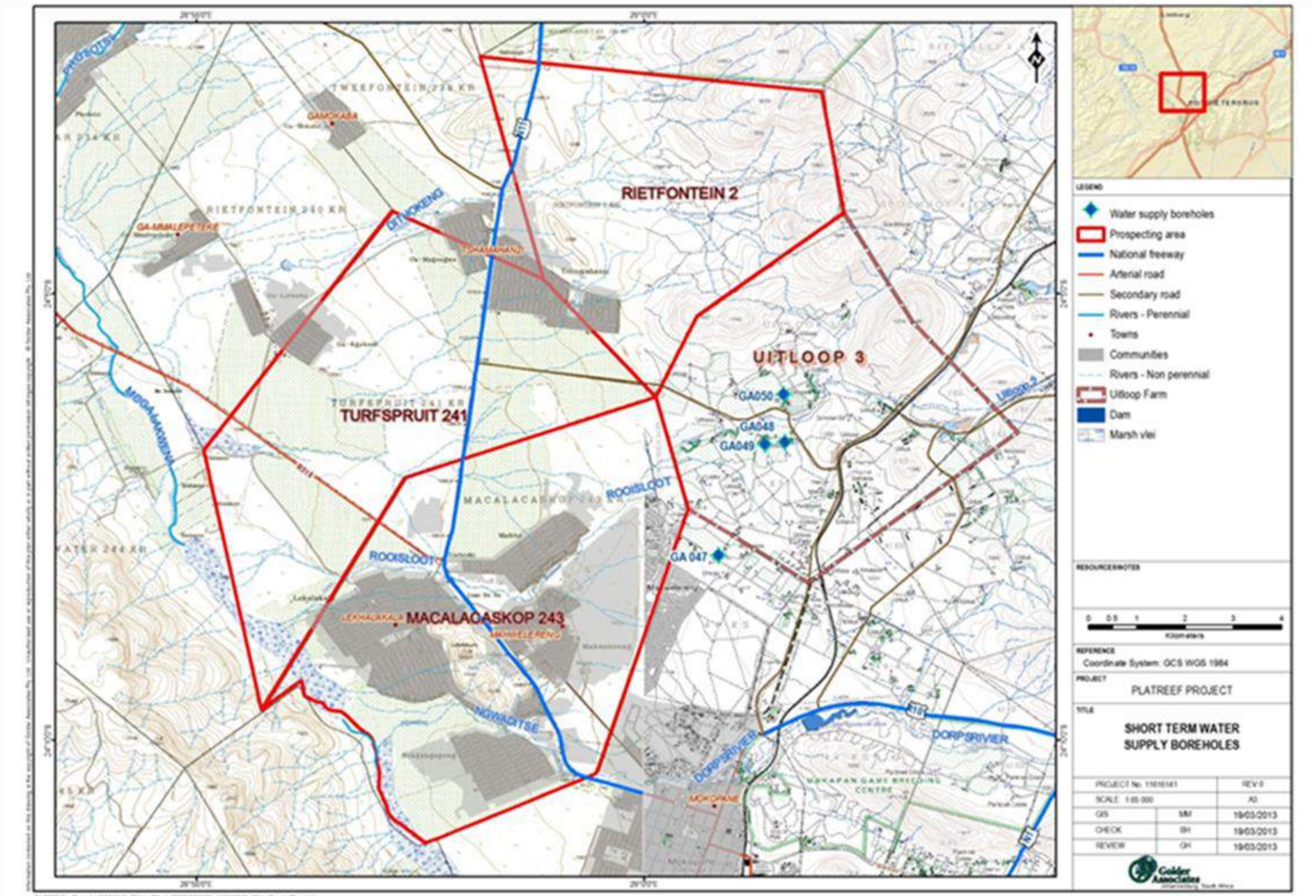


Figure courtesy Ivanhoe, 2013.

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

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

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 2016 Resource Technical Report.

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

Work completed by Ivanhoe consists of geological mapping, airborne and ground geophysical surveys, limited trenching, percussion drilling over the Platreef sub-crop, core drilling, petrography, density determinations, geotechnical and hydrogeological investigations, metallurgical testwork, 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 testwork, and conceptual mining studies that considered open pit scenarios. Results of this work indicated that the mineralisation on the Turfspruit 241 KR and Rietfontein 2 KS farms was more likely to support a mining operation than the mineralisation on the Macalacaskop 243 KR farm.

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

Mineral Resource estimates for the underground deposit were updated multiple times in internal documentation between 2007 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 a preliminary economic assessment titled "Platreef 2014 PEA" (www.sedar.com, Peters et al., 2014) and in January 2015 a prefeasibility study titled "Platreef 2014 PFS" (www.sedar.com, Peters et al., 2015).

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. 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

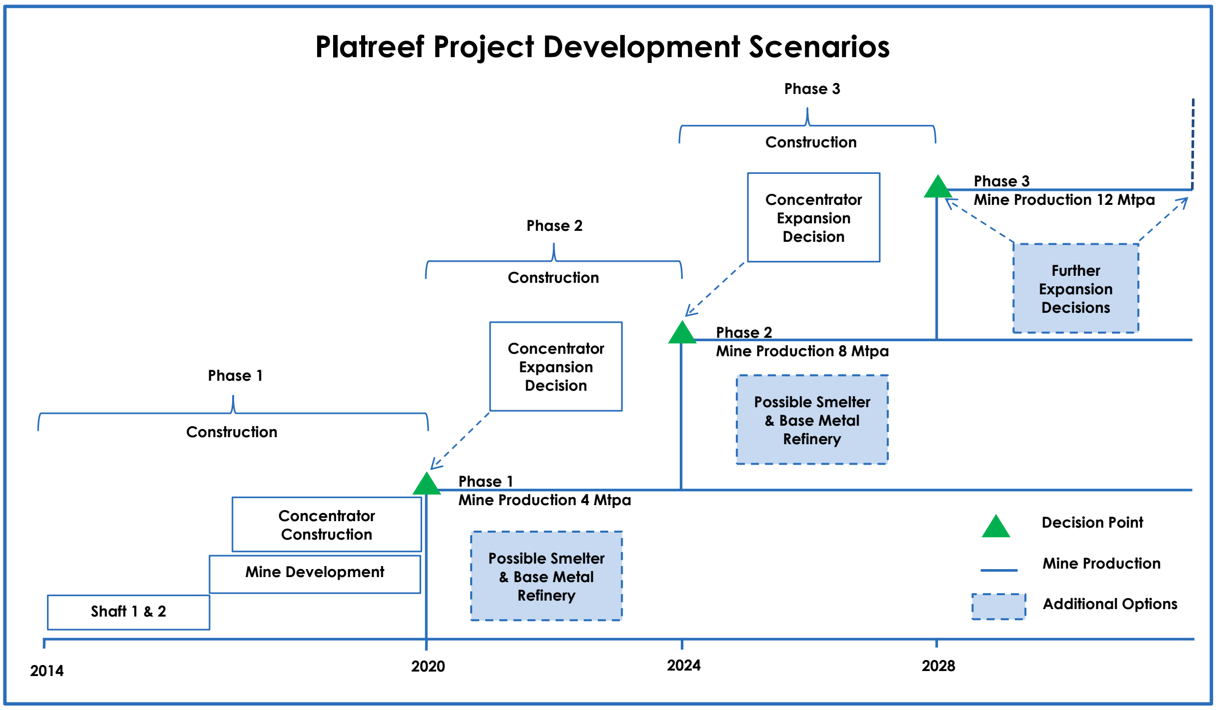


Figure by OreWin 2014.

Table 6.1 Platreef 2014 PEA Results

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.
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.

6.3 Summary of Financial Results in Platreef 2014 PEA

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

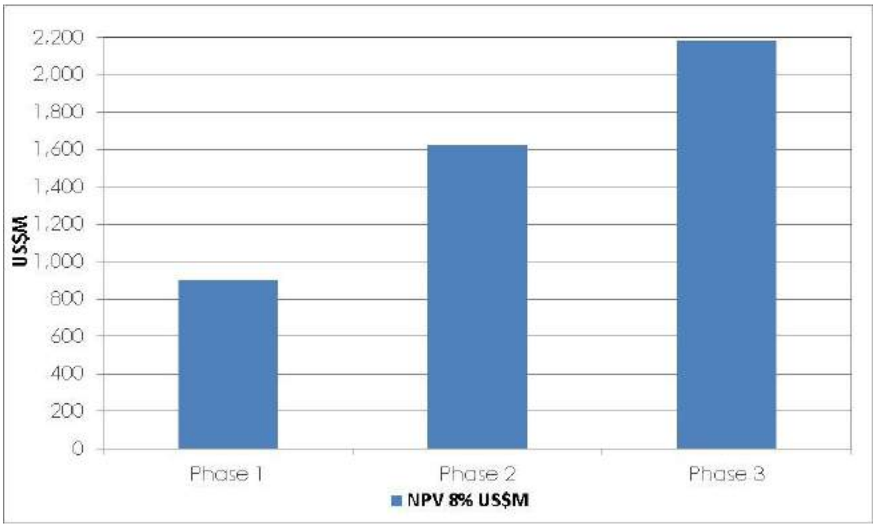


Figure by OreWin 2014.

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

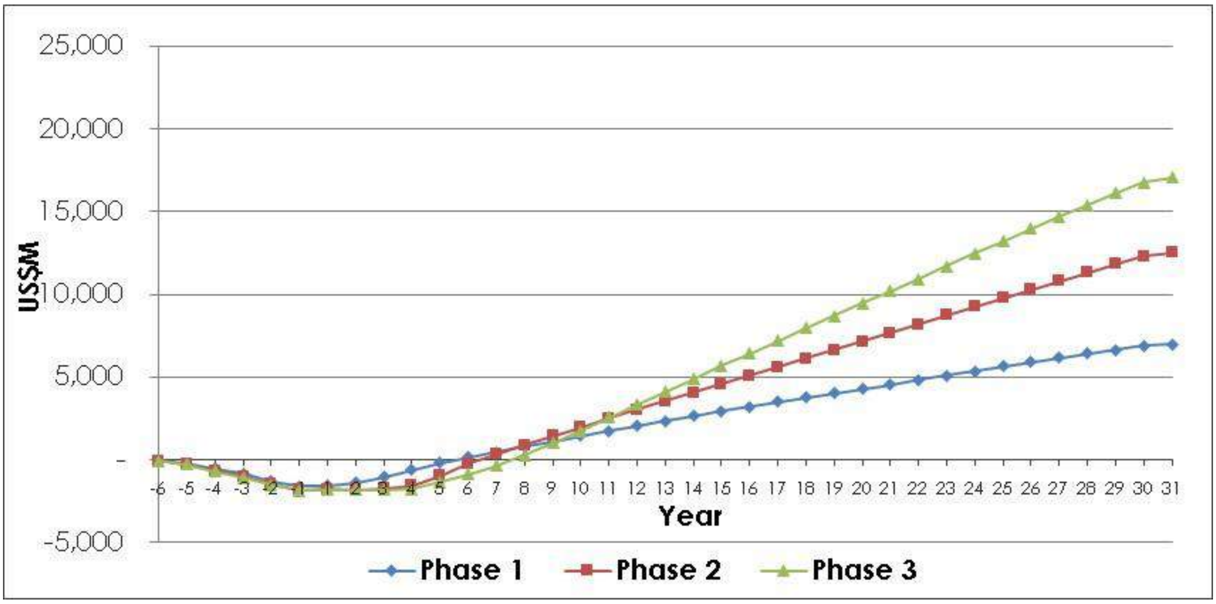


Figure by OreWin 2014.

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

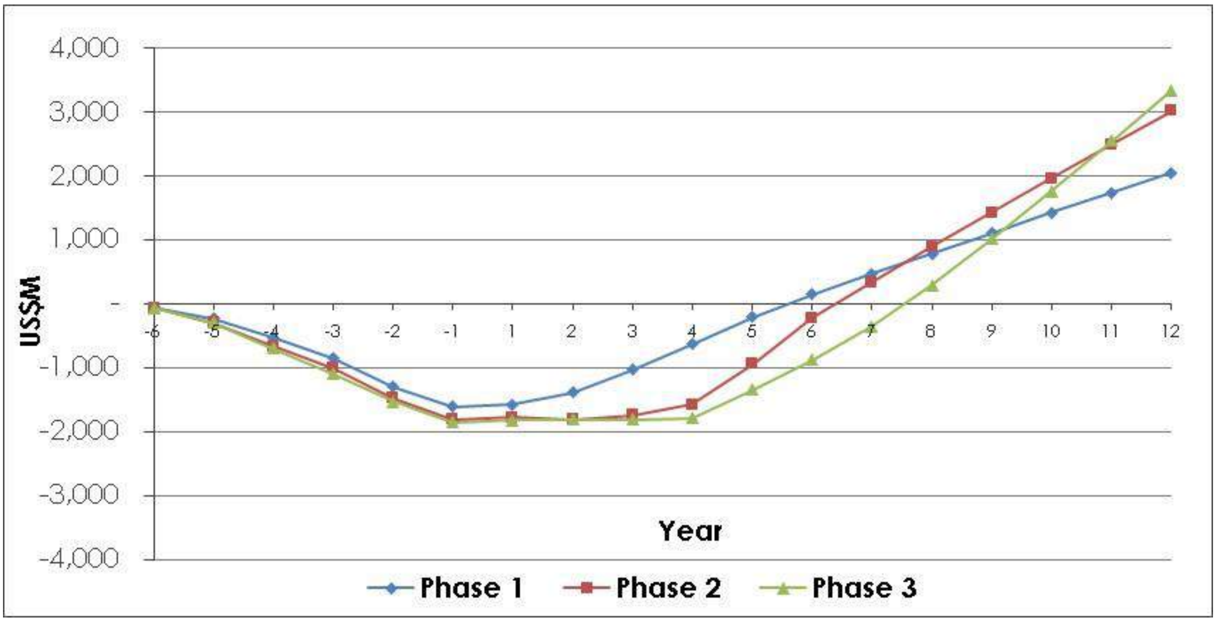


Figure by OreWin 2014.

6.4 Production Summary

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

Table 6.3 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

6.5 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 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 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 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 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 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 Unit Operating Costs and Revenues

	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 decending 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.

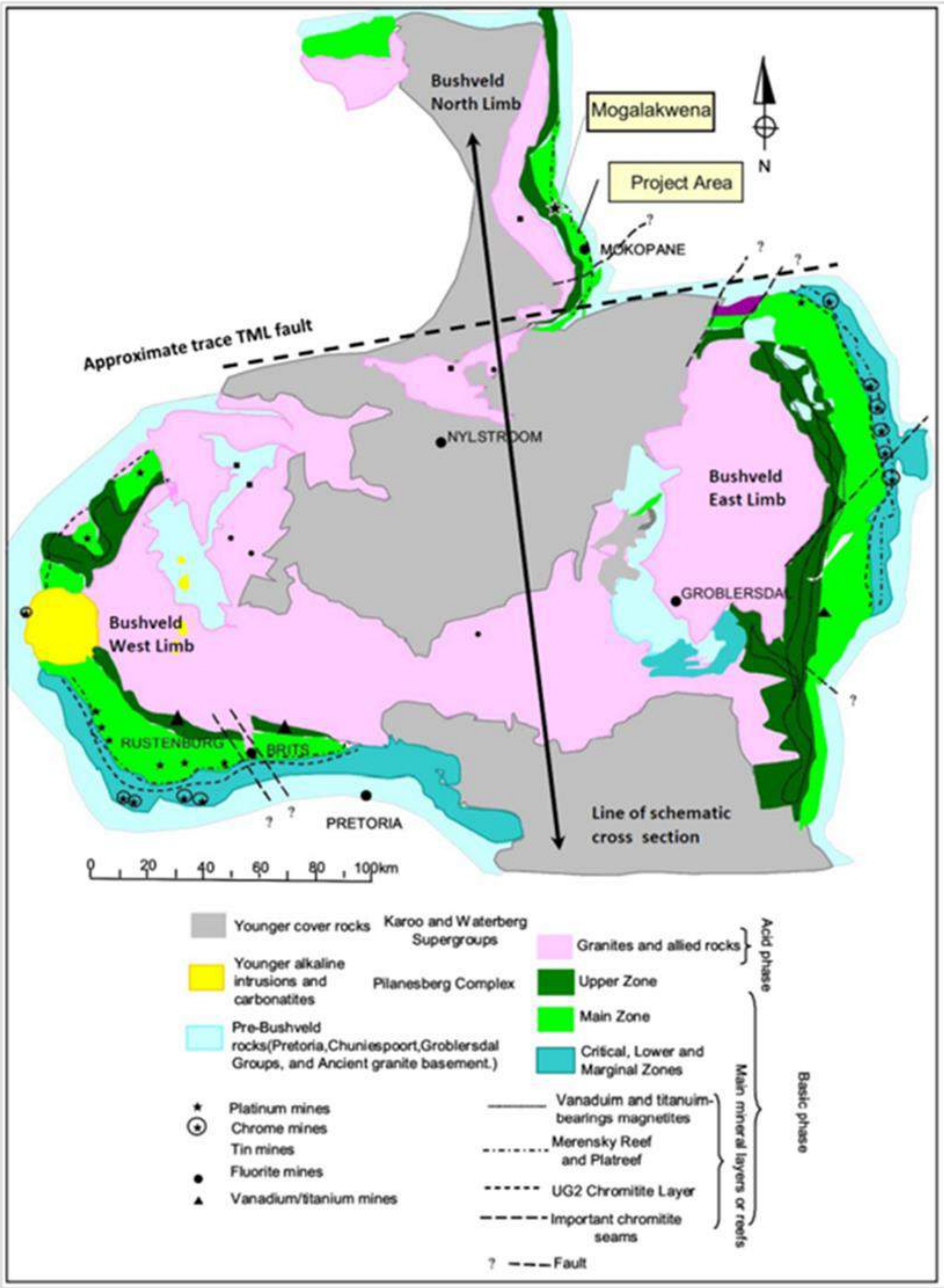
In the East and West Limbs of the BIC, the RLS was intruded into the Magaliesberg Formation of the Proterozoic Transvaal Supergroup. In the North Limb, the RLS intrudes progressively older country rocks northward (Magaliesberg Formation, Malmani Subgroup and Duitschland Formation). Figure 7.1 provides 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

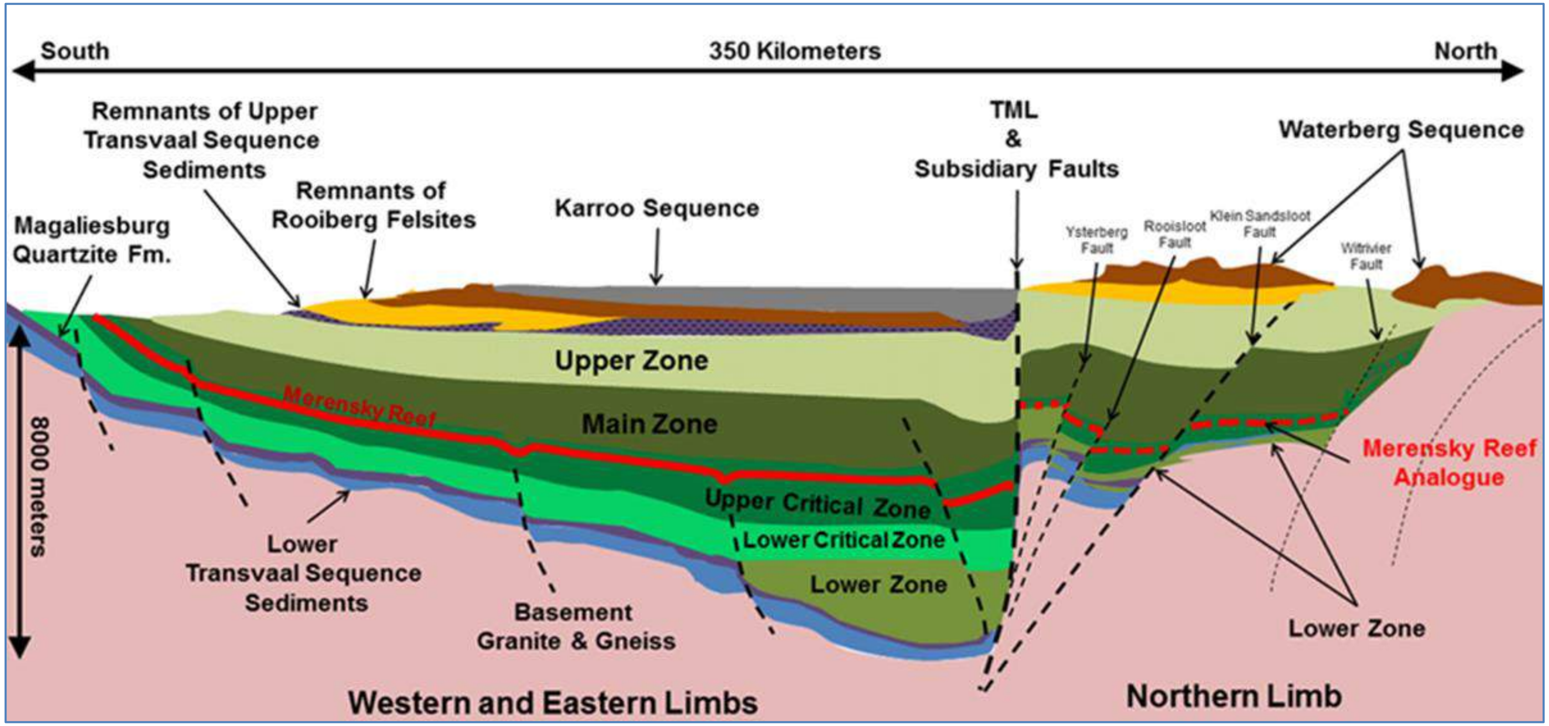
Figure 7.3 shows the Project geology projected to surface. The locations of zones 1 to 5 referred to in the following geologic discussion are shown in Figure 7.4.

Figure 7.1 Regional Geological Plan of the Bushveld Complex



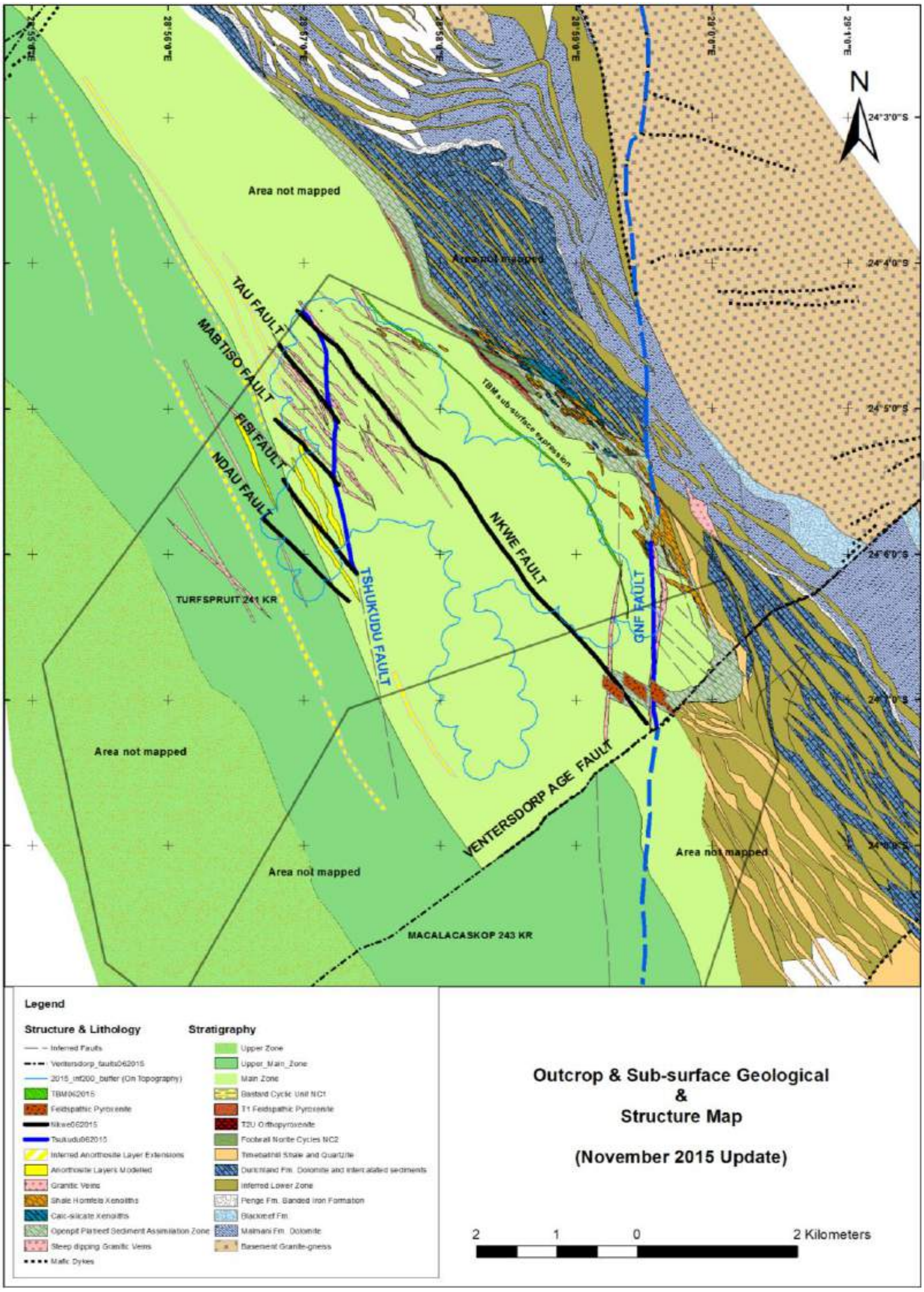
Modified after Viljoen and Schürmann, 1998; section line represents location of section in Figure 7.2

Figure 7.2 Schematic Cross-Section through the Bushveld Igneous Complex



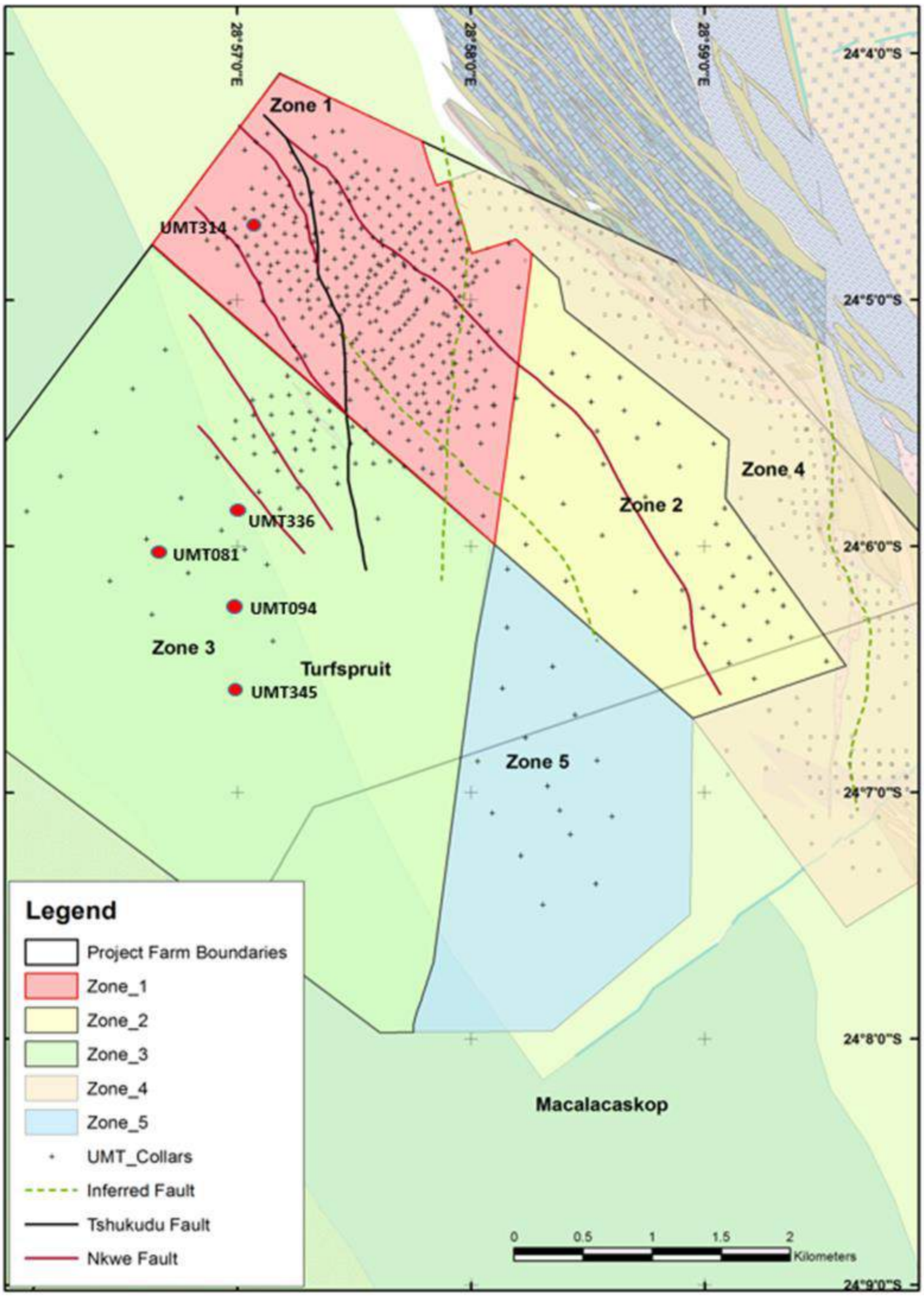
Courtesy Ivanhoe, 2012; modified after Kruger, 2005. Figure is schematic and not to scale. Section line illustrated is shown on Figure 7.1.

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.

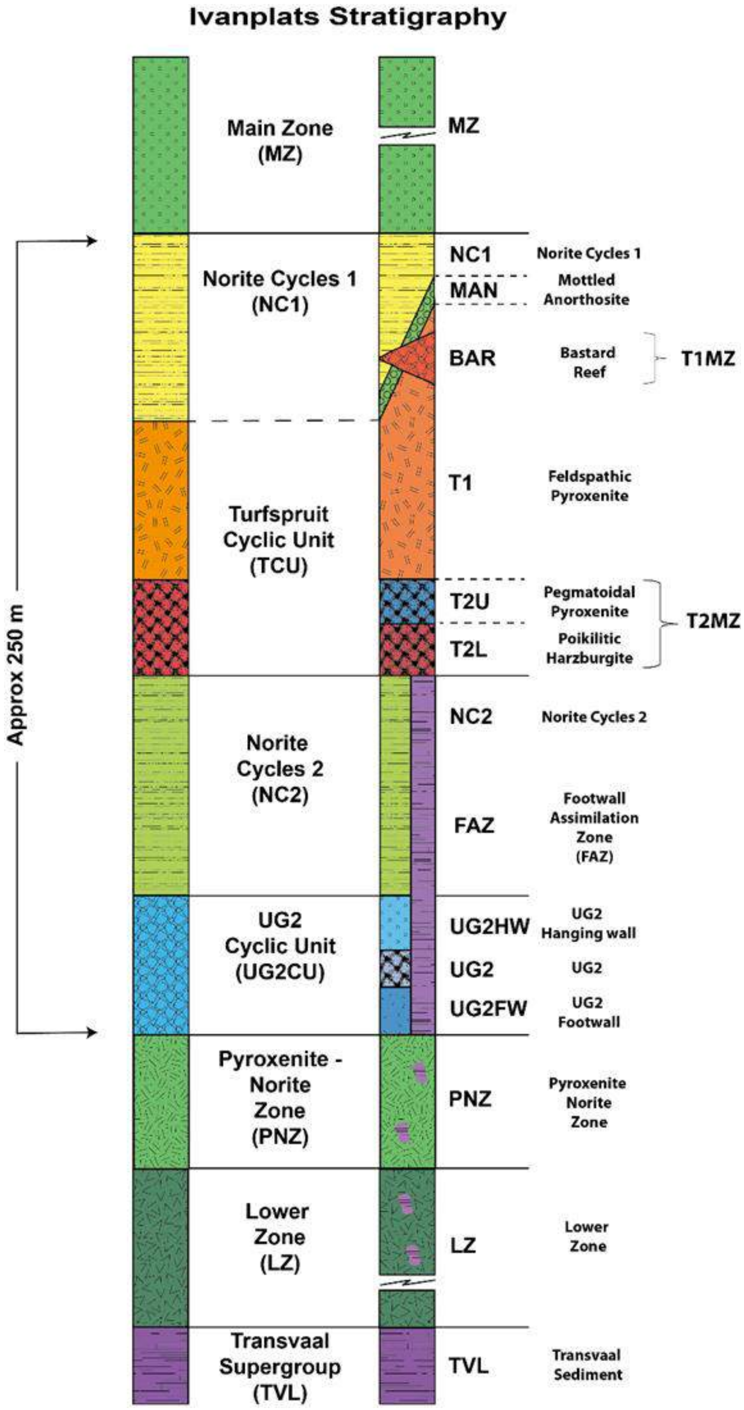
Figure 7.5 shows a comparison between a clean magmatic stratigraphy (left column) and a magmatic stratigraphy including significant sediment interaction (right column).

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).

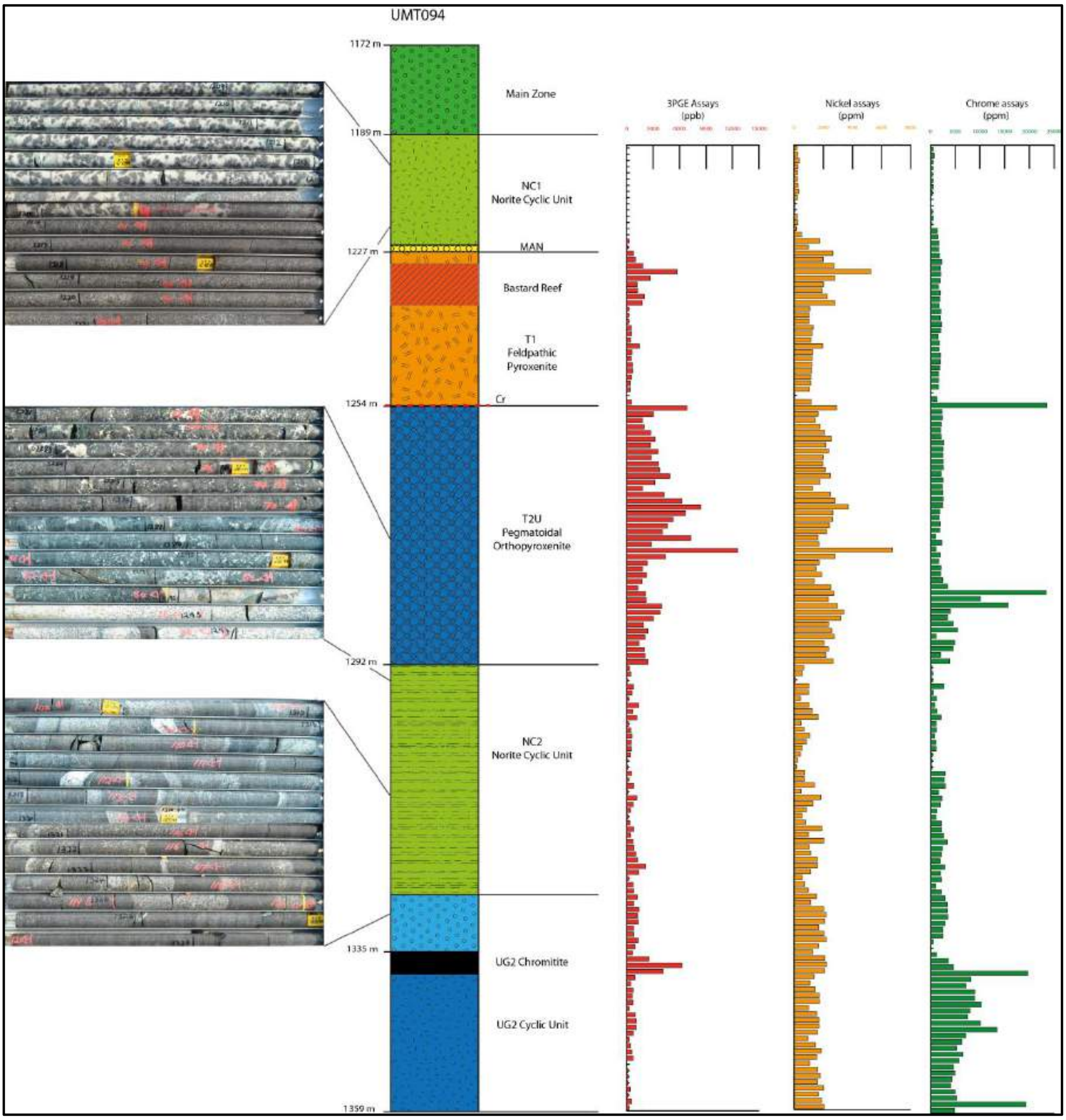
In addition to the above described magmatic stratigraphy, undifferentiated lithologies have also been recognised on the project. The major occurrence is the Footwall Assimilated Zone (FAZ) that occupies a similar stratigraphic position as the NC2 in the well drilled Zone 1 area. The FAZ is a zone of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

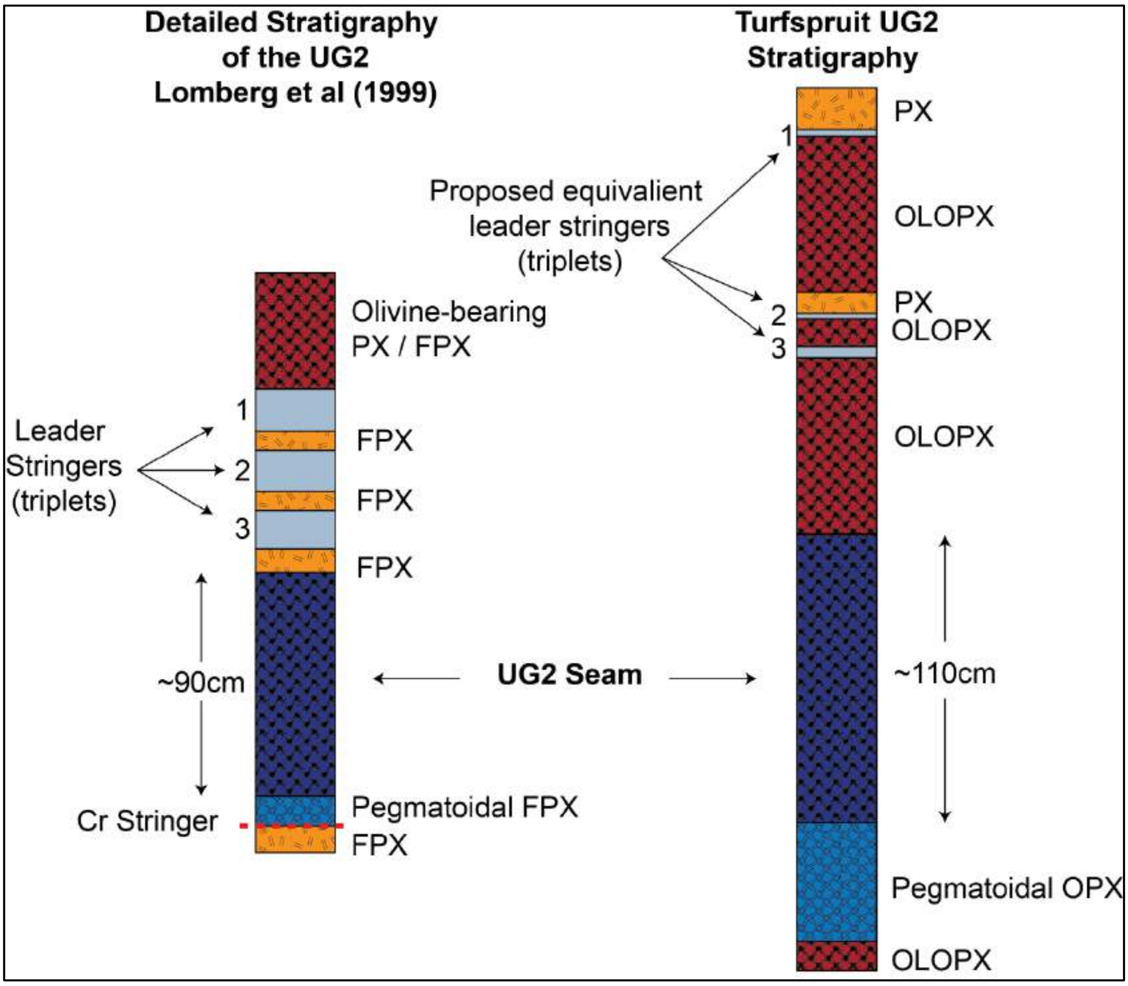
7.2.5 Pyroxenite-Norite-Zone (PNZ)

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 drill holes. 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)

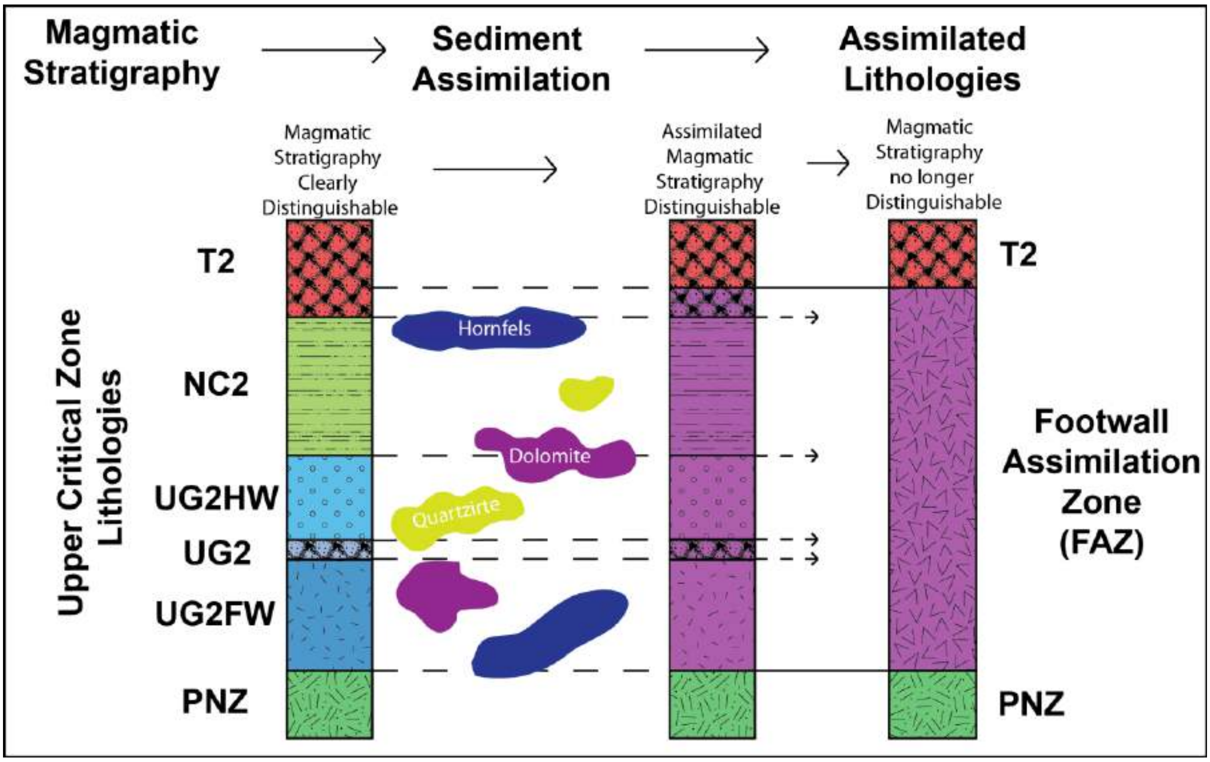


Courtesy, Ivanhoe 2016.

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 theTCU. 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 Assimulated 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.

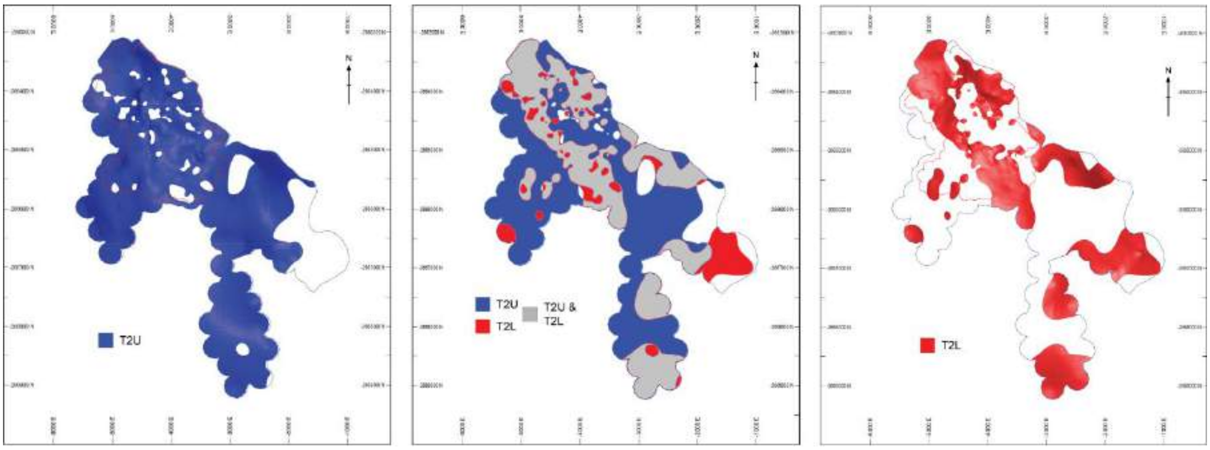
7.2.8.1 T2U and T2L

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 mineralized zone (T2MZ) is defined based on a 1 g/t 3PE+Au cutoff 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 be as much >100 m thick locally.

7.2.8.3 T1MZ and T2MZ Mineralized 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 2g/t 3PE+Au cutoff. 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 mineralization situated just above the basal contact of the T1 should be included with the T2MZ.

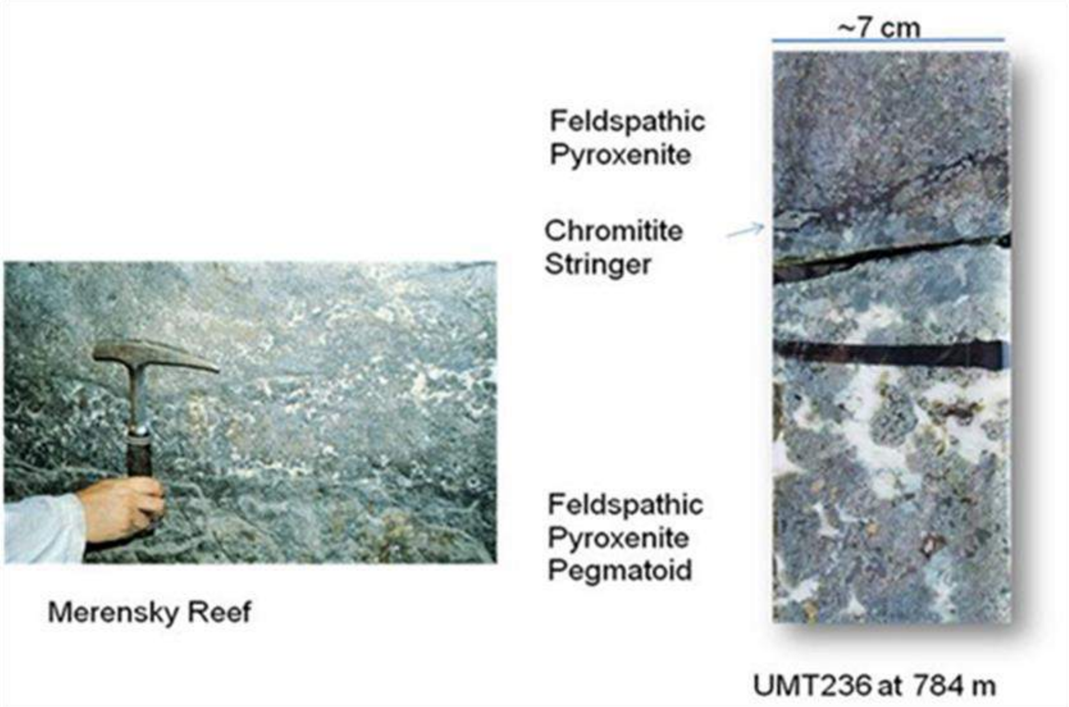
The T1MZ is therefore correlated with the Bastard Reef, and the mineralisation found above the top contact of the T2 pegmatoid, within the feldspathic pyroxenite, is correlated with the M1, as postulated by Davey (1992) and Lea (1996). This implies that the T2 pegmatoid correlates with the M2.

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

Table 7.1 True Thickness of Minzones

Unit	Minzone	Num Drill Holes	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 gray. 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)

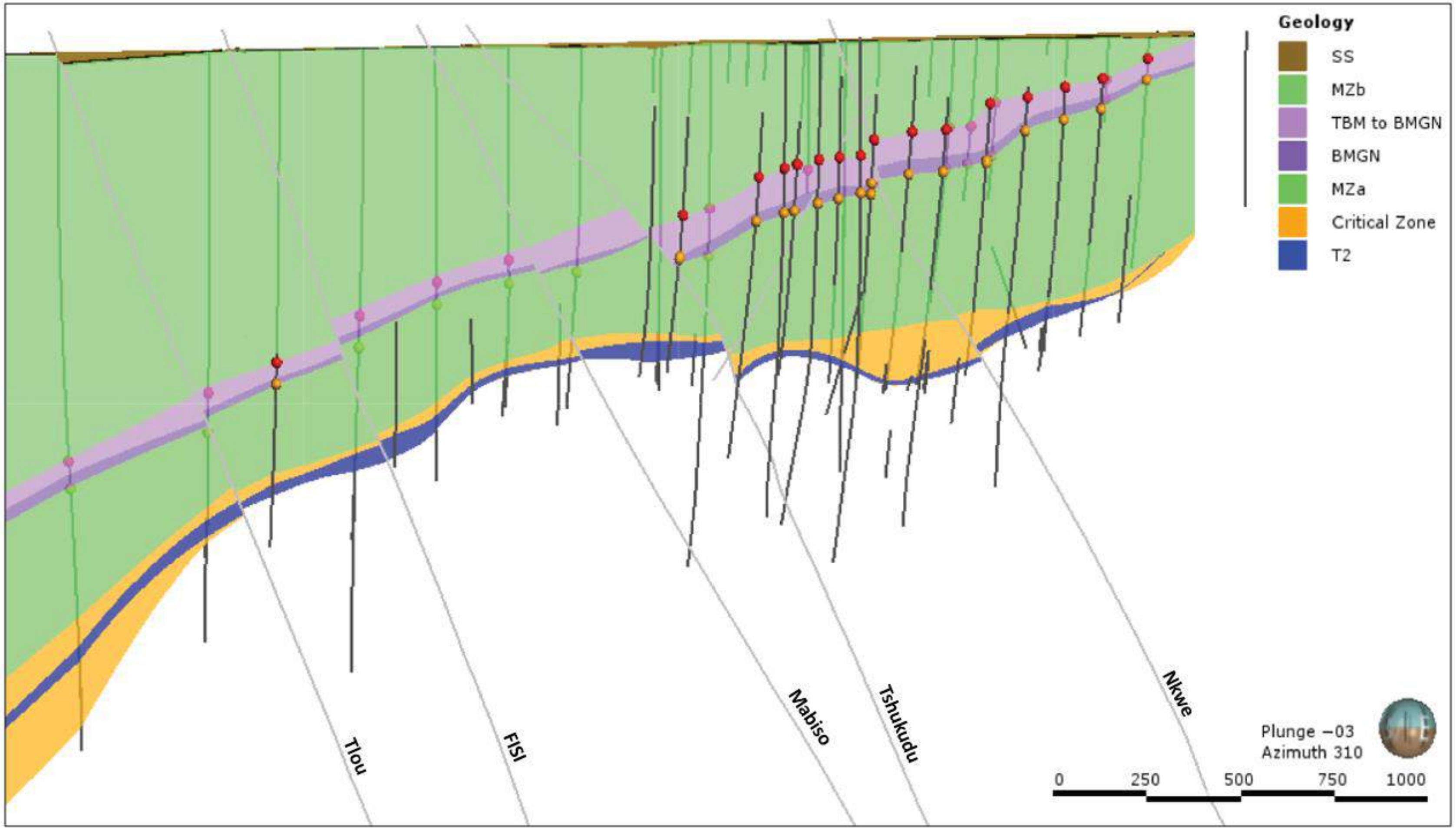
Overlying the magmatic rocks of the CZ is a succession of leucocratic to melanocratic norite and gabbronorite of the MZ. The MZ is the uppermost unit of the RLS observed in drill holes in the Platreef Project area, and MZ forms the hanging wall to the UCZ. Drilling has intersected the MZ up to a vertical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

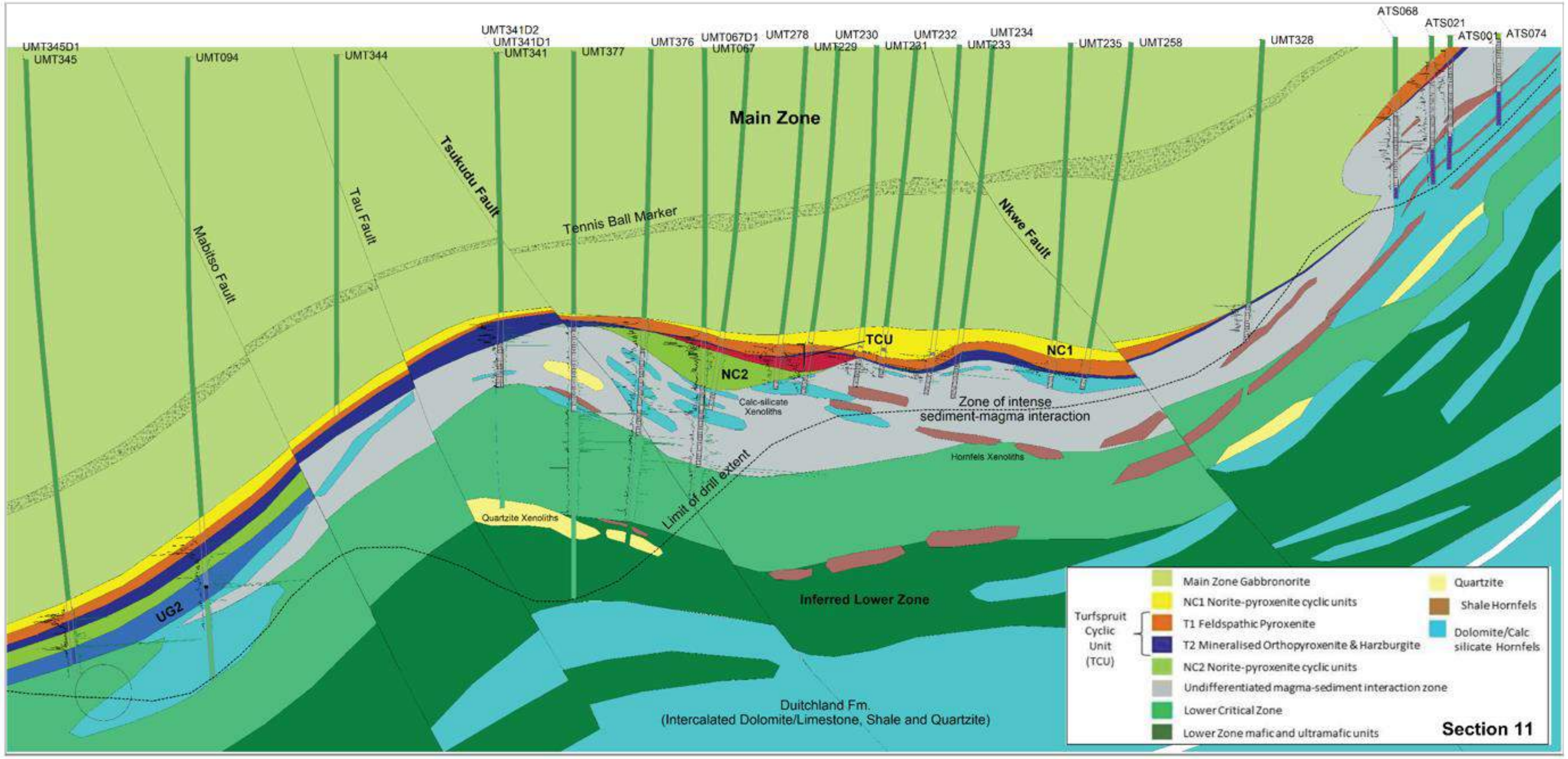
The stratigraphically lower and the up dip magmatic units on the Turfspruit and Macalacaskop farms are characterised by interactions between Bushveld magma and the Transvaal Supergroup host sequence that is composed of diverse dolomite to alumino-silicate sedimentary rocks of the Duitschland Formation. 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 800m 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 mineralized intercepts for the drill holes in Figure 7.12.

Figure 7.12 Diagramatic Dip Section



Courtesy Ivanhoe, 2016

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

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
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

Lengths approximate true thicknesses, as most holes are drilled sub-perpendicular to the Platreef.

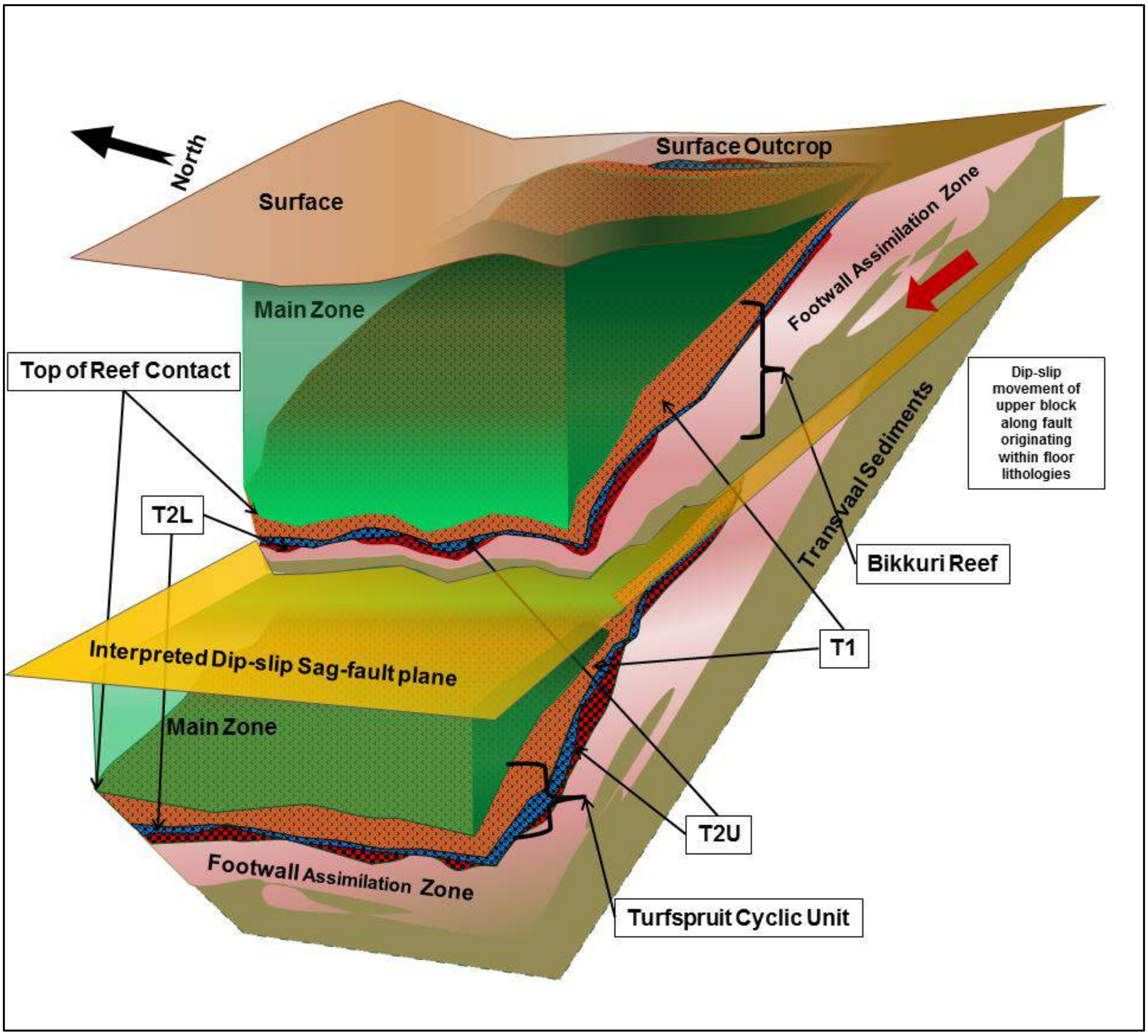
7.4 Bikkuri

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 400m during the 2010–2011 drill program, where normally reef intercepts were expected at 700m depths in that area. A second Bikkuri zone has been interpreted in the southern area of Zone 2 where similar mineralization is located stratigrpaahically 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 Schematis 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 which 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 can be summarized 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.

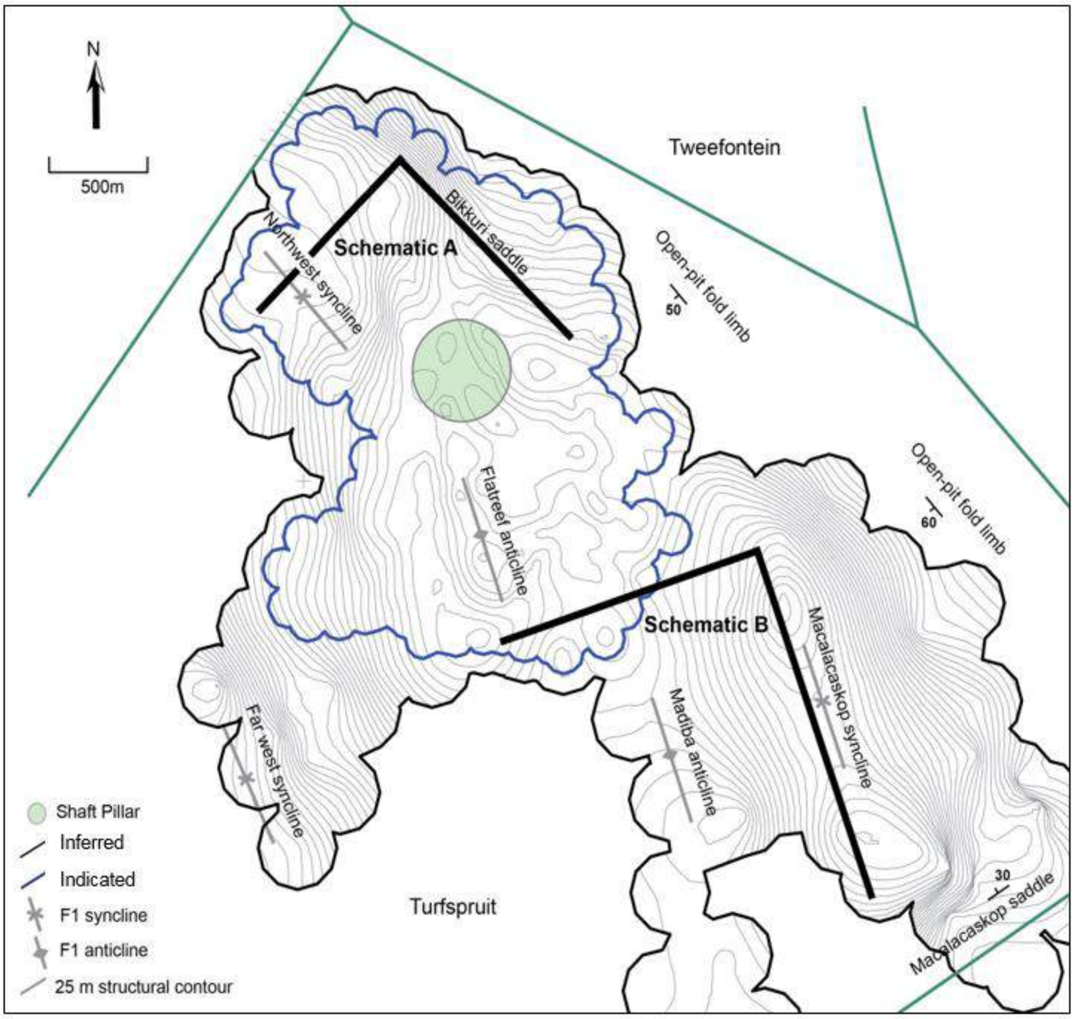
7.5.2 Project Folding

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.

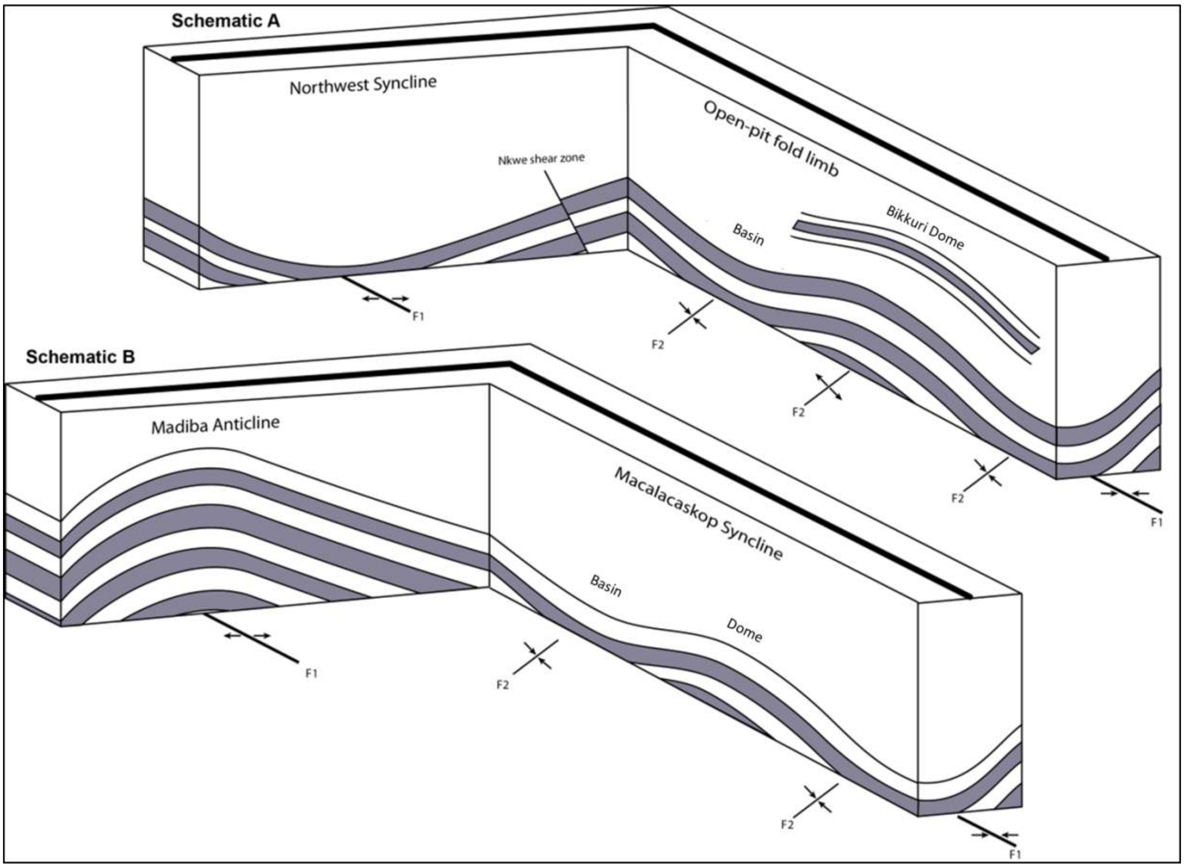
Figure 7.14 shows a Project-scale view of the major (F1) low amplitude open folding and Figure 7.15 is a schematic of the interpreted folding derived from metasedimentary interlayers or "rafts".

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 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 interlayers "rafts". Figure 7.12 hows the location of the fence lines.

7.5.3 Project Fault Structures

An updated structural model for the Project area was constructed using 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 which 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.

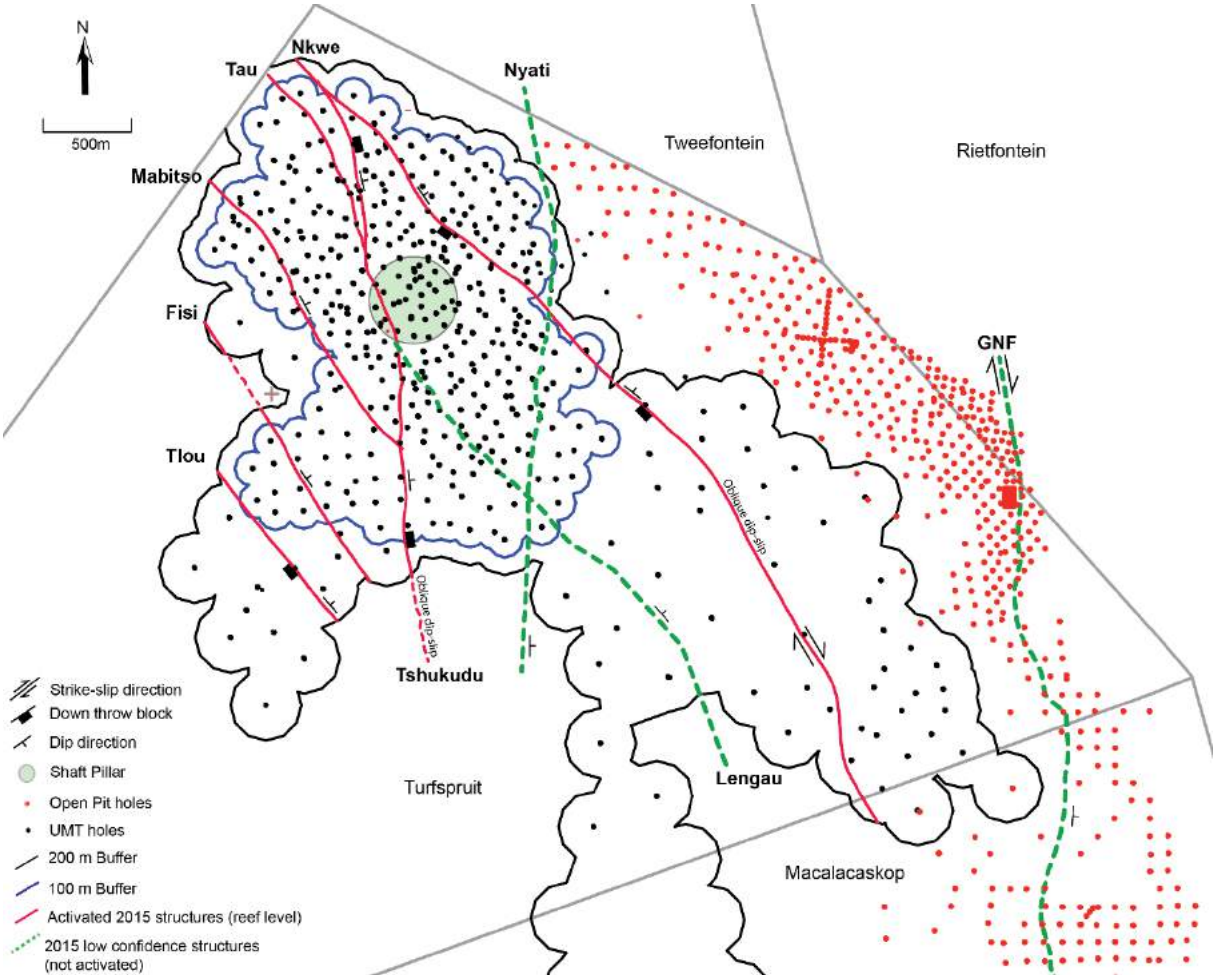
The Tshukudu Fault Zone is a brittle structure that transgresses the central portion of Zone 1. It represents a significant geotechnical hazard and comprises a wide zone of imbricate fracturing in its hanging wall and intense brecciation within the fault zone. Major fall-of-ground hazards can be expected 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 recognized 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 1400 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 1400 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 asre available; dips northeasterly; 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.

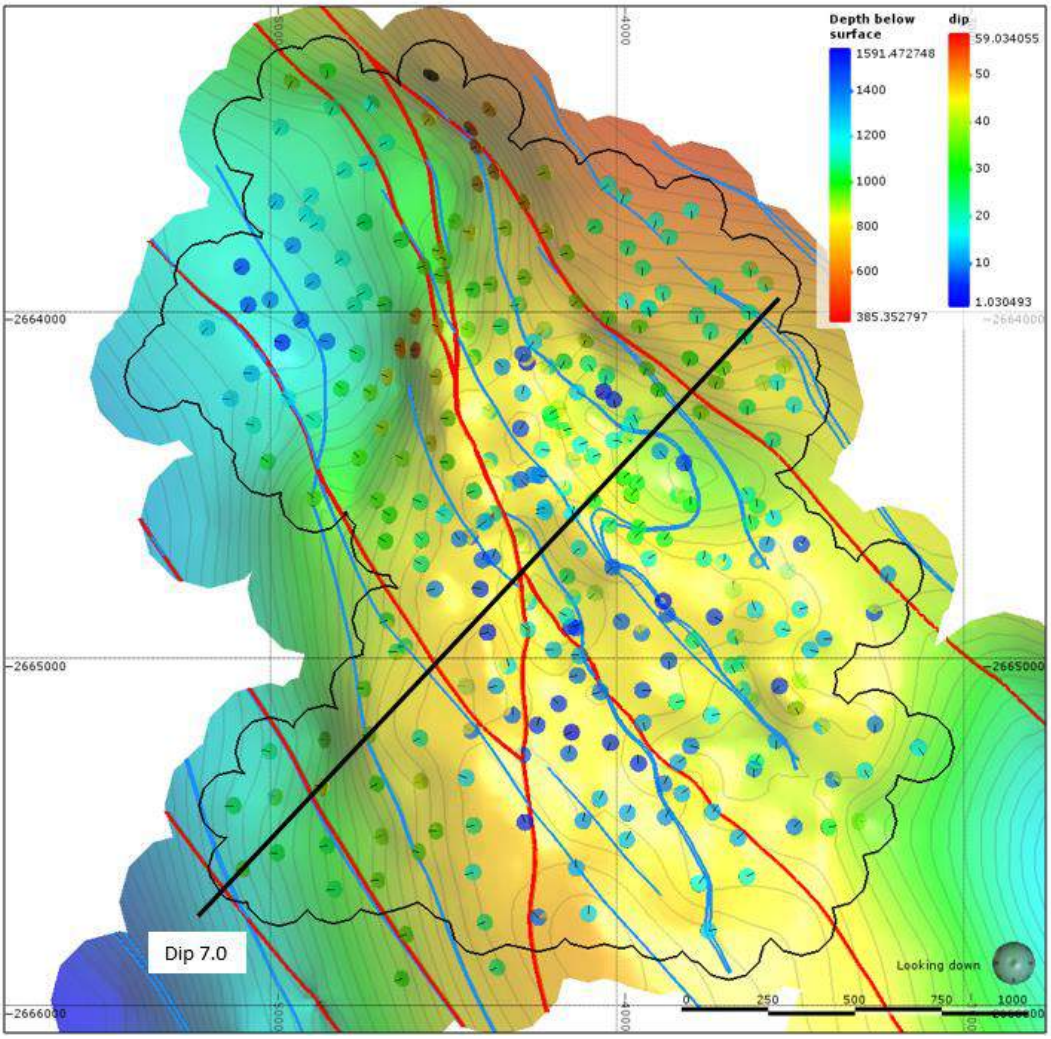
The granite dykes form a dyke swarm cutting through the Project area. The dykes strike northwest and dip steeply (60° to 75°) towards the northeast. The dykes range in thickness from several centimetres to tens of metres. Granite dykes > 2 m thick have been modelled. The granite dykes are commonly orientated 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 will need to take into consideration 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 northwestern portion of the Zone 1. The CPX is a distinct lithological domain that hosts continuous low-grade Ni mineralization with local 3PE+Au mineralisation. The CPX can form a continuous zone of mineralization below the base of the T2MZ. No meta-sedimentary xenoliths have been identified withing 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. Mineralization is typically 1 g/t 3PE+Au, but locally can be 2 - 5 g/t 3PE+Au. Mineralization 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 drill holes 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 suphide (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 mineralization 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 lessr pyrite. Sulphide distribution and concentration vary and ranges 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 assembleages 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.

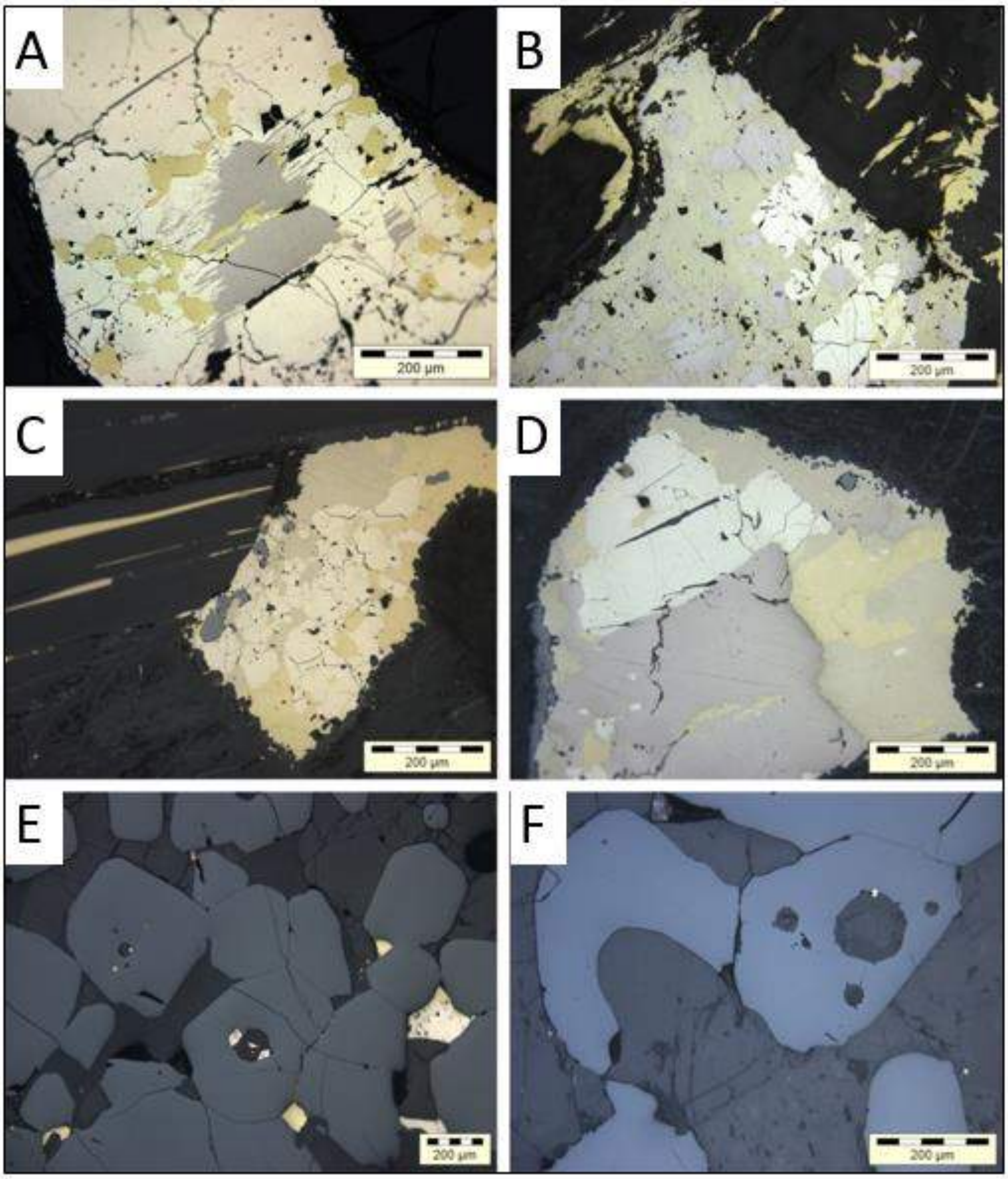
Yudovskaya (2015) determined that clear magmatic assemblages (Merensky-like trends) can be distinguished from an original assemblage influenced and overprinted by secondary effects. The zonation of PGM distribution favours in-situ crystallisation where modal PGE mineral assemblages are controlled by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

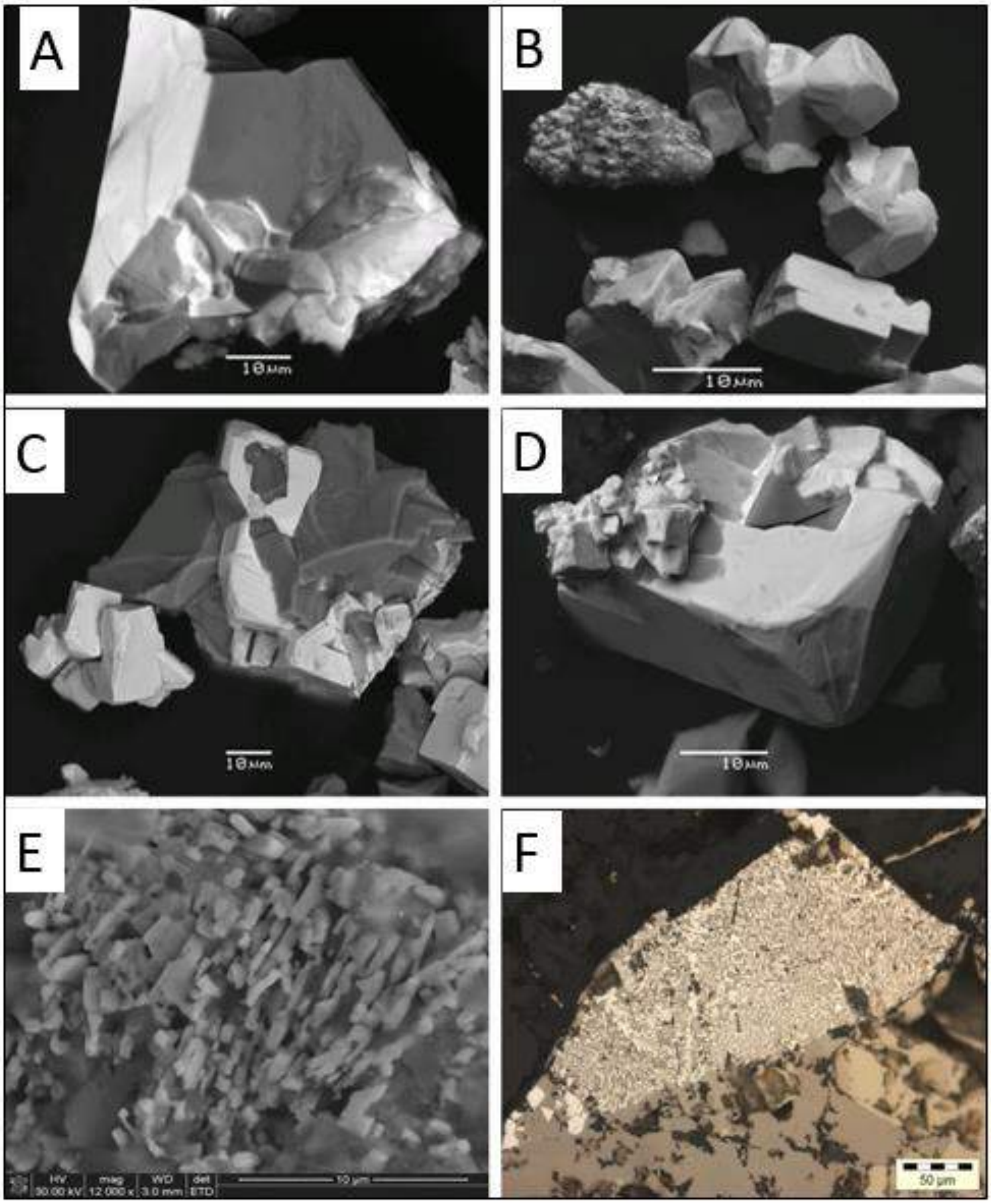
The yellowish mineral is chalcopyrite; the dull purplish mineral is pyrrhotite; the light cream mineral with higher reflectance and some cleavage is pentlandite. Core photographs courtesy Ivanhoe, 2011.

Figure 7.19 Mineral Assemblages Found in the T2U



A-Pyrrhotite remplacement of flame like pentlandite (UMT314 at 1135.5m). B – Relics of pyrrhotite in cubanite. Granular pentlandite is white, chalcopyrite is remobilized 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: RuS_2 intergrown with IrAsS (UMT314 at 1135.54); B: PtS and Pt_3Fe crystals as well as fine-grined 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 sulfides (UMT314 at1135.71); F: the same type as in E eutectoid intergrowth of Pt_3Fe 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 $^{\varepsilon}$Nd values and $^{187}Os/^{188}Os$ 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, drill holes 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.

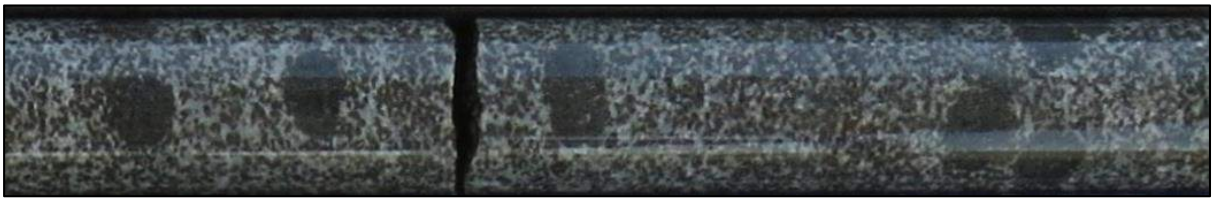
This initial exercise was expanded upon in 2013 to include near-surface information gained from close spaced drilling. The depth of weathering is controlled mainly by rock type, structure and alteration and is most pronounced along the mafic to ultramafic units and along the major fault traces at surface.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

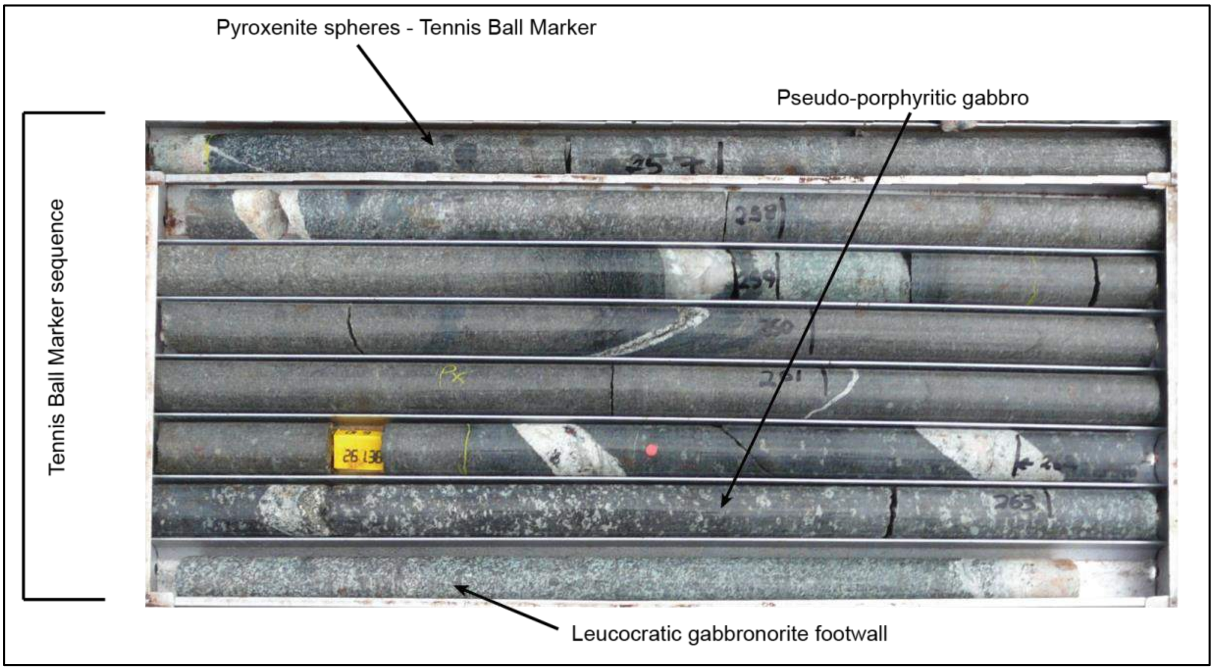
Well-defined anorthosite layering in the upper portion of the Main Zone, and the lower Main Zone layering (TBM and BMGN) are distinct and define the orientation of the layered intrusion (Figure 9.1 and Figure 9.2). This is a major distinction from the orientation of Platreef-type mineralised layers.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

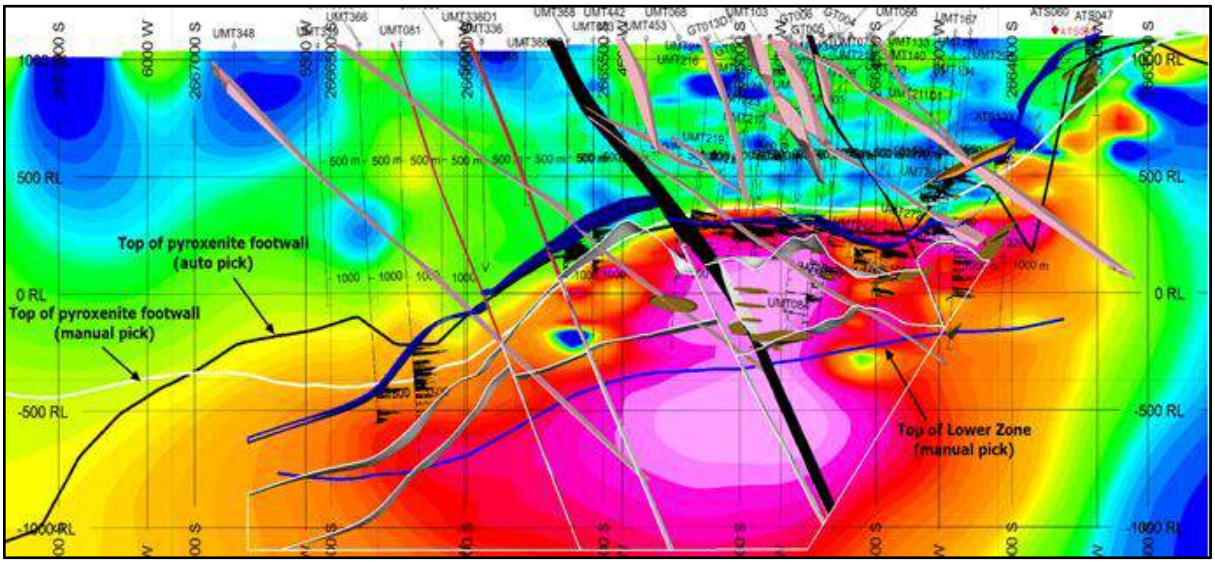
Geophysical survey methods at the Platreef Project have included aeromagnetics, gravity gradiometer and a number of downhole geophysical methods including caliper; self-potential (SP)/point resistance (PR); electrode-array-focussed resistivity (EAL); magnetic susceptibility (Msus); temperature/conductivity; 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 km2 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 3-D seismic survey was run by seismic specialist company CGG, headquartered in Paris, France, in Q4 2013 for the purpose of confirming and enhanceing 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 3-D 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 (drill holes 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 Contrained 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

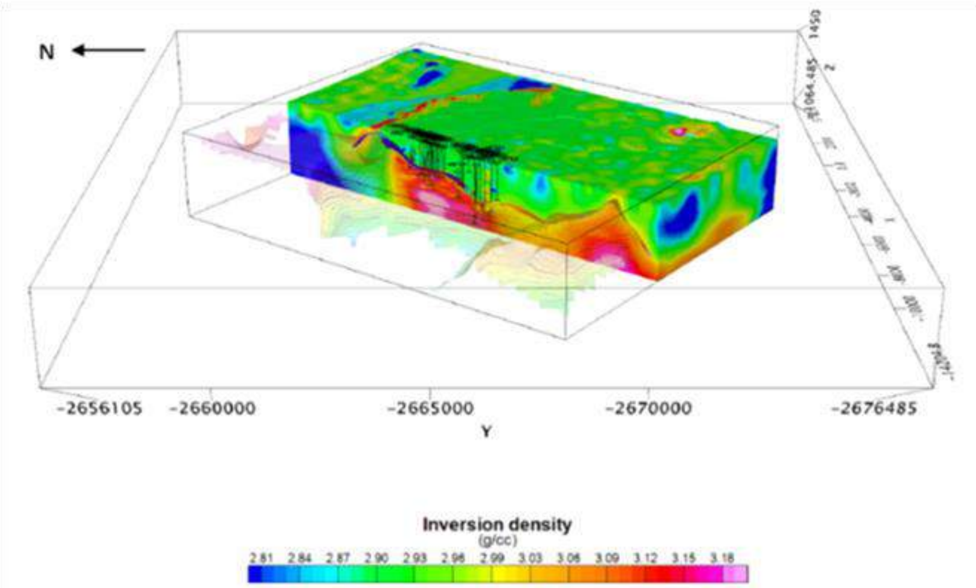
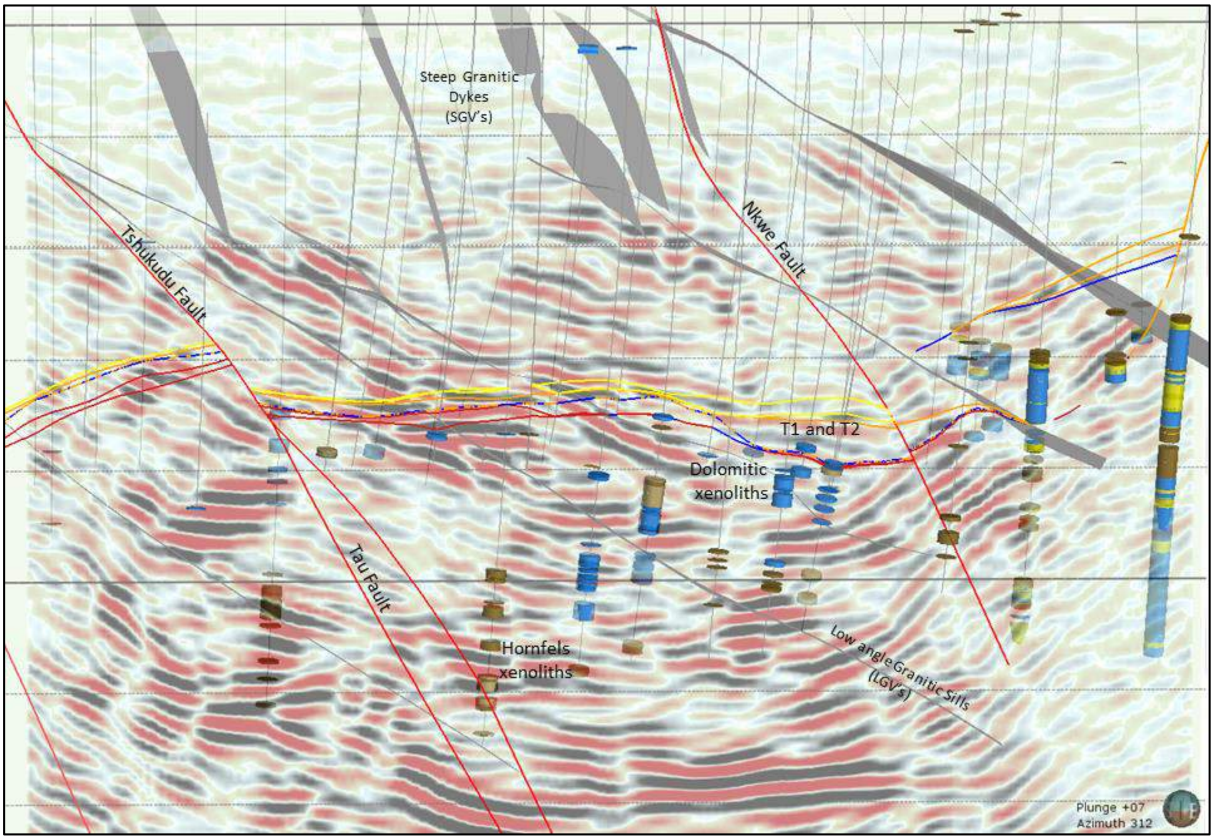


Figure courtesy Ivanhoe, and sourced from Williams (2012). Inversion sliced along a north-east oriented section. Image shows computed depth to >2.97 g/cm3 isosurface which maps the gabbronorite/pyroxenite contact and thereby depicts the approximate structure of the mineralised reef.

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

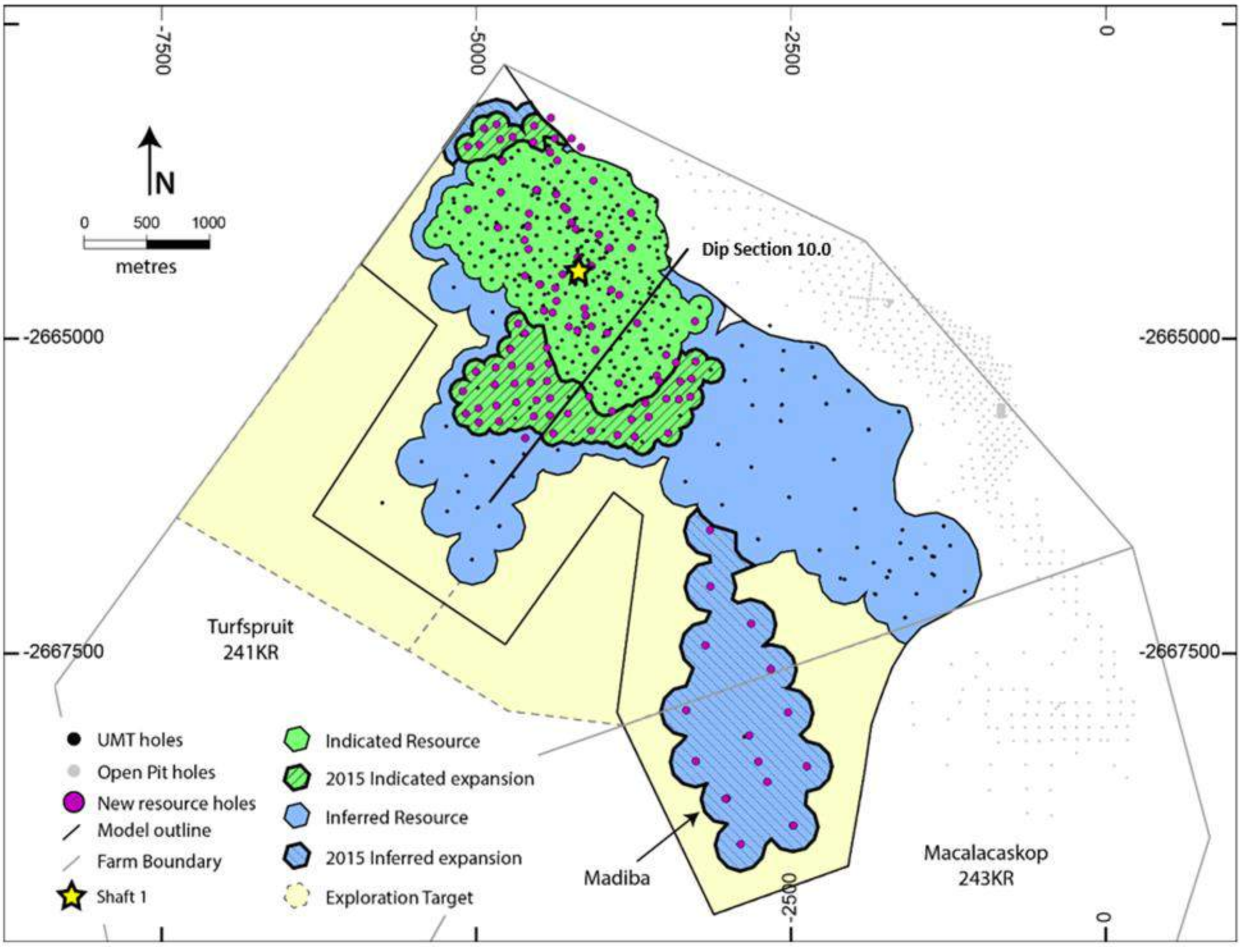
The Platreef mineralisation remains open along strike and down-dip. There is opportunity to expand the extent of known mineralisation with further drilling, down dip (Subsequent limited drilling within the Zone 5 area has shown significant grade values as part of the extension of the Flatreef towards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

9.7 Petrology, Mineralogy, and Research Studies

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.

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



Courtesy Ivanhoe, 2016.

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 drill holes (PUM001 and PUT001) drilled in 2012 which are located in Zone 4. These drill holes are grouped with the open-pit drill holes.

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.

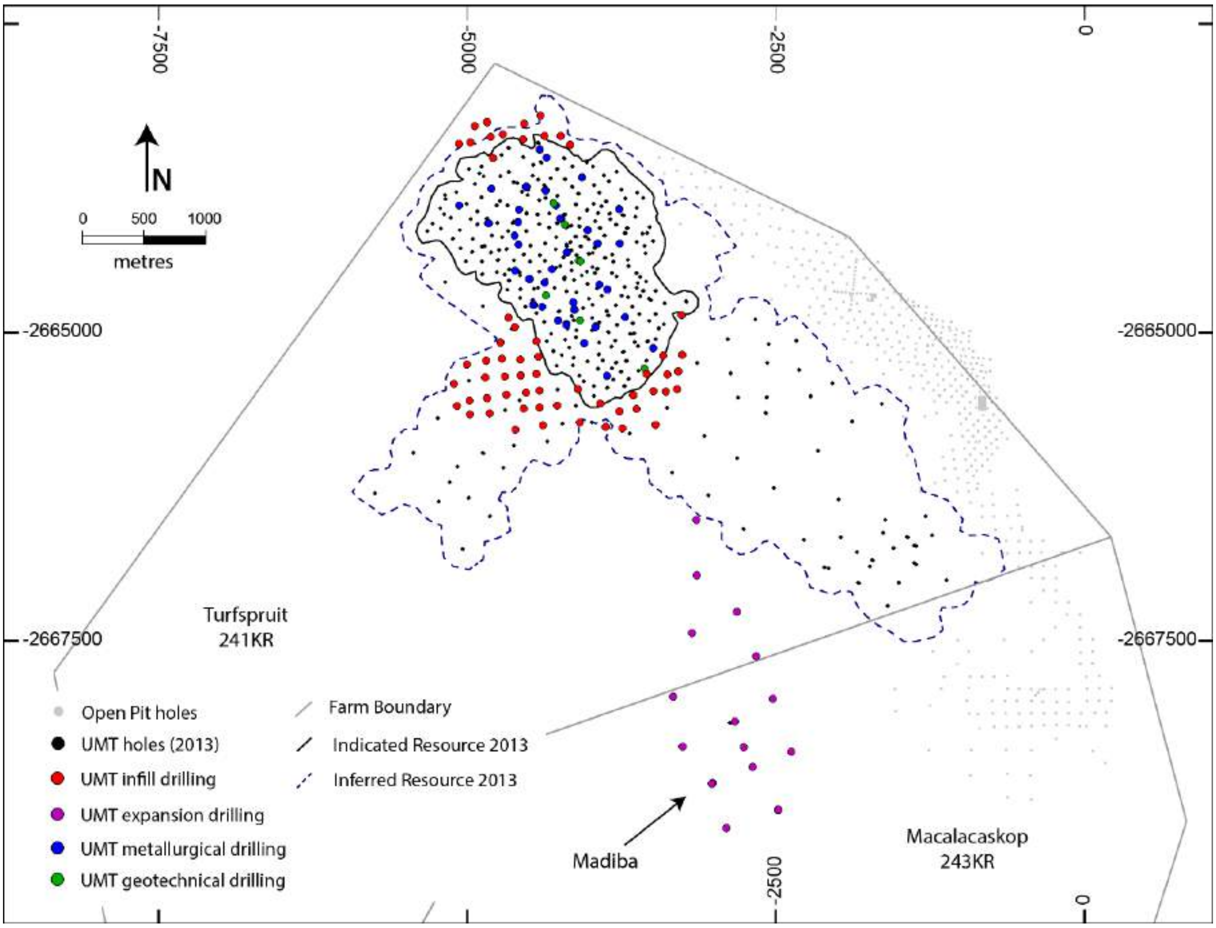
The database includes Phase 2 drilling totalling 574 core drillholes (excluding abandoned and suspended drillholes) totaling 501,638 m completed by 11 February 2015. No drilling for resource estimation purposes has occurred between this date and the Report effective date. Depths for deflections are calculated based on point of 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 purposes (Figure 10.1).

The Phase 2 drilling is summarized 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 defelctions	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 were 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 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 Zone 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 drill hole without downholesurvey

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 memo from site (Ivanplats, 2015) discussing a review of the downhole surveys states that EMS downhole surveys were selected over gyro survey results for 70 drill holes.

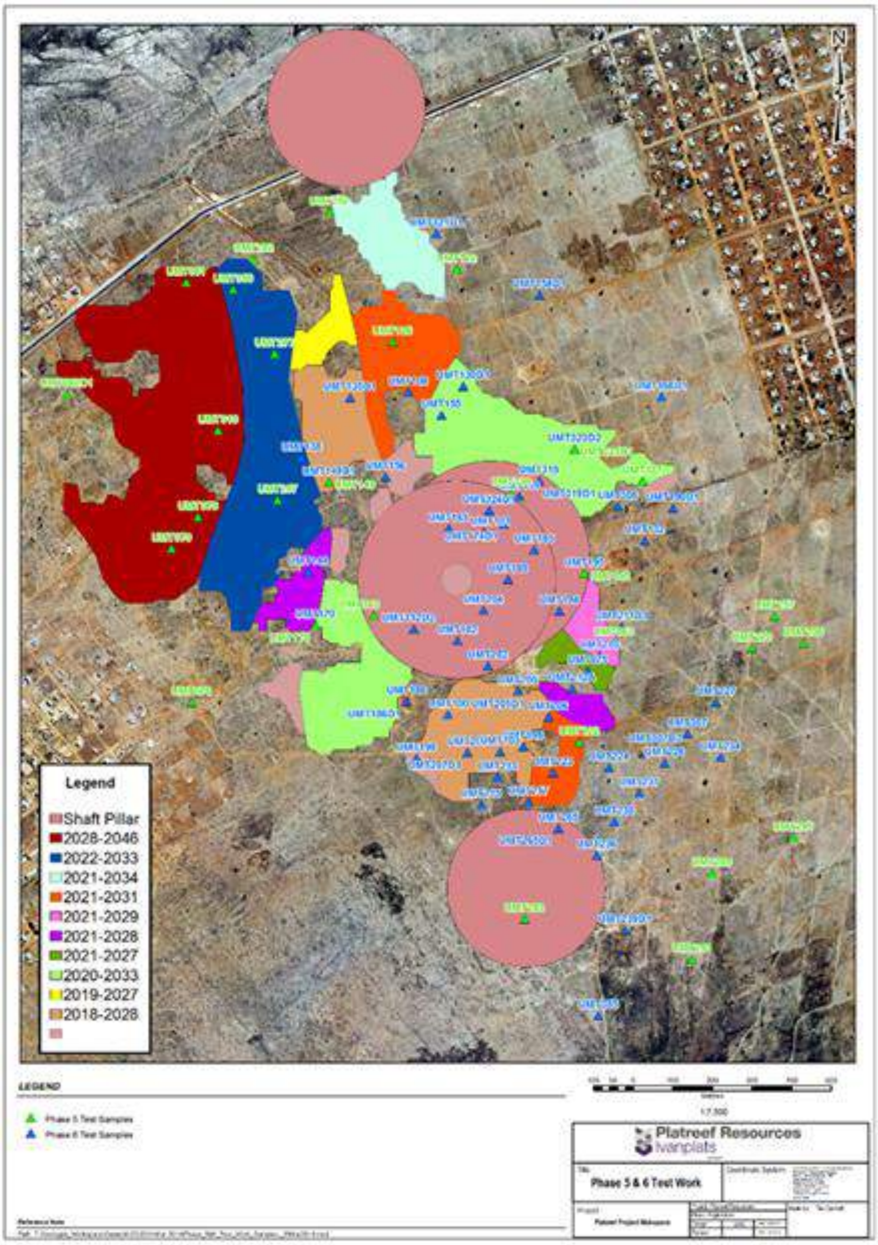
10.7 Metallurgical Drilling

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 (red green markers) laboratories on which the metallurgical testwork is based (Figure 10.2).

Figure 10.2 Metallurgical Drillhole Map



10.8 Summary of Drill Intercepts

Selected drill intercepts showing typical grades and thicknesses of mineralization 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 Elevation	Easting	Northing	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE+Au (g/t)	Cu (%)	Ni (%)
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 testwork and mineralogical analysis for the purposes of a pre-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 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 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

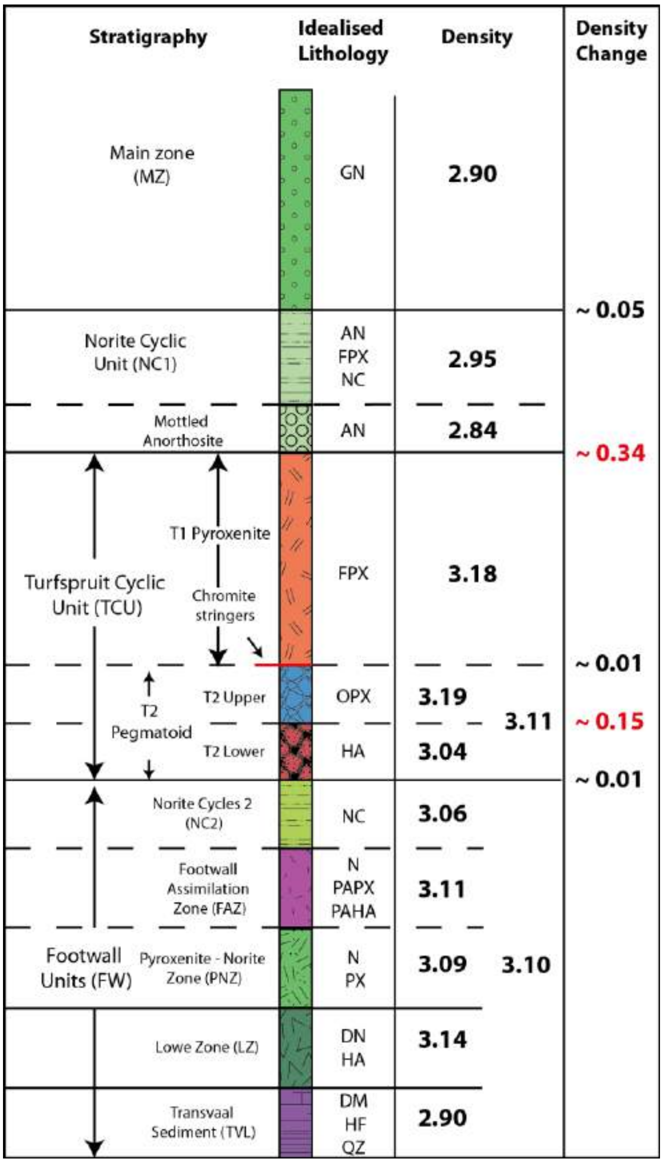
MSTRAT	Description	MSTRAT	MCODE	Number	Average	Minimum	Maximum	C.V.
Bikurri Hanging Wall	Main Zone	BKHW	10	3,772	2.91	2.55	3.30	0.03
	Bikkuri Norite Cycles 1	BKNC1	11	244	2.98	2.62	3.26	0.04
	Bikkuri Mottled Anorthosiite	BKMAN	12	31	2.83	2.63	2.95	0.02
Bikurri	Bikkuri B1	B1	13	264	3.15	2.62	3.34	0.03
	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 Anorthosiite	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 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 through a riffle splitter three times. 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 SO_2 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 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 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 drill holes, 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 drill holes 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 drill holes. Assay data were checked for 5% of the assays from the additional 53 drill holes. 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 drill holes 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 drill holes 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 drill holes 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 drill hole 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 drill holes completed after February 2014. Amec Foster Wheeler compared collar surveys, downhole surveys ang 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 program 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 drill holes completed after October 2014. Amec Foster Wheeler compared collar surveys, downhole surveys ang 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 drill holes against laboratory certificates.

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 precison 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 recommened finer grinding (85-90% passing 75µm).

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 of260 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 from recent drill holes 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 1000 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 testwork at Mintek.

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

The principal finding from this initial submission was a high bias of 10 to 15% in Mintek's Ni results; this was shown both by the blind insertions of AMIS standards and by comparison with the original Ultra Trace results on a split of the same pulp. In March 2013, 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 testwork, together with Mintek's assay results for those materials. These samples included various kinds of tails and concentrate samples. A few samples of Mintek's SRMs were obtained as well. These samples were submitted to Ultra Trace for base metal analysis in April 2013. The Ultra Trace assay report in May 2013 confirmed acceptable accuracy on all elements except Ni, which showed Mintek metallurgical assay results have high bias exceeding 10%. A regression equation for adjustment of Mintek Ni assays was recommended by Long (2013b):

$$\text{Adjusted Ni} = 0.87{*}\text{Mintek ICP Ni} + 207\text{ ppm.}$$

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

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

12.11 Comments on Section 12

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 QAQC 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 comleted an audit of the UMT drill holes 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 Previous Metallurgical Testwork

Various metallurgical testwork campaigns have been conducted on the Platreef resource 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 2013 a series of metallurgical testwork campaigns were carried out on the Platreef mineralised material as detailed below.

13.1.1 Phase 1: SGS Booysens 2010

Testing at SGS Booysens in 2010 was undertaken on three (3) composite samples from the Platreef deposit namely;

- TLZ-PX (Top Loaded Zone – Pyroxenite)
- BLZ-PX (Bottom Loaded Zone – Pyroxenite)
- TLZ-SP (Top Loaded Zone – Serpentinite)

The testwork program included grind optimisation testing, reagent scouting tests and locked cycle tests.

The locked cycle tests indicated that a concentrate grade of 58 g/t – 74 g/t could be achieved.

The recovery of base metal sulphides at a coarse grind followed by a regrind step to allow for improved PGE recovery at the finer grind was thought to be the optimal processing route. This circuit is known as an MF2 configuration and is common in South African PGE concentrators.

13.1.2 Phase 2: Xstrata Process Support 2011/2012

In 2011 a metallurgical testwork campaign was undertaken at Xstrata Process Support (XPS) in Ontario, Canada. The first Phase of XPS testing was based on stratigraphic interpretation consisting of five geometallurgical units from the UCZ. This Phase of testing included grind optimisation testing followed by baseline rougher flotation tests.

An optimised two stage milling and two stage flotation flow sheet, MF2 circuit in South African processing terms, was developed based on results from grind optimisation testing and reagent dosage testing.

The development testing 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%.

13.1.3 Phase 3: SGS Lakefield 2012

Following on from the identification of the T1, T2U and T2L geometallurgical units during the 2012 geological reassessment and subsequent to the Phase 2 XPS testing, further flow sheet development testing and reagent scouting tests were conducted at SGS Lakefield in Phase 3- under the management of Ivanplats and Amec Foster Wheeler.

During this Phase of testing on the Phase II Master Composite (MC II) sample, a reagent suite that included oxalic acid and thiourea addition with conditioning in the mill prior to flotation indicated that a PGE recovery of 83% could be achieved at a concentrate PGE (3E) grade of 123 g/t. 440 C high chrome media was used as grinding media in the laboratory mill. During this testing combined rougher concentrate was treated in a simple 3 stage cleaner circuit.

Testing on the MC II sample indicated that a reagent suite containing oxalic acid and thiourea was suited to the Platreef ore type and formed the basis for further flow sheet development work.

13.1.4 Phase 4: Mintek March 2012 – January 2013

Phase 4 testwork was conducted at Mintek from March 2012 – January 2013 under the management of Ivanplats and Amec Foster Wheeler. The testwork was conducted on drill core samples on the mid-2012 geological reassessment namely, T1, T2U and T2L.

The testwork conducted in Phase 4 included rougher kinetic testing, which indicated that the optimum mill grind was 80% passing 75 µm. Flotation testing was conducted on a composite sample containing T1, T2U and T2L to confirm the results obtained at SGS in Phase 3.

The findings of the Mintek Phase 4 testing were 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 3PE+Au grade of 120 g/t during locked cycle testing. 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 those using carbon steel media. During this testing combined rougher concentrate was treated in a simple 3 stage cleaner circuit.

Locked cycle tests in Phase 4 had an oxalic acid and thiourea in-mill conditioning time of 109 minutes which would not be possible to replicate for a full scale milling operation.

13.1.5 Phase 5: SGS Lakefield 2013

Phase 5 testwork was conducted at SGS Lakefield under the management of Ivanplats. Comminution testwork was conducted at SGS Lakefield on Phase 5 quarter core drill samples from the Platreef deposit. Work was done on the 3 mineralised zone geomet units as well as 4 footwall composites.

The ball mill work indices were found to be in the range 20.0 kWh/t – 22.3 kWh/t; which indicates that the Platreef material can be classified as hard to very hard at the size fraction tested with the bond wall work index test (<3.35 mm).

Bench scale flotation testing, composites were made up of various quarter core NQ intersections from diamond drillholes. The composite samples represented the three geometallurgical units as well as a bulk composite of an indicated mine blend.

The testwork conducted at SGS in Phase 5 included rougher kinetic testing, which indicated that the optimum mill grind was 80% passing 75 µm.

In addition to the rougher kinetic testing, open circuit batch cleaner tests were performed during the flow sheet development phase. The results of open circuit batch cleaner testing with 5 minutes of in-mill conditioning time at SGS Lakefield using 440C high chrome grinding media indicated the following:

- PGE recovery to final concentrate was in the range 69.5% - 74.7% for a simple three stage cleaning circuit treating a combined bulk rougher concentrate.
- The inclusion of 30 minutes of post mill conditioning at 60% solids prior to flotation resulted in improved PGE recovery and upgrade in the cleaner circuit.
- Additional oxalic acid and thiourea addition to the post mill conditioning stage did not improve PGE recovery or upgrade in the cleaner circuit.
- 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 improved PGE, Cu and Ni recovery.

Four (4) locked cycle tests were conducted using a flow sheet that included oxalic acid and thiourea addition with 5 minutes conditioning in the mill prior to flotation. One test was conducted for a simple 3 stage cleaner circuit treating a combined rougher concentrate. Three tests were conducted using a split cleaner configuration, treating the fast, medium and slow floating PGE fractions separately.

- The split cleaner configuration gave improved metallurgical performance when compared to the tests conducted using a simple 3 stage cleaner and indicated that a PGE recovery of 85% could be achieved at a concentrate 3PE+Au grade of 85 g/t.

During Phase 5, Rheological tests were performed by SGS, Lakefield, on a combined tailings sample from initial development testing using a reagent suite containing oxalic acid and thiourea.

The results of this testing were as follows:

- The optimised conditions from dynamic settling tests, predicted 0.10 and 0.014 m2/t/day thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively.
- Based on rheology testing, the critical solids density (CSD) of the combined tails underflow sample was approximately 64% solids (w/w).
- Vacuum filtration tests produced filter cake with a thickness that ranged from 12–35 mm. The dry solids capacity ranged from 247 to 1365 kg/ m2.h, with filter cake residual moisture content of 13.5% to 21.5 % (w/w).

13.1.6 Phase 6A: Mintek March 2013 – October 2013

The testwork conducted at Mintek in Phase 6A was done under the management of Ivanplats and DRA. Focus was on bench scale flotation testing for the PEA flow sheet development.

A series of open circuit batch cleaner tests were performed during the flow sheet development Phase. The results of these open circuit batch cleaner tests with 5 minutes of in-mill conditioning time were largely in agreement with the findings of the Phase 5 SGS testwork which ran concurrently. The Phase 6A testing indicated the following:

- Using stainless steel grinding media, the PGE recovery was in the range of 72.8% – 77.4% for a simple three stage cleaning circuit treating a combined bulk rougher concentrate.
- The use of carbon steel grinding media negatively impacted PGE recovery with a drop to 67.7% recovery in final concentrate. In addition to this a poor grade-recovery profile for PGEs and copper was observed in the cleaner circuit.
- The inclusion of post mill conditioning (60 min at 30% solids or 30 min at 60% solids) solids prior to flotation resulted in improved PGE recovery and upgrade in the cleaner circuit.
- 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 improved PGE, Cu and Ni recovery as compared to tests treating a bulk rougher concentrate in a 3 stage cleaner circuit.

A single locked cycle test was conducted using a flow sheet that included oxalic acid and thiourea addition with 5 minutes conditioning in the mill prior to flotation with a split cleaner configuration, treating the fast, medium and slow floating PGE fractions separately. This locked cycle test indicated that a PGE recovery of 87.8% could be achieved at a concentrate PGE (3E+Au) grade of 93g/t.

As part of the Phase 6A testwork programme, ALS Minerals in Johannesburg conducted modal mineralogy and PGE investigations on 7 concentrate and tails samples generated during open circuit flotation tests using a reagent suite containing oxalic acid and thiourea.

- 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%). PGE losses to the rougher tailings were found to be as a result of poor liberation, with >86% locked primarily with silicate gangue. Liberated PGE's in the rougher tailings were all in the <10 µm size fraction.
- The PGE in the cleaner tailings were comprised predominantly of PGE tellurides (48%) with lesser PGE arsenides (25%), gold (17%), alloys (5%), and PGE sulphides (4%). PGE losses to the cleaner tailings were as a result of poor liberation, with >60% locked primarily with silicate gangue. Liberated PGE's in the cleaner tailings were all in the <20 µm size fraction. The >20 µm particles in tailings were found to be mostly locked with middlings.

- The HG concentrate produced from split cleaner testing was comprised of 66% (w/w) base metal sulphides, 31% (w/w) silicates and ~2%w/w oxides. The base metal sulphide fraction in the HG concentrate was comprised predominantly of pentlandite (53% w/w) with lesser amounts of chalcopyrite (35% w/w) and minor pyrrhotite (7% w/w) and pyrite (5% w/w). PGE in HG concentrate was comprised predominantly of PGE tellurides (86%) with lesser PGE arsenides (8%) and minor PGE alloys, electrum and sulphides. Maslovite (56%) and moncheite (23%) were the main PGE minerals.
- The medium grade (MG) concentrate produced from split cleaner testing was comprised of 51% (w/w) silicates, 46% (w/w) base metal sulphides, and ~2%w/w oxides. The base metal sulphide fraction in the MG concentrate was comprised predominantly of pyrrhotite (65% w/w) with pentlandite (23% w/w), pyrite (6% w/w) and chalcopyrite (6% w/w) making up the balance. PGE in MG concentrate was comprised predominantly of PGE tellurides (47%) with the balance comprised of PGE electrum (33%), PGE arsenides (13%), PGE sulphides (4%) and PGE alloys (3% w/w). Electrum (33%) and maslovite (26%) were the main PGE minerals. A lesser amount of sperrylite (13%), kotulskite (12%) and moncheite (7%) made up the balance.

13.2 Current Metallurgical Testwork

This section summarises the metallurgical testwork carried out between November 2013 and October 2014 at Mintek in Phase 6B under the management of Ivanplats and DRA.

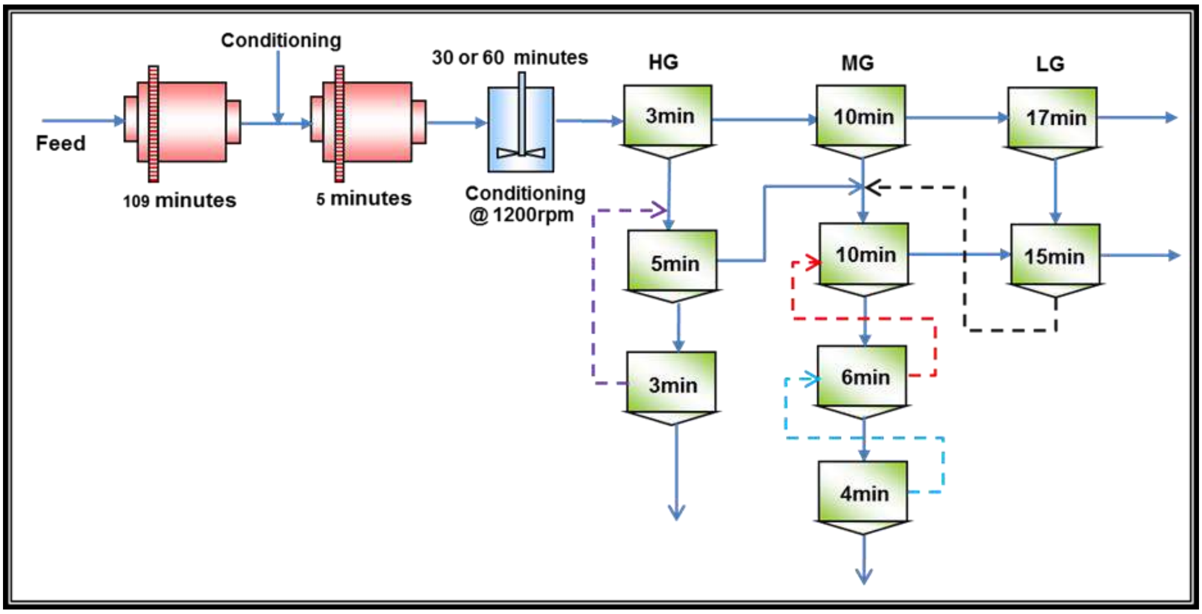
This testwork was used in conjunction with testwork from Phase 5 and Phase 6A, to develop the pre-feasibility study flow sheet. The Phase 6B testwork at Mintek was conducted on drill core samples representing the geometallurgical units T1, T2U, T2L and footwall. Development testwork is ongoing and variability and mini pilot plant work is planned for Q1'15.

The PEA flow sheet included oxalic acid and thiourea addition with 5 minutes conditioning in the mill prior to flotation. The cleaner flotation circuit was based on a split cleaner configuration, treating the fast, medium and slow floating PGE fractions separately. The flow sheet used as the basis for the PEA is presented in Figure 13.1.

Phase 6B testwork carried out between November 2013 and October 2014 was aimed at optimizing the PEA flow sheet and reagent suite. The testwork evaluated the effect of conditioning, depressant addition, alternate reagent suites and circuit configurations on metallurgical performance as compared to the results of locked cycle testing published as part of the PEA in Phase 5 and Phase 6A.

In addition to the bench scale flotation testwork conducted in Phase 6B, comminution variability testwork was conducted on drill core samples representing the first 5 years of mining.

Figure 13.1 Platreef PEA Flow sheet



13.2.1 Comminution Testwork

Year 1-5 comminution variability testwork was conducted at Mintek on approximately 1346kg of HQ drill core sample from the Platreef deposit which was delivered to Mintek in November 2013.

The Phase 6 Mintek Year 1-5 comminution variability testwork indicated the following:

- The UCS test results indicate UCS values in the range 44.1 Mpa – 251 Mpa with an 85th percentile of 181 Mpa and an average value of 146.8 Mpa. Based on the UCS data the ore tested can be classified as medium to hard.
- The impact test results indicate crusher work index in the range 7.2 kWh/t – 31.6 kWh/t with an 85th percentile of 18.2 kwh/t and an average value of 14.2 kWh/t. Based on the crusher work index data the ore tested can be classified as soft to medium hard.
- Abrasion indices were found to be in the range 0.060 g – 0.555 g with an 85th percentile of 0.419 g and an average value of 0.350 g.
- Bond rod work indices were in the range 14.5 kWh/t – 20.9 kWh/t with an 85th percentile of 19.0 kWh/t. Based on the Bond rod work indices the ore tested can be classified as medium to hard with respect to rod milling.
- Bond ball work indices at a closing screen size of 106 µm were in the range 19.0 kWh/t – 23.7 kWh/t with an 85th percentile of 22.9 Wh/t. Based on the Bond ball work indices the ore tested can be classified as very hard with respect to ball milling. Evaluation of the Bond ball work index test data indicated that the Bond ball work indices at a closing screen size of 106 µm resulted in an 80% passing product size of approximately 86 µm. This further indicated that the ore is particularly hard to mill to a fine size fraction of 75 µm.

Grindmill testwork was conducted in order to determine specific rates of breakage for the various ore types. Grindmill test data was simulated in conjunction with Bond rod and Bond ball mill tests in order to obtain an estimate of the ore specific breakage rate relative to top size.

13.2.2 MF1 Circuit Optimisation Testing for an Oxalic Acid and Thiourea Reagent Suite

As part of this testing, a series of tests were conducted to evaluate the effect of oxalic acid and thiourea addition, collector type, Eh, depressant strength and aeration on flotation performance. The head assays for the samples used during this testing is presented in Table 13.1.

Table 13.1 Phase 6 Mintek Sample Head Assays for MF1 Oxalic Acid and Thiourea Optimisation Testing

Sample ID	Head Grade							
	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3E+Au (g/t)	Cu (%)	Ni (%)	S (%)
T1	2.52	2.26	0.12	0.35	5.25	0.20	0.40	0.98
T2U	1.91	1.96	0.09	0.28	4.24	0.24	0.44	1.26
T2L	2.05	2.24	0.09	0.30	4.68	0.21	0.43	0.94
15%T1 : 42.5%T2U : 42.5%T2L	2.00	2.10	0.10	0.30	4.50	0.22	0.45	1.10

The following test conditions were evaluated for a split cleaner circuit configuration:

- Re-baseline of oxalic acid and thiourea PEA flow sheet test conditions with reference to Test 47.
- Eh control using peroxide/aeration to maintain a positive Eh during rougher flotation.
- Collector variations (SIPX/SEX).
- Depressant dosage at 50% strength in an attempt to improve froth stability and reduce skinning effects.
- Testing with no addition of oxalic acid and thiourea.

The flow sheets used for these tests is presented in Figure 13.2 and the results of these tests are presented in Table 13.2.

Figure 13.2 Flow Sheet Used for MF1 Circuit Optimisation Testing for an Oxalic Acid and Thiourea Reagent Suite

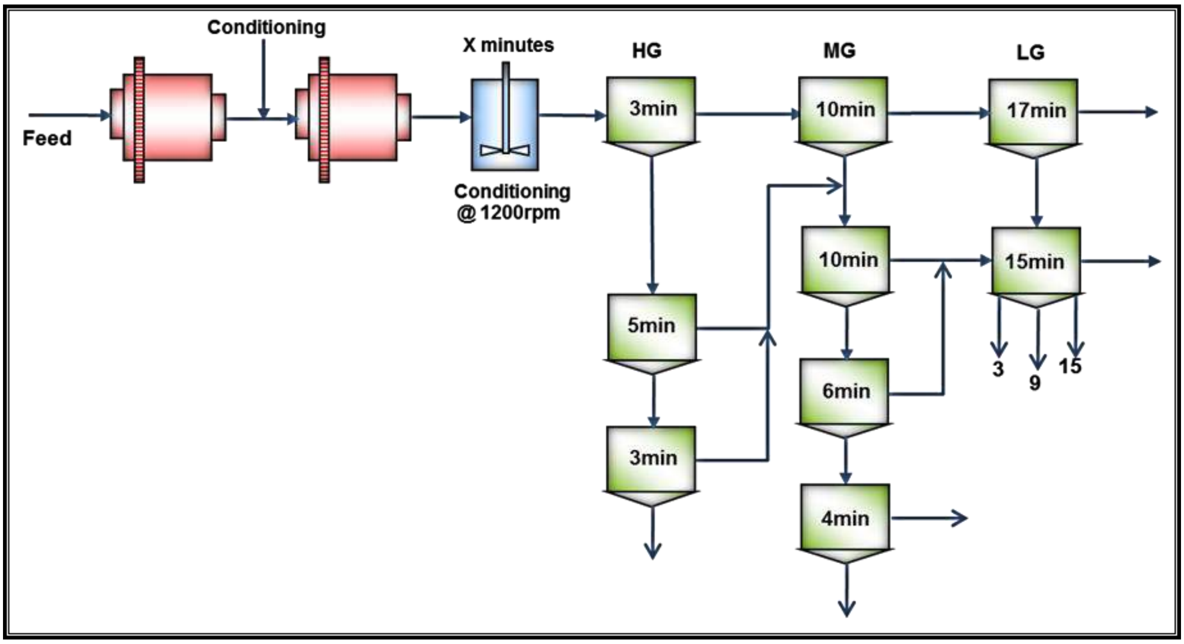


Table 13.2 Results of MF1 Optimisation Tests Using an Oxalic Acid and Thiourea Reagent Suite

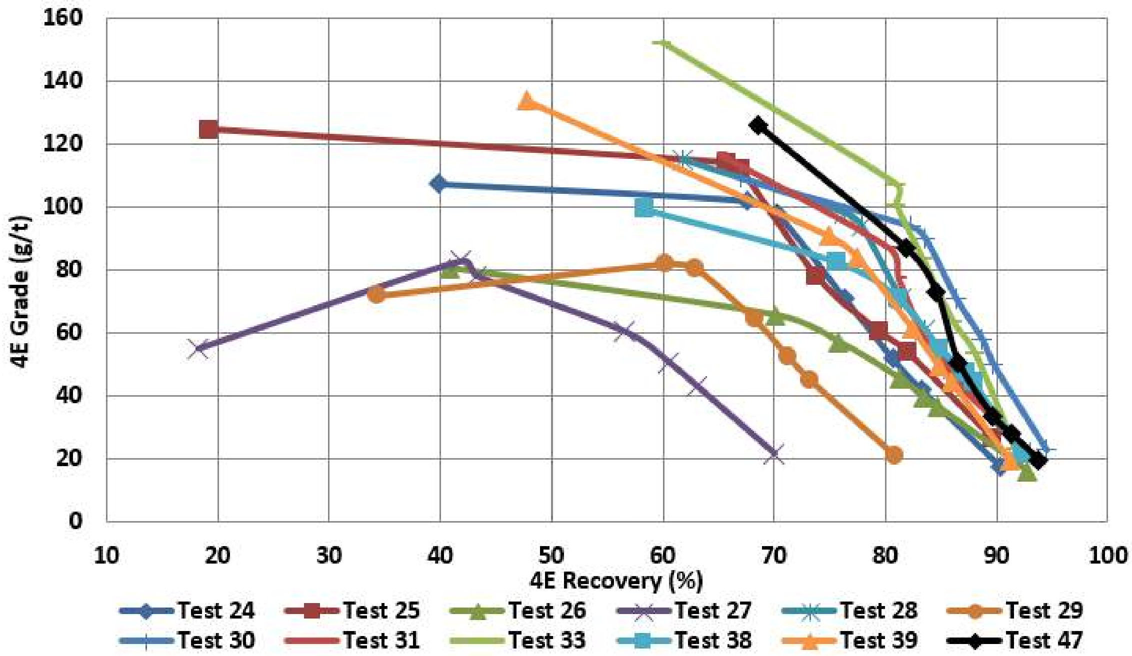
Test No.	Products	Mass (%)	Grade					Recovery				
			3E+Au (g/t)	Cu (%)	Ni (%)	Fe (%)	S (%)	3E+Au (%)	Cu (%)	Ni (%)	Fe (%)	S (%)
47 Baseline	Comb. Conc	4.38	86.72	4.24	8.03	27.84	20.92	81.91	87.37	71.05	12.73	81.87
27	Comb. Conc	2.25	82.77	5.27	8.55	21.58	19.47	41.86	56.06	43.93	5.08	38.67
28	Comb. Conc	3.35	98.03	5.59	8.27	32.39	25.58	76.37	86.00	62.53	11.45	76.01
29	Comb. Conc	3.25	81.93	5.25	7.31	32.39	24.22	60.19	82.30	53.93	11.38	72.80
30	Comb. Conc	3.92	93.95	4.78	7.89	31.72	22.38	82.21	87.63	67.49	12.73	79.74
31	Comb. Conc	4.11	87.10	4.46	7.31	30.09	22.41	80.30	87.22	65.57	12.46	79.15
33	Comb. Conc	3.49	107.05	4.76	7.92	10.15	22.63	81.00	85.60	62.43	10.15	75.06
24	Comb. Conc	2.76	102.05	6.23	9.10	21.96	20.04	67.54	81.61	55.22	6.09	47.21
25	Comb. Conc	2.47	114.10	6.60	10.44	32.22	25.48	65.73	78.01	55.77	8.09	53.03
26	Comb. Conc	4.48	65.67	4.08	5.45	16.24	11.79	70.16	82.95	56.34	7.30	47.28
38	Comb. Conc	3.96	82.38	5.37	6.36	20.33	15.91	75.63	88.98	58.88	8.31	61.61
39	Comb. Conc	3.54	90.72	4.84	6.05	22.64	17.88	74.95	82.15	57.64	8.17	61.15

The results of the MF1 circuit optimisation tests using an oxalic acid and thiourea reagent suite indicated the following:

- Addition of peroxide to maintain an Eh > 150 mV in test 27 resulted in reduced PGE, Cu, Ni and S recoveries as compared to the baseline test 47.
- In test 28 peroxide addition to the roughers was reduced so as to target an Eh>1, this resulted in improved PGE, Cu, Ni and S recovery as compared to test 27. PGE recovery to final concentrate was 76.4% as compared to 83.6% for baseline test 47.
- In test 29 where no SIPX collector was added the PGE, Cu, Ni and S recoveries were significantly reduced as compared to the baseline test 47.
- The use of SEX collector in test 30 and the dosage of depressant at 50% strength in test 31 produced a similar result as that achieved for the baseline test 47, with comparable PGE, Cu, Ni and S recoveries and grade profiles.
- The inclusion of aeration in the pre-rougher conditioning stage in test 33 resulted in reduced mass pull with an improvement in final concentrate grade to 107 g/t as compared to 81 g/t as achieved for the baseline test 47.
- Tests 24, 25, 26, 38 and 39 were conducted without the addition of oxalic acid and thiourea. These tests achieved lower PGE, Cu, Ni and S recoveries to final concentrate as compared to test 47 conducted with the addition of oxalic acid and thiourea.

The PGE grade recovery curves are presented in Figure 13.3.

Figure 13.3 PGE Grade – Recovery Curves for the MF1 Optimisation Tests Using an Oxalic Acid and Thiourea Reagent Suite



13.2.3 Evaluation of Alternative Flowsheets and Reagent Suites

In addition to testing conducted to further evaluate and optimize the oxalic acid and thiourea flowsheet, a series of tests were conducted to evaluate the performance of alternate reagent suites and flowsheet configurations. The following flowsheets and reagent suite configurations were tested in order to compare these to the optimised oxalic acid and thiourea flowsheet:

- MF1 flow sheet using a typical Platreef ore type reagent suite
- MF2 flow sheet using a modified Platreef ore type reagent suite
- MF1 flow sheet using a reagent suite containing a targeted copper collector

These tests were conducted using a mineralised zone composite containing 15% T1, 42.5% T2U, and 42.5% T2L. The head assay of the composite was 3.99g/t 3E + Au.

Testing of alternate flowsheet configurations and reagent suites indicated that:

- The MF1 flow sheet using a typical Platreef reagent suite gave the lowest PGE and nickel recoveries with the highest mass pull to final concentrate.
- A similar PGE recovery and grade could be achieved for the MF2 flow sheet using a typical Platreef reagent suite and the MF1 flow sheet using a targeted copper collector as that achieved for oxalic acid and thiourea flow sheet.
- The oxalic acid and thiourea flow sheet gave a higher sulphur recovery than the MF1 flow sheet using a targeted copper collector.

Initial testing indicated that a similar metallurgical performance to the optimised oxalic acid and thiourea flowsheet can be achieved for both the MF2 circuit with a modified Platreef ore type reagent suite and the MF1 circuit with a targeted copper collector. These circuits present opportunity to remove the requirement for conditioning prior to flotation and remove the risk associated with scale up of the non-standard oxalic acid and thiourea reagent suite. Further development and locked cycle testing has been initiated to confirm these results.

13.3 Locked Cycle Tests on Composite Samples Representing T1, T2U, and T2L

13.3.1 Locked Cycle Testing of a Three Stage Cleaning the Oxalic Acid and Thiourea Reagent Suite and MF1 Circuit Configuration

As part of the PEA flow sheet development, three locked cycle tests were conducted using a split cleaner circuit with an in-mill conditioning time of 5 minutes on mineralised ore blends during Phase 5 and Phase 6A at SGS and Mintek respectively. In addition to this, a further three locked cycle tests were conducted in Phase 6B. The flow sheet used for these tests is presented in Figure 13.4.

The head grades of the split cleaner circuit locked cycle test composites are presented in Table 13.3 while the results of the tests with 5 minutes of in-mill conditioning time conducted as part of the PFS are presented in Table 13.4.

For the composites discussed, the various mineralised zone material ratios were abbreviated as follows:

- 15 % T1, 42.5 % T2U and 42.5 % T2L (Composite 0FW).
- 80 % min zone and 20 % footwall (Composite 20FW).
- 100% footwall (Composite 100FW).
- 12 % T1, 32 % T2U, 50 % T2L and 6 % FW (Composite 6FW).

Figure 13.4 Flow Sheet Used for the Phase 5 and 6 Locked Cycle Tests Using a Split Cleaner Circuit Configuration

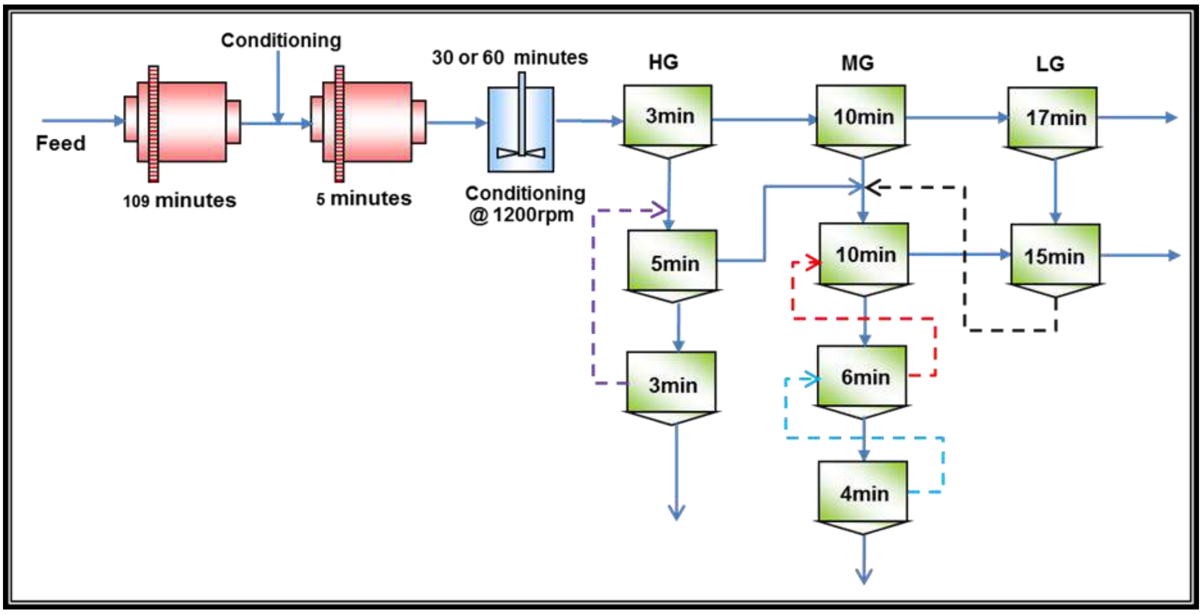


Table 13.3 Split Cleaner Circuit with 5 Minutes In-Mill Conditioning Time Locked Cycle Test Measured Head Assays

Test Phase & Laboratory	Sample Blend	Grade									
		3PE+Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
Mintek Phase 6 #2	Composite 20FW	4.27	1.92	2.01	0.10	0.24	0.20	0.40	1.04	24.0	9.2
Mintek Phase 6 #3	Composite 100FW	2.47	1.04	1.23	0.05	0.15	0.11	0.22	0.73	20.9	7.9
Mintek Phase 6 #4	Composite 6FW	3.87	1.57	1.98	0.11	0.21	0.21	0.40	0.99	25.40	10.33

Table 13.4 Split Cleaner Circuit with 5 Minutes In-Mill Conditioning Time Locked Cycle Test Results

Test Phase & Laboratory	Sample Blend	Mass (%)	Grade						Recovery					
			3PE+Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (g/t)	Fe (%)	3PE+Au (%)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
Mintek Phase 6 #2	Composite 20FW	3.85	96.4	5.2	8.1	22.4	9.0	31.6	87.5	87.0	72.6	88.3	1.5	12.6
Mintek Phase 6 #3	Composite 100FW	3.08	56.7	3.1	4.6	17.8	11.8	28.0	76.6	83.6	60.0	79.2	1.8	10.0
Mintek Phase 6 #4	Composite 6FW	3.77	77.6	4.9	7.1	20.2	11.0	27.2	82.8	86.4	71.2	82.9	1.7	10.3

13.3.2 Conclusion

In the Phase 6B locked cycle test conducted at Mintek:

- The final concentrate mass pull was in the range 3.0% - 3.8%.
- For Composite 100 FW, the calculated PGE (3E + Au) recovery was 76.6% at a final concentrate grade of 57 g/t. The calculated copper recovery was 83.6% at a final concentrate copper grade of 3.1%. The calculated nickel recovery was 60.0% at a final concentrate nickel grade of 4.6%.
- The Composite 100FW test produced lower recoveries by a few percent when compared to the other tests. However, the concentrate grade produced at 57 g/t PGM was lower than saleable concentrate grade of approximately 85 g/t PGM. This is attributed to the lower feed head grade (2.47 g/t PGE) of this sample and it is unlikely that 100% footwall will be presented as feed to the plant.
- The Composite 20 FW test gave a calculated PGE recovery of 87.6% at 96 g/t. The calculated copper recovery was 87.0% at a final concentrate copper grade of 5.2%. The calculated nickel recovery was 72.6% at a final concentrate nickel grade of 8.1%. The calculated sulphur recovery was 88.3% at a final concentrate sulphur grade of 22.4%.
- For the Composite 6FW, the final concentrate mass pull was 3.8%. The calculated PGE (3E + Au) recovery was 82.8% at a final concentrate grade of 78 g/t. The calculated copper recovery was 86.4% at a final concentrate copper grade of 4.9%. The calculated nickel recovery was 71.2% at a final concentrate nickel grade of 7.1%.
- The Composite 6FW had a calculated head grade of 3.87 g/t which was lower than the estimated average for life of mine grade of 4.02 g/t in the mining plan. This composite achieved a lower overall PGE recovery of 82.8% as compared to previous locked cycle testing in Phase 5 and Phase 6 on mineralised zone material using a split cleaner circuit configuration with PGE recovery in the range 85.0% - 87.8%. The lower recovery is attributed to the lower head grade.

13.4 Process Plant Recovery Estimate

The plant recovery estimates are based on locked cycle testing conducted at SGS (Phase 5) and Mintek (Phase 6). The locked cycle test results that were used were from tests conducted using a split cleaner circuit configuration and a reagent suite that includes oxalic acid and thiourea.

13.4.1 Plant Recovery Testwork

The following locked cycle test results were used to derive the process plant recovery estimate:

- SGS Phase 5 – Locked cycle test 1 and 2
- Mintek Phase 6 – Locked cycle test 1, 2 and 4

Mintek locked cycle test #3 was not used as this was conducted on a blend of 100% footwall. In addition, SGS Phase 5 locked cycle tests conducted using 3 stage cleaner circuit to treat a combined rougher concentrate were not used as the results did not represent the expected performance for the optimised split cleaner configuration.

These test results are summarised in Table 13.5.

Table 13.5 Split Cleaner Circuit with 5 Minutes In-Mill Conditioning Time Locked Cycle Test Results

Test Phase & Laboratory	Mass (%)	Grade						Recovery					
		3PE+Au (g/t)	Cu (%)	Ni (%)	Total S (%)	MgO (g/t)	Fe (%)	3PE+Au (%)	Cu (%)	Ni (%)	Total S (%)	MgO (%)	Fe (%)
SGS Phase 5 #1	4.62	68.2	2.7	4.5	12.8	14.9	21.4	86.0	88.3	69.9	80.4	3.0	10.1
SGS Phase 5 #2	3.50	87.7	3.6	5.9	15.8	12.4	24.7	84.9	86.7	68.0	76.7	1.9	8.9
Mintek Phase 6 #1	4.07	93.0	4.9	8.3	24.4	6.0	29.9	87.8	88.4	73.2	87.2	1.0	13.6
Mintek Phase 6 #2	3.85	96.4	5.2	8.1	22.4	9.0	31.6	87.5	87.0	72.6	88.3	1.5	12.6
Mintek Phase 6 #4	3.77	77.6	4.9	7.1	20.2	11.0	27.2	82.8	86.4	71.2	82.9	1.7	10.3

The results obtained for the locked cycle tests were used to develop preliminary recovery estimates for project. Recovery correlations used, were based on individual platinum palladium, rhodium, gold, copper and nickel recoveries obtained from locked cycle testwork and applied to head grades aligned with the preliminary mine schedule. No discount has been applied to account for operational risks associated with full scale operation.

A summary of the Platreef 2014 PFS mine schedule, plant feed grade and run-of-mine ore blend, is presented in Figure 13.5 and the expected tonnage profile for the 4 Mtpa base case mining plan is presented in Figure 13.6.

Figure 13.5 Base Case 4 Mtpa Mine Plan, PGE, Copper and Nickel Head Grades

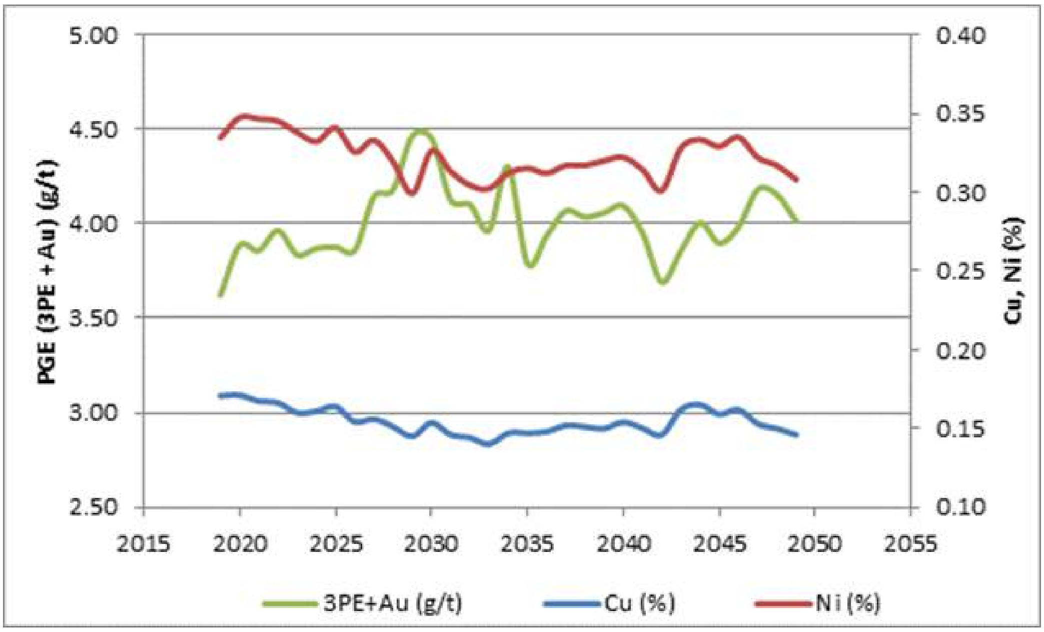
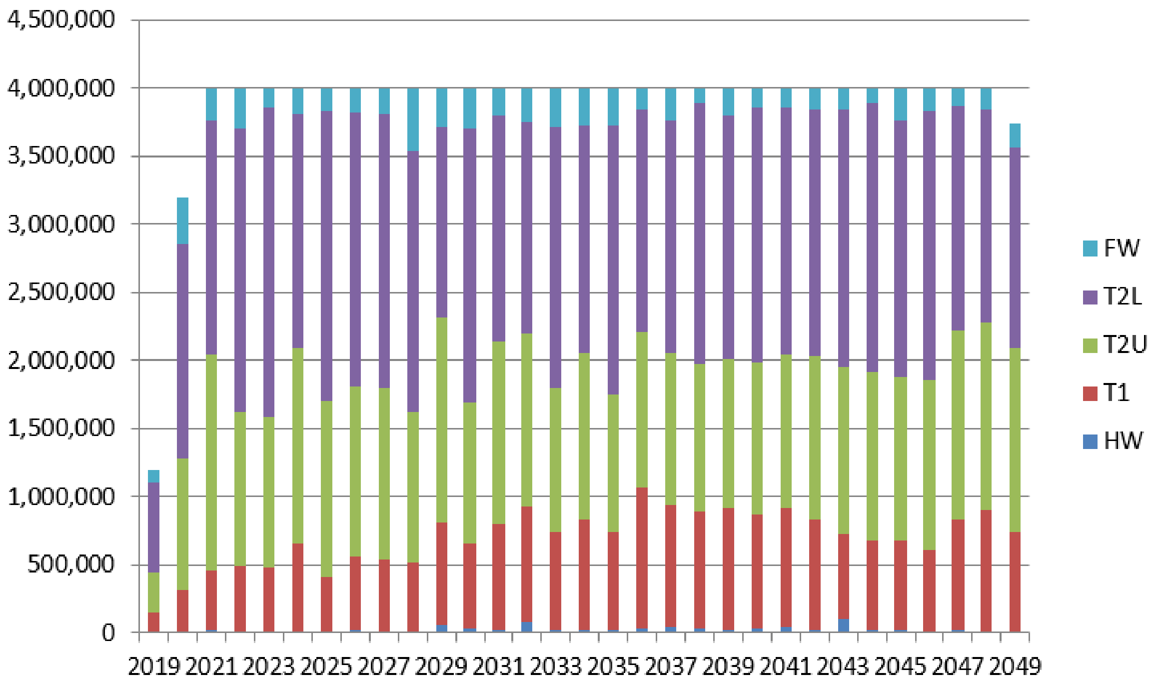


Figure 13.6 Base Case Process Plant Production Profile



The expected process plant tonnage profile and feed grades as presented in Figure 13.5 and Figure 13.6 were used to derive a process plant production schedule for the 4Mtpa base case mining scenario.

Using the locked cycle test results, a final tailings grade for the combined tailings stream comprising of rougher tailings and scavenger cleaner tailings was calculated for each of the 3PE+Au, Cu, and Ni.

Since both the SGS and Mintek test results provide a spread of sample feed grades, within the expected mine plan range, both sets of results were used in the recovery estimates for 3PE+Au, Cu and Ni recovery.

The 3PE+Au metal grades in the flotation feed for each locked cycle test were plotted against the final metal recoveries for the corresponding test. A trend-line was then fitted to the data in order to provide a PFS level estimate for a weighted, best-fit, grade-recovery curve. For a known head grade of the flotation feed, the equations can be used to estimate the overall recovery to be expected for the respective elements. The equations used to derive the recovery estimate for each metal are presented in Table 13.6. It should be noted that the recovery equations have only been validated for the range of head grades evaluated during testwork.

Table 13.6 Equations Used to Derive Recovery Estimates

Element	Equation to Estimate Tailings	Head Grade Range Applicable to Equation
Pt	$Pt_{recovery} = 0.7711x\ Head\ Grade^{0.2178}$	1.37 g/t – 1.92 g/t
Pd	$Pd_{recovery} = 0.8132x\ Head\ Grade^{0.1054}$	1.63 g/t – 2.06 g/t
Rh	$Rh_{recovery} = 3.3018x\ Head\ Grade^{0.5874}$	0.06 g/t – 0.12 g/t
Au	$Au_{recovery} = 1.1168x\ Head\ Grade^{0.2752}$	0.22 g/t – 0.25 g/t
Cu	$Cu_{recovery} = 0.8594x\ Head\ Grade^{-0.011}$	0.14 g/t – 0.23 g/t
Ni	$Ni_{recovery} = 0.8262x\ Head\ Grade^{0.1619}$	0.30 g/t - 0.46 g/t

The derived recoveries relative to head grade are presented in Figure 13.7 to Figure 13.9. These curves are only valid for the ranges indicated in Table 13.6.

Figure 13.7 3PE+Au Recovery as a Function Of Head Grade

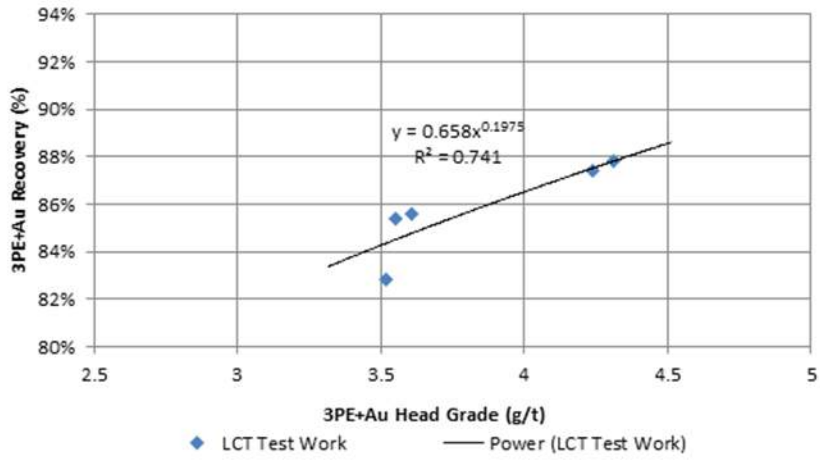


Figure 13.8 Cu Recovery as a Function of Head Grade

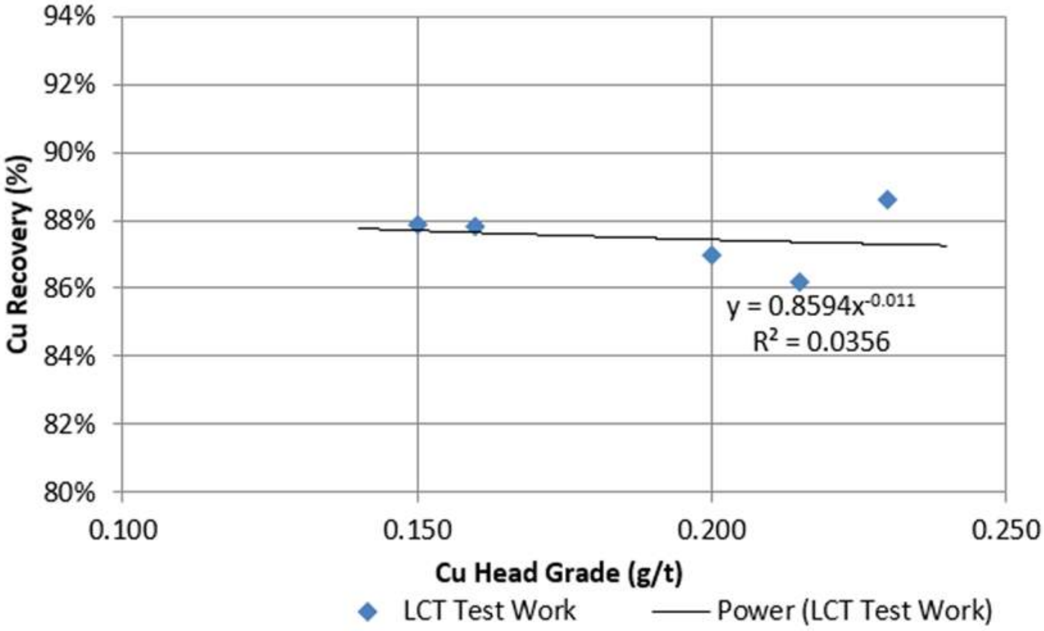


Figure 13.9 Ni Recovery as a Function of Head Grade

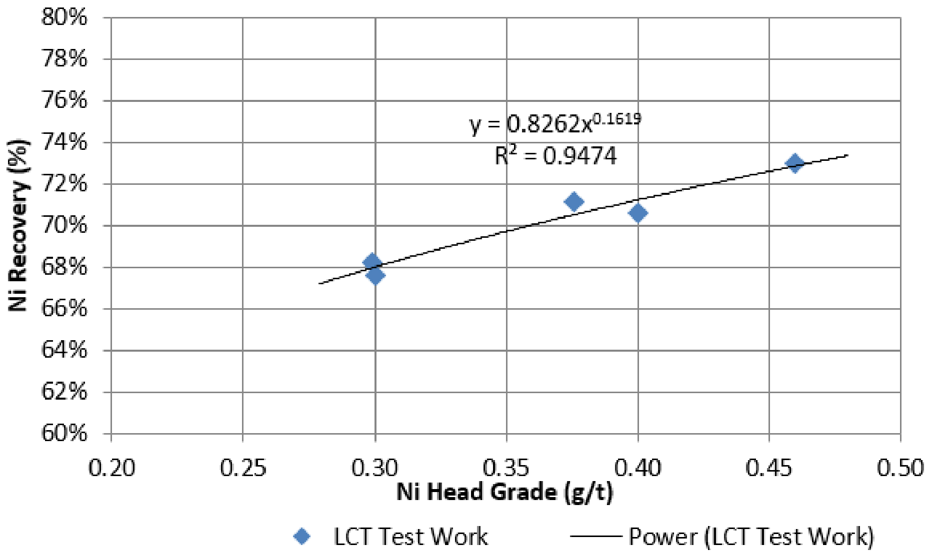


Figure 13.10 Pt Recovery as a Function of Head Grade

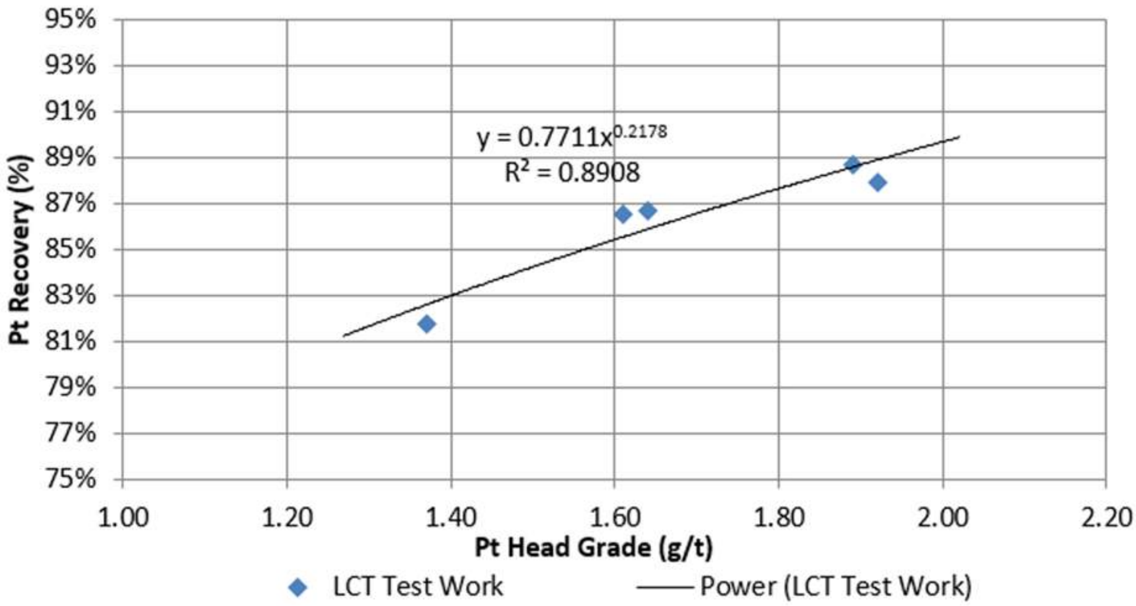
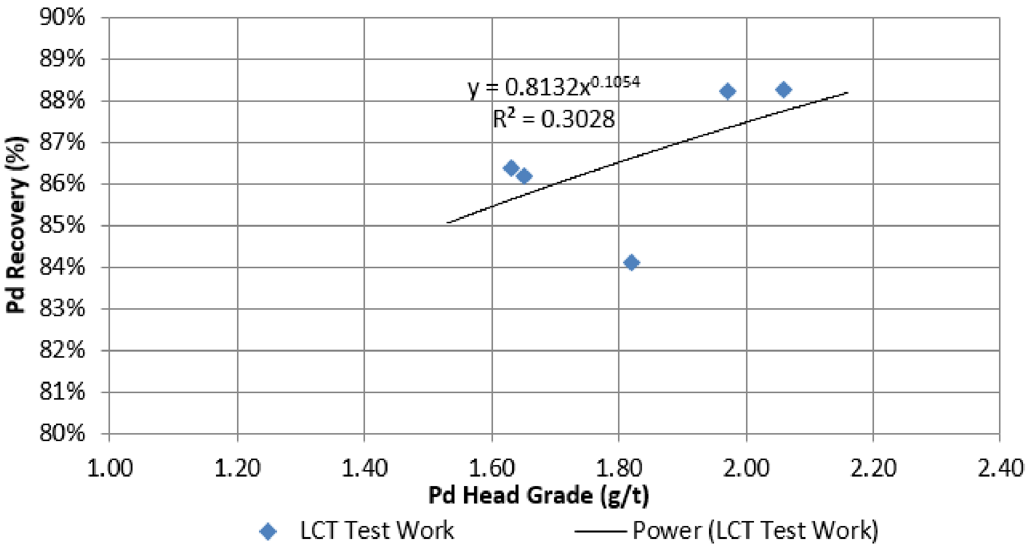


Figure 13.11 Pd Recovery as a Function of Head Grade



Gold and Rhodium recoveries have not been illustrated as they comprise only a minor proportion of the overall 3PE+Au recovery. The graphs show that, in most instances, a generally good correlation is observed between plant feed grade, modelled recovery and testwork recovery. The results do show a fair degree of scatter, highlighting both opportunity and risk. Subsequent testwork phases will focus on completing variability testwork to reduce the uncertainty and describe the recovery variability. These modelled life of mine recovery estimate ranges, for the individual metals, are presented in Table 13.7.

Table 13.7 Platreef Recovery Estimate

	Estimated Recovery	
Description	**LOM Head Grade Range PGE (g/t), Cu, Ni (%)**	**Expected LOM Recovery Range Estimate**
Copper (Cu)	0.14 - 0.17	88%
Nickel (Ni)	0.30 - 0.36	68%-70%
Platinum (Pt)	1.52 - 1.95	85%-88%
Palladium (Pd)	1.71 - 2.10	86%-88%
Gold (Au)	0.23 - 0.29	75%-79%
Rhodium (Rh)	0.11 - 0.15	88%-92%

3PE+Au recovery was estimated to be in the range 85% - 88% based on a head grade range of 3.58g/t – 4.46 g/t from the mine production schedule and the grade recovery relationships derived from testwork as presented in Table 13.6. The average LOM recovery is estimated to be 86.5% based on a head grade of 4.0 g/t.

This recovery estimate is in agreement with results from five (5) locked cycle tests conducted in Phases 5 and 6, where 3PE+Au recovery was in the range 83% - 88% for calculated test head grades in the range 3.53g/t – 4.31 g/t.

Variability has been seen in the testwork, with Mintek locked cycle test #4 conducted on Composite 6FW having a lower 3PE+Au recovery of 83% at a calculated test head grade of 3.53g/t as compared to the 85% - 88% 3PE+Au recovery achieved in the other four locked cycle tests with calculated test head grades of 3.55 g/t – 4.31 g/t. Variability testwork that will be conducted as part of the FS will aim to quantify the variability across the deposit and between each of the geometallurgical units in order to further define the expected recovery range.

Locked cycle testing in Phases 5 and 6 achieved concentrate grades (3E+Au) in the range 68g/t – 96 g/t for mass pulls in the range 3.5% - 4.6%. The average mass pull was 4.0% and the average grade was 85g/t 3PE+Au as compared to the saleable concentrate target grade of 85g/t for full scale plant operation. Based on the head grade range of 3.58g/t – 4.46g/t from the mine production schedule and the grade recovery relationships derived from testwork as presented in Table 13.2 the expected mass pull for full scale plant operation has been calculated to be in the range 3.5% - 4.1%. This is in agreement with the mass pull range achieved in locked cycle tests. In some instances, the rhodium tailings assay values were below the detection limits for the lead collection analytical method used. Given the uncertainty, the rhodium recovery could lie within the extreme range of 55% - 100% range. In order to quantify the effect of this uncertainty, the sensitivity of the revenue stream generated by the recovery of 3PE+Au was tested for the uncertainty range. It was calculated that the rhodium recovery would have a maximum effect of a 1.5% variation in the revenue stream generated. Rhodium recovery can be defined more fully during the subsequent study phase using more accurate analytical methods, such as the nickel sulphide fire assay method for the collection of the platinum group elements and gold.

13.5 Concentrate Specification

The flotation concentrate to be transported by road to a smelter is to have a target 3PE+Au grade of 85 g/t. The flotation mass pull will be instrumental in achieving this targeted specification, and is expected to be in the range of 3.6% - 4.1%.

Mass pull will also affect the recoveries achieved for PGE's (3PE+Au) and sulphur. Sulphur grade in the concentrate is a consideration for the take-off agreement for smelting and refining operations of the concentrate.

The base metal sulphide content in the concentrate has an effect on the downstream processing steps. Concentrates with a high base metal sulphide content generally result in high matte falls during electric furnace smelting. The high matte falls could place pressure on converting, sulphur capture and copper and nickel refining capacities. The Platreef concentrates recovered with the oxalic acid and thiourea reagent suite during testwork have tended to have sulphur grades of >15%. High matte fall containing concentrates such as these can be blended with low matte fall containing concentrates (UG2-type) to produce a blend suitable for smelting.

Furthermore, the low chromite content of the Platreef concentrate provides a measure of mitigation when blending with the typically higher chromite content in UG2-type concentrates.

Future testwork phases will look to reduce the recovery of sulphide gangue (chiefly pyrrhotite) into the concentrate and target an overall sulphide grade <15%. This will allow for downstream processing of the concentrate without pre-treatment processes such as roasting.

13.6 Metallurgical Variability

The geometallurgical units chosen, under the guidance of the geological team, and the sample blends tested are adequate plant feed blends for the Platreef deposit and are suitable for a PFS level of study. Variability testwork is required to provide data for the feasibility study in order to quantify the variability across the deposit and between each of the geometallurgical units as identified in the mine plan, and the effect of variability on metal recovery and processing costs. The variability drilling program has been conducted.

In addition to the variability associated with the deposit, laboratory testing has indicated that the choice of grinding media has an impact on the metallurgical performance with improved performance when using wear resistant grinding media such as stainless steel or high chrome grinding rods. Testwork has indicated that 440C high chrome grinding media gives similar results to stainless steel media and taking into consideration that stainless steel grinding media is not commercially available, it is the recommendation that future testing should be conducted with high chrome media.

13.7 Comments on Section 13

It is the opinion of the qualified person responsible for this section of the report, Mr. Val Coetzee, that commensurate with the level of study, the amount of mineral processing testwork that has been conducted is considered to be adequate. The metallurgical characterization testwork has confirmed the findings of the mineralogical analysis and the recovery prediction made by the Mintek mineralogy department in Phase 4.

Based on testwork conducted to date, an overall PGE recovery in the range 83% - 88% can be expected at a target concentrate grade of 85g/t (3E + Au) with the implementation of grade measurement and mass pull control. Mass pull is expected to vary between 3.6% - 4.1% with varying head grades as indicated in the PFS mine plan. Testwork results indicate that a degree of scatter is to be expected around the estimates provided. Once concentrate specifications and off-take terms, by others, progress, testwork will be aligned to optimize the process efficiencies. Future testwork should aim to describe the metallurgical response variability.

Further testwork will be required to obtain parameters for the design of the process plant. These would include:

- Further development of the MF1 flow sheet to identify a reagent suite that allows for minimisation of the concentrate sulphur grade while maintaining PGE grade targets and recovery.
- Opportunities to optimise any future flotation circuit configuration.
- Optimise the final concentrate grade and specifications aligned with marketing off-take negotiations.
- Variability testwork.

Based on the testwork performed as part of the pre- feasibility study, two major processing risks have been identified.

13.7.1 Reagent Suite

The Platreef flotation circuit development testing indicated that a reagent suite with oxalic acid and thiourea addition, allowed for improved PGE recovery and grade in final concentrate. The use of oxalic acid and thiourea has been published in a flotation reagent text book: Bulatovic, SM 2007, Handbook of Flotation Reagents: Chemistry, Theory and Practise: Flotation of Sulfide Ores, Elsevier Science & Technology Books, ISBN: 0444530290. However, the use of this reagent suite has not been employed, on commercial scale, at any South African platinum flotation operation.

The PFS flow sheet is based on an oxalic acid and thiourea reagent suite with the inclusion of a post mill conditioning stage.

A standard flotation reagent suite, similar to that employed by platinum flotation operations in the vicinity of the Platreef deposit, has been tested and to date was found to give comparable results to the oxalic acid and thiourea reagent suite when using an MF2 (mill-float followed by mill-float) circuit configuration. Furthermore, 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. Further development work on this flow sheet is ongoing, but sufficient results on these flow sheets are not yet available in order to be able to draw definitive conclusions.

Potentially, these flow sheets may be preferable to the oxalic acid and thiourea reagents suite for the following reasons, and should be explored during the feasibility study phase:

- The oxalic acid and thiourea suite flow sheet requires adequate conditioning time for optimum effectiveness. The conditioning time requirement could be removed with an alternative flow sheet.
- The oxalic acid and thiourea reagent suite has not been used in the South African platinum industry and use of one of the alternative flow sheets could serve to remove the risk associated with the use of this novel reagent suite.
- Preliminary open circuit testing of the MF1 circuit using a reagent suite containing a targeted copper collector has shown potential to achieve lower first pass sulphur recovery to final concentrate.

13.7.2 Recovery Estimate

The recovery estimate derived was based on the results achieved from five locked cycle tests conducted on composite samples. The mine plan includes geometallurgical units T1, T2U and T2L as well as fractions of hanging wall and footwall in varying ratios. Further testing on mineralised ore blends with the inclusion of footwall and hanging wall in the correct ratio (aligned with the mining method and schedule) would be required to better quantify the effect on recovery and operating costs. Further to this, focused variability testwork would be required in future phases of study to more accurately quantify and describe the expected recovery and highlight what degree of variability could be expected.

It should be noted, that flotation recovery for full scale operations can be lower than that achieved in a laboratory due to operational inefficiencies such as those listed below:

- Variation in ore types/blends: The mine plan indicates that 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. Further variability drilling and testwork would be required to quantify the variability across the deposit and between each of the geometallurgical units as identified in the mine plan.
- Power: The laboratory flotation cell power (and air) inputs are high (typically 10 kWh/m3). This may tend to give higher recoveries due to the improved fines (<20 mm) recovery.
- Milling type: The milling in the laboratory is generally undertaken using a 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 these operational challenges, 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" to enable improved flotation control. Process operators need to be trained and supervised so as to reduce the occurrence of losses due to poor operational practices. Variability testing during the feasibility study phase will aim to quantify the variability across the deposit and between each of the geometallurgical units as identified in the mine plan.

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 Resouce Area for the UMT-TCU

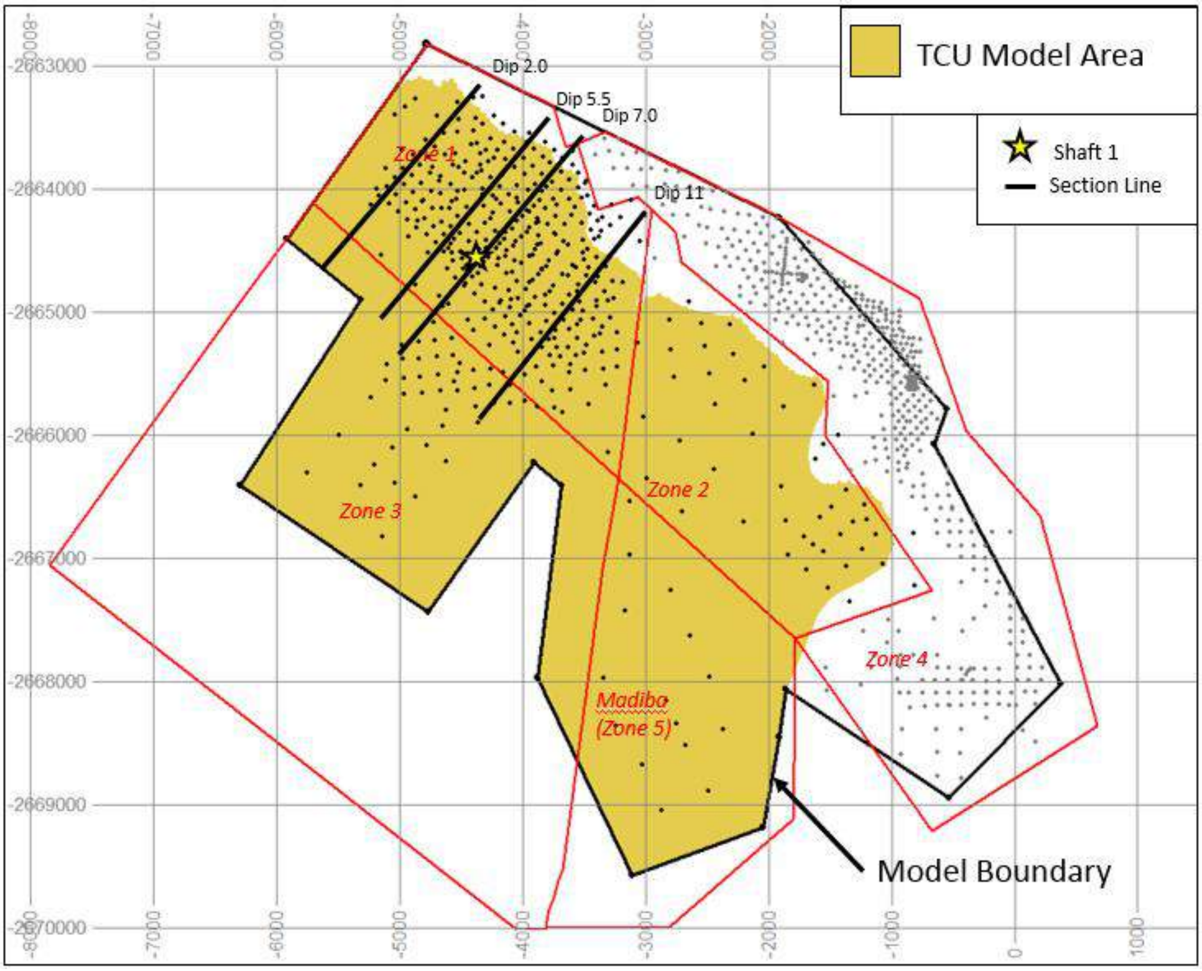
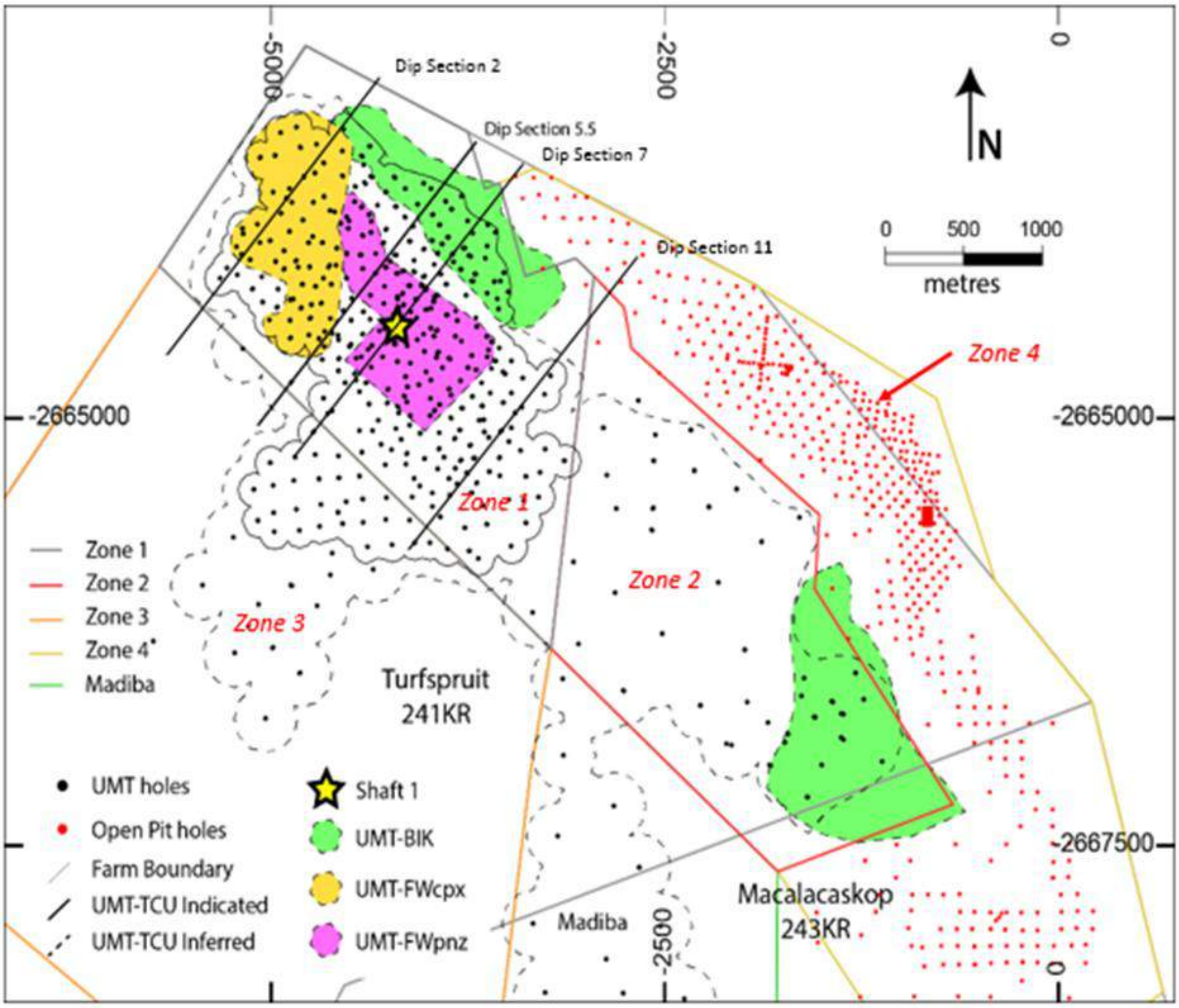


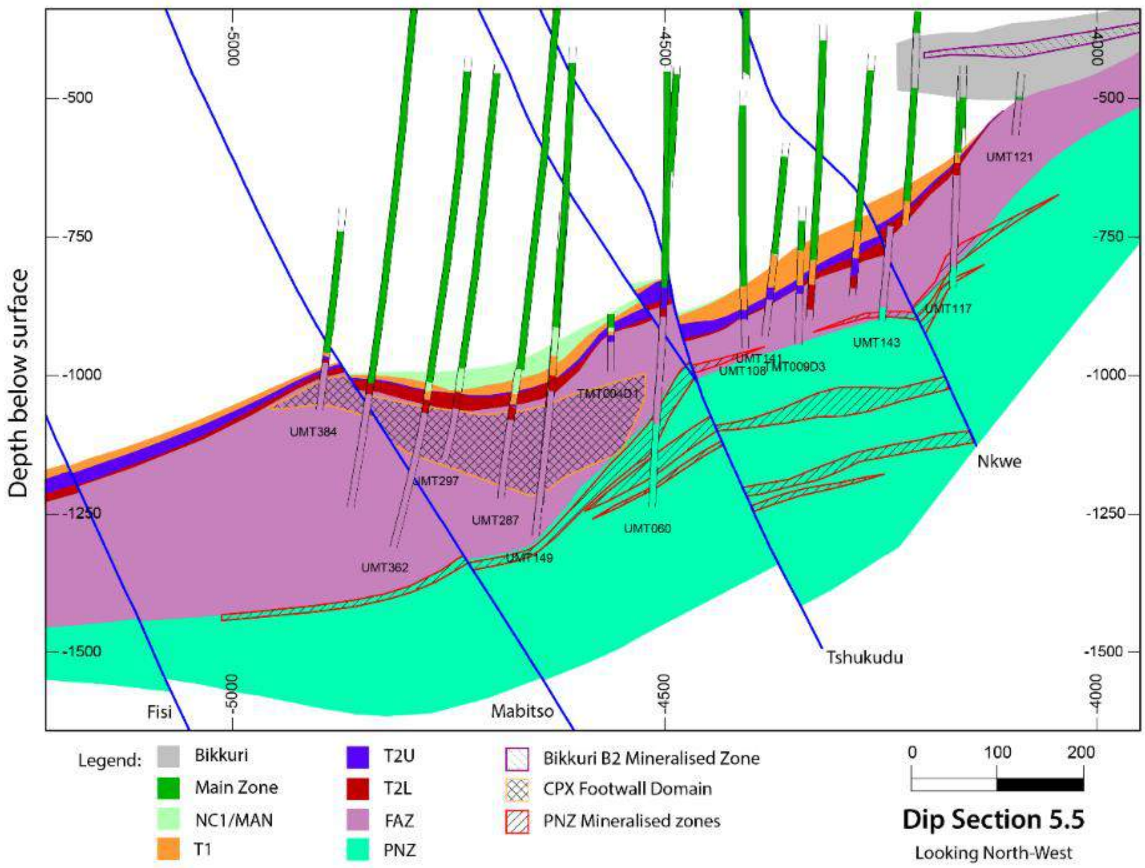
Figure prepared by Amec Foster Wheeler, 2016.

Figure 14.2 Mineral Resouce 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 summarized in (Table 14.1).

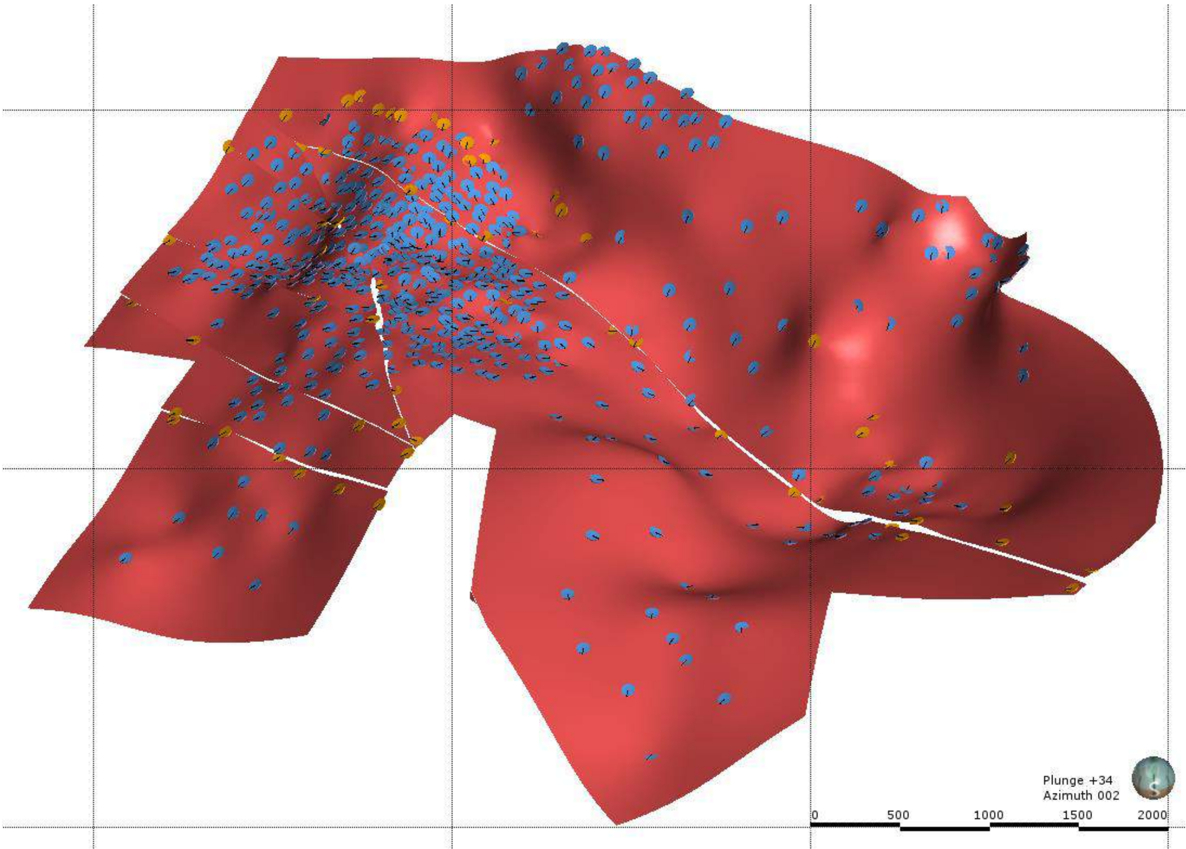
Relogging of drill core, infill drilling in Zones 1 and 2, expansion drilling in the Madiba area and a revised structural model since 2013 has resulted in modifications incorporated in the 2016 geology model.

The revised structural model identified or inferred numerous faults. Only faults with a high degree of confidence were used for the geology model. These incluce 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 drill hole 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 Assimulated Zone	FAZ	30
Pyroxenite Norite Zone	PNZ	31
Lower Zone	LZ	32
FW	FW	FW

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 contructed 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 contrain higher grade mineralization. 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 cutoff date for the drillhole database used for the Mineral Resource resource estimate was 24 July 2015.

Drill holes 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 drill holes 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 drill holes and provide a single intercept per cluster of wedge holes. The analysis determined that differences in mineralized 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 (see Table 7-16).

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 450m 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 20m above the top of the T1MZ. The lower surface of the model evelope 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 contrain the grade estimation. The nested grade shells were indentified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE (Pt+Pd+Rh)+Au cutoffs. The stratigraphic location of the mineralized intercepts was considered in the grade shell construction. The grade-shell intercepts were coded into the drillhole database. The grade-shell drill hole intercepts were validated on dip and strike sections to ensure consistency. The grade-shell drill hole 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).

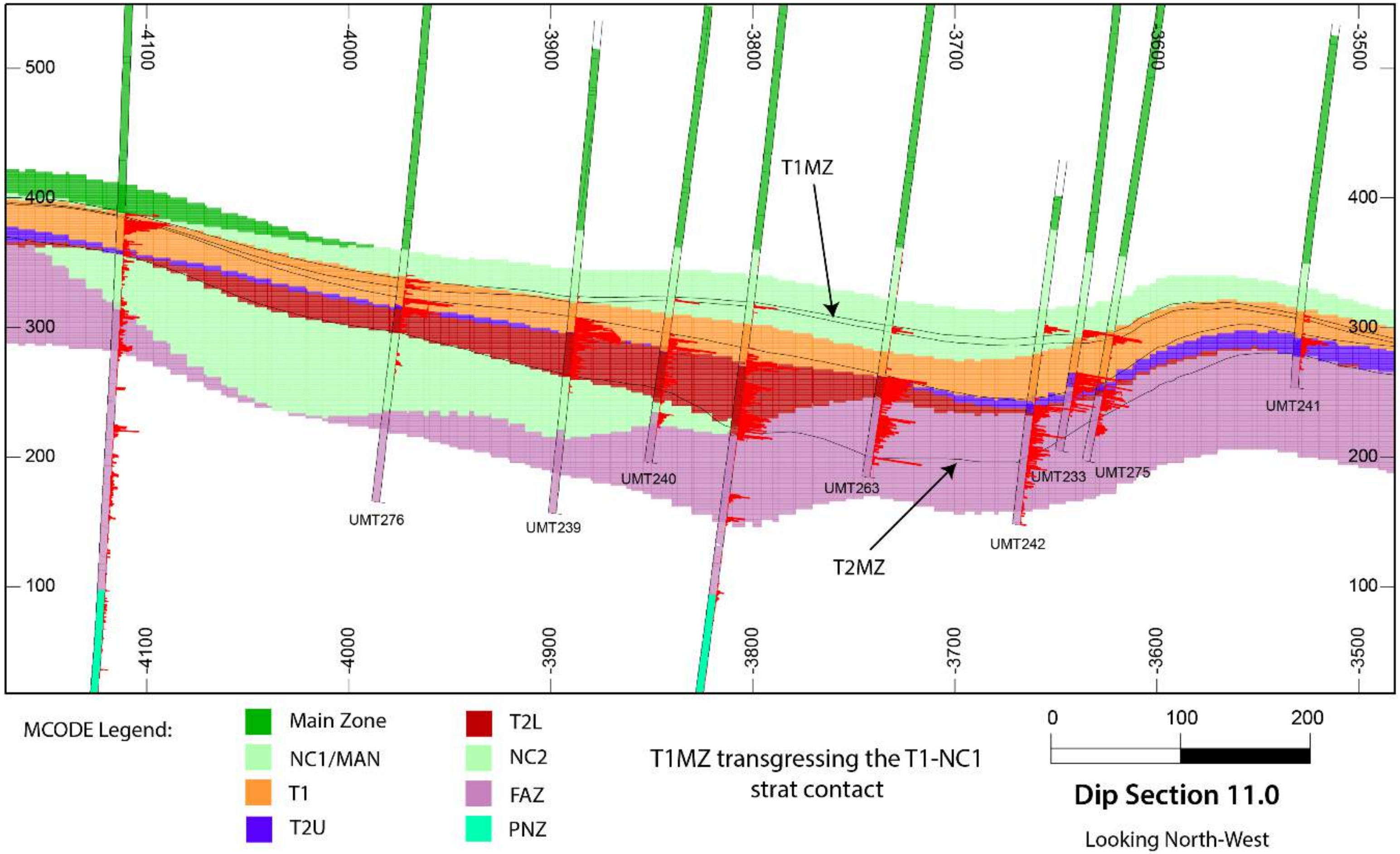
14.2.4.2 T2MZ

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

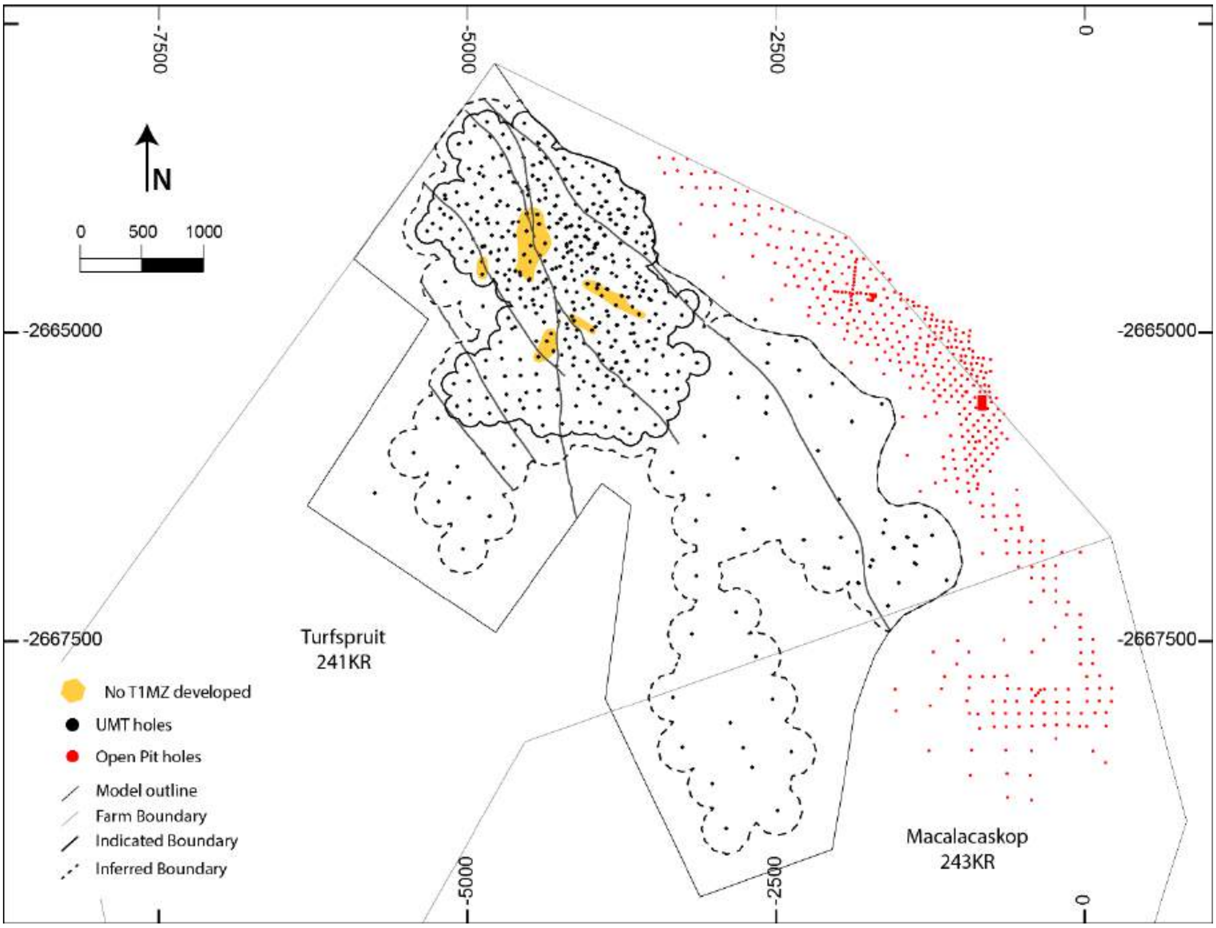
There is scattered mineralisation adjacent to the TCU mineralised zones that is locally continuous. Floating stope software is used in mining-related studies, and mineralisation adjacent to the TCU mineralised zones can be included in the resultant stopes; hence there is a need to estimate grades in 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
	NCBK, MANBK	(Outside Grade Shells)	0
		B1MZ 1g	301
		B1MZ 2g	302
		B1MZ 3g	303
TCU-BIK	B1	(Outside Grade Shells)	1
		B2MZ 1g	401
		B2MZ 2g	402
		B2MZ 3g	403
	B2	(Outside Grade Shells)	2
	LZBK	(Outside Grade Shells)	0
	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
UMT-TCU		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

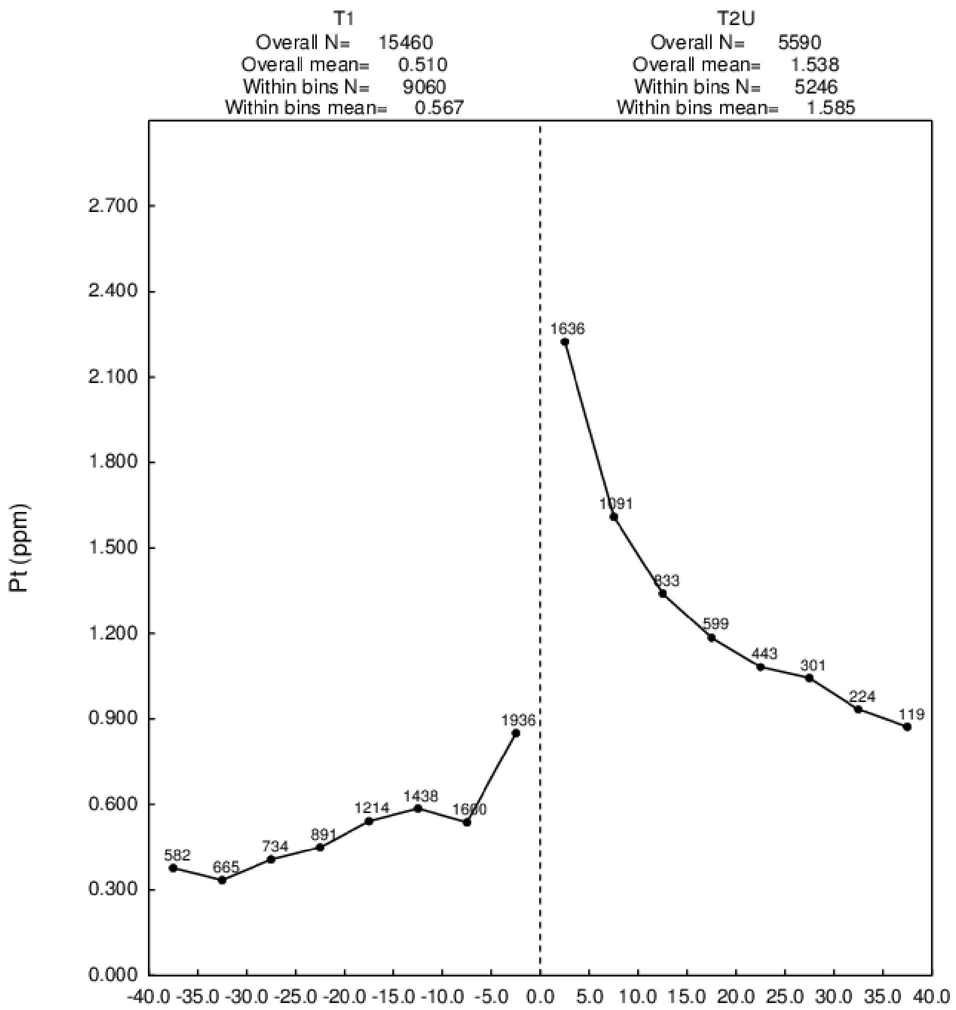


Figure prepared by Amec Foster Wheeler, 2015.

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

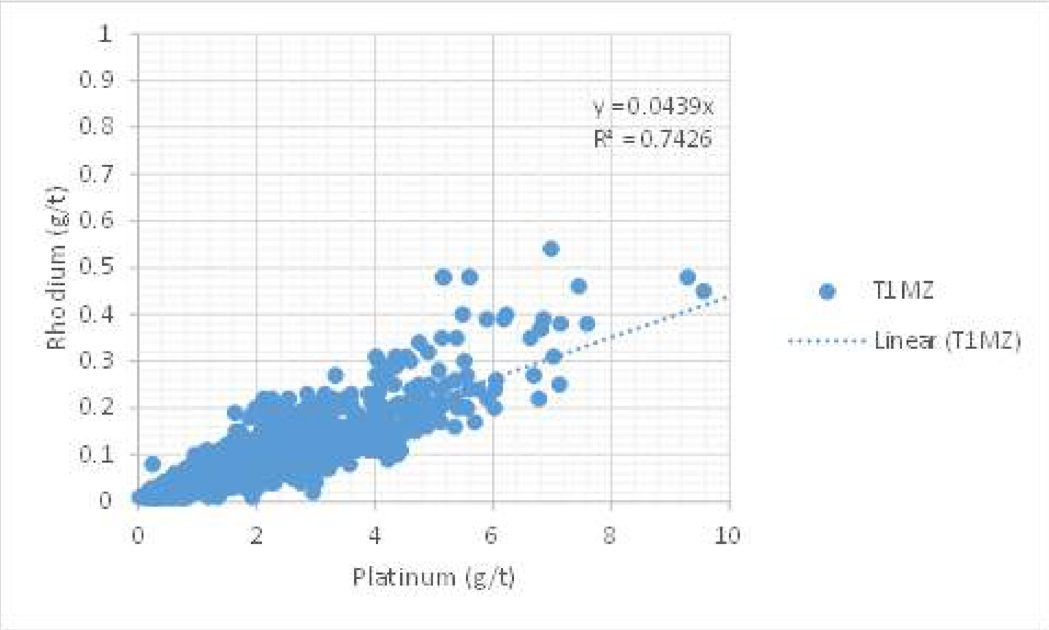


Figure prepared by Amec Foster Wheeler, 2015.

Figure 14.9 Rhodium Regression for the T2

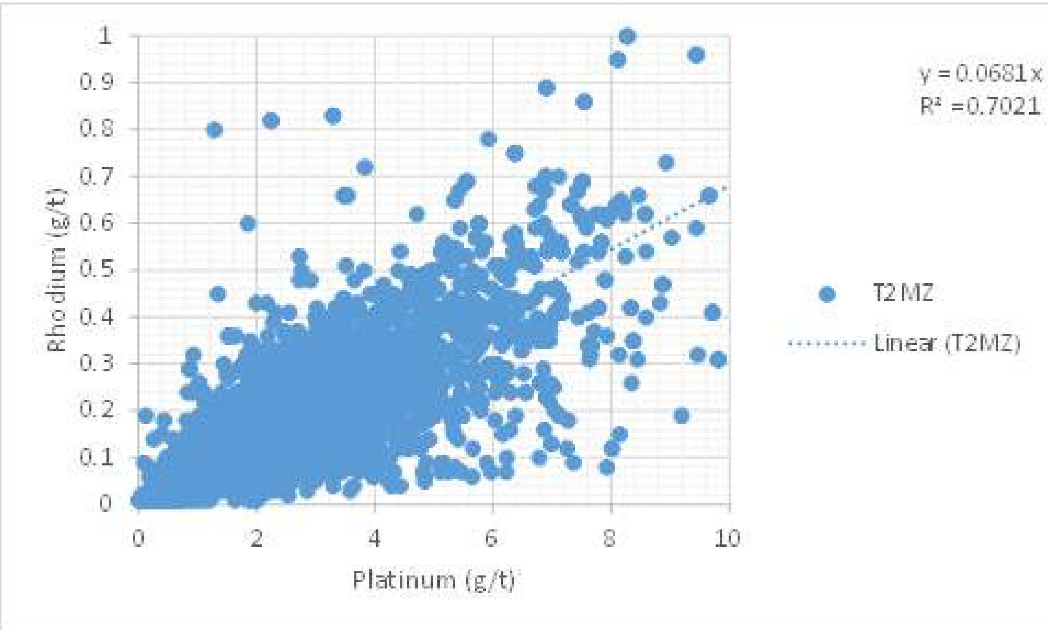


Figure prepared by Amec Foster Wheeler , 2015.

Table 14.3 Proportions of Rhodium Assays by Strat Code and 3PE+Au Grade Shell

Inside Mineralized 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 Mineralized 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

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	1026	453	573	56
	2 g/t	303	239	64	21
	3 g/t	1939	1406	533	27
T2MZ	1 g/t	6383	2781	4057	59
	2 g/t	5006	3515	1491	30
	3 g/t	8417	7812	605	7

14.2.7 Variography

Pair wise relative variograms were completed by grade shell. Figure 14.11 shows a downhole pairwise relative variogram for Platinum. Figure 14.11 shows a directional pairwise relative variogram for Platinum (aziumuth 137, dip 0).

Figure 14.10 Downhole Pair Wise Relative Variogram for Platinum

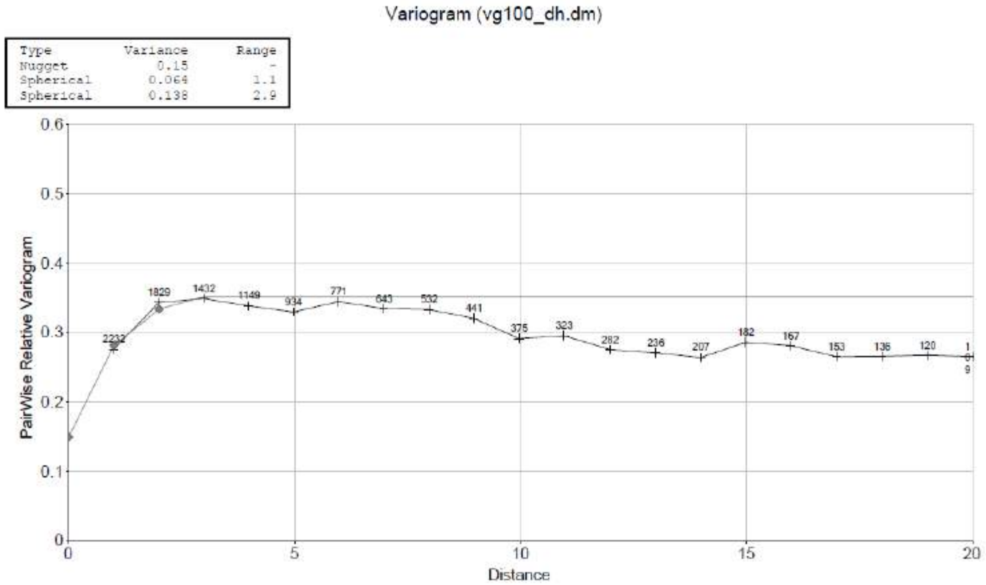


Figure prepared by Amec Foster Wheeler, 2015. Lag distances are in metres.

Figure 14.11 Directional Pair Wise Relative Variogram Model for Platinum (Az=135)

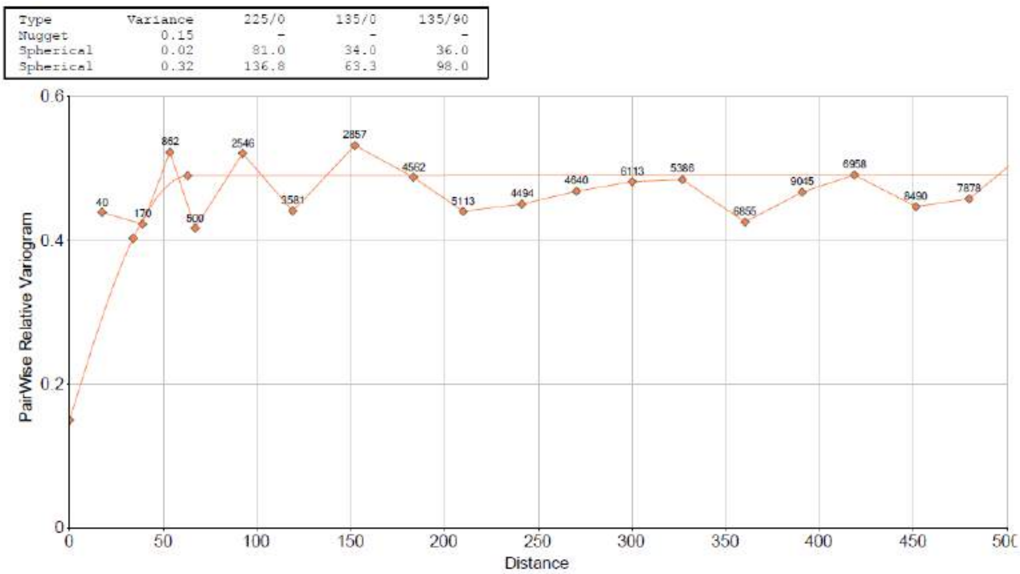


Figure prepared by Amec Foster Wheeler, 2015. Lag distances are in metres.

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 10m x 10m x 2m. Sub-celling was 5m x 5m x 0.5m. 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 1000 m elevation.

The zones were hung at the center of the stratigraphic units or mineralized zone with the exception of the T2MZ. The mineralization in the T2MZ is commonly top-loaded and the 1g/t, 2g/t and 3g/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 Sample	Max per Drill Hole
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 Stratigaphic Units (MCODE)

Zone	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	Ni (%)	Cu (%)	S (%)
MZ	0.40	0.50	0.10	0.05	0.22	0.30	2.0
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

14.2.9.6 Regularisation

Following grade estimation, the UMT-TCU subcell model was regularized to 10m x 10m x 2m blocks. However, the T1MZ was regularized to 10m x 10m x 1m blocks. The T1MZ is narrower than the T2MZ and commonly bounded on both contacts by barren material. The smaller regularized blocks in the T1MZ reduced dilution on the top and bottom contacts of the T1MZ mineralised zone. The regularisation softened the hard boundaries used in the grade estimation.

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	17368	2.90	0.03
NC1/MAN	1184	2.95	0.05
T1	2387	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	6453	3.11	0.05
PNZ	4001	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 Resourceand 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 mineralization. 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 drill hole spacing suitable for Indicated Mineral Resources to the wider drill hole spacing below the TCU suitable for Inferred Mineral Resources. This surface was used to define a vertical boundary between Indicated and Inferred Mineral Resources. A similar methodology was applied in the Madiba area where the drill hole 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 14.2. No Measured Minerl 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 drill hole 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 14.3 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

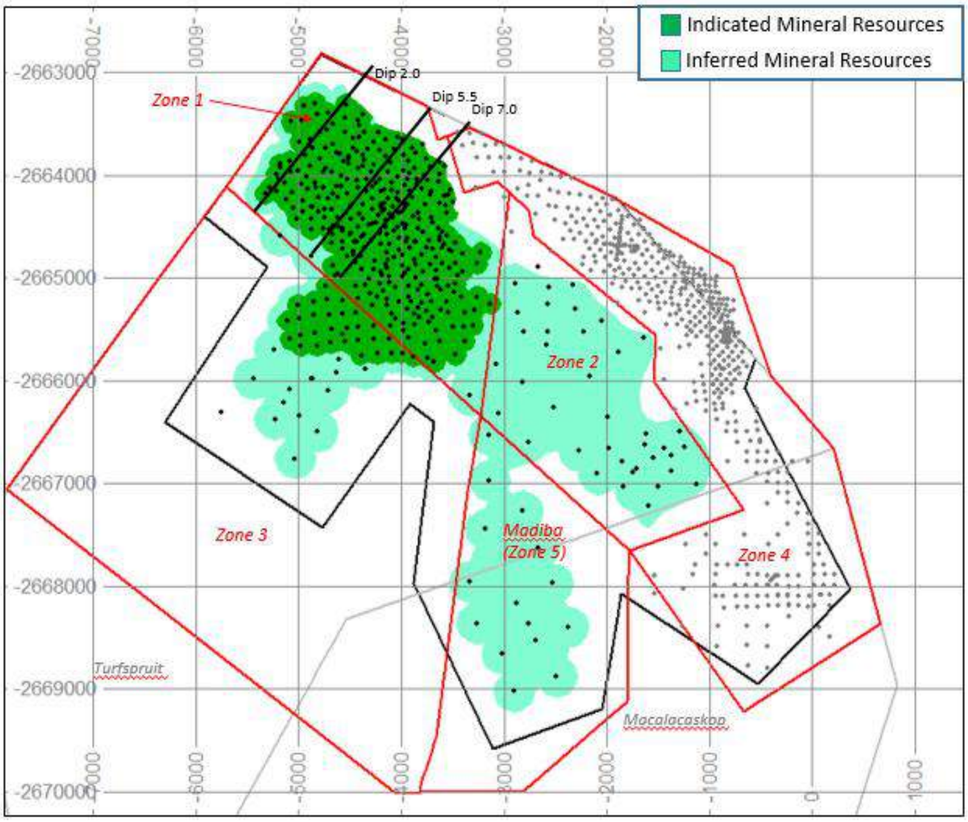
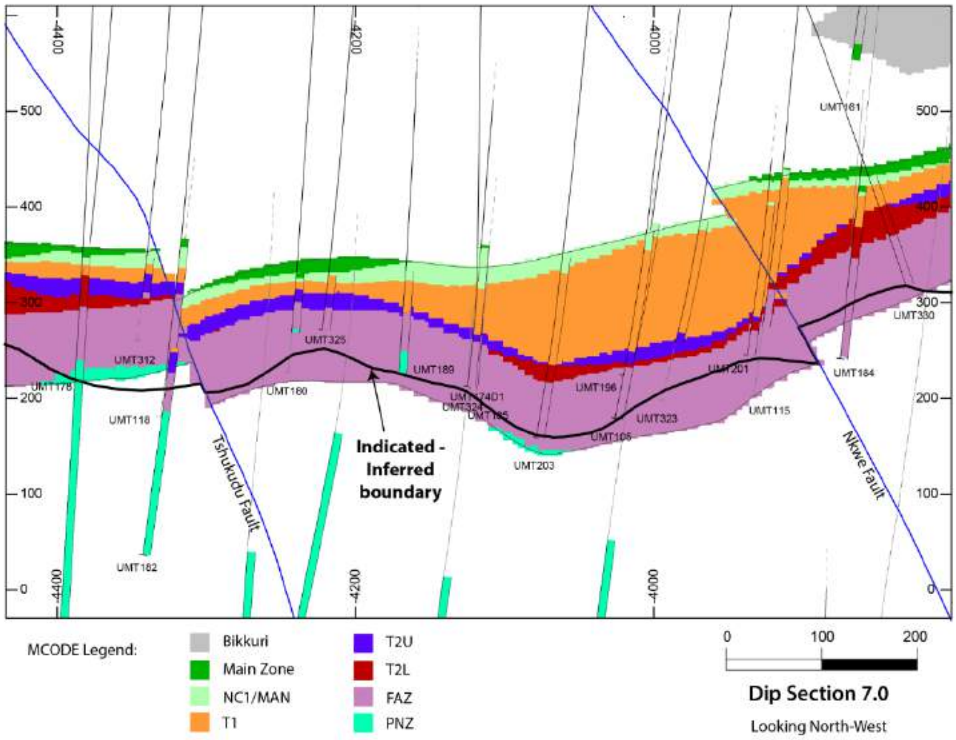


Figure prepared by Amec Foster Wheeler, 2016.

Figure 14.13 Dip Section 7.0; Vertical 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 Drill Hole Projection) Looking Northwest

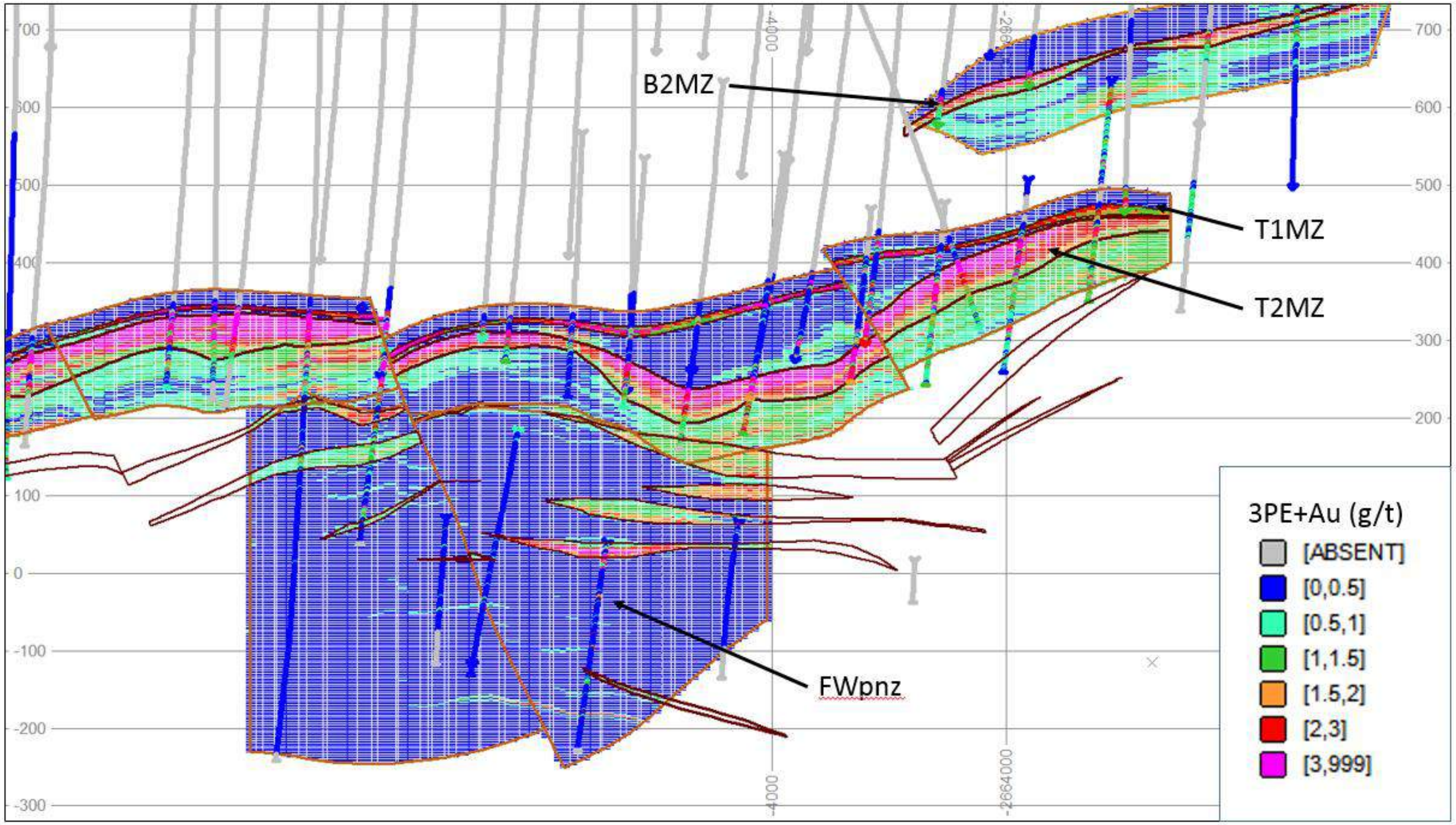


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 7 is shown on Figure 14.2.

Figure 14.15 Dip Section 7.0 Showing Ni% (50 m Drill Hole Projection) Looking Northwest

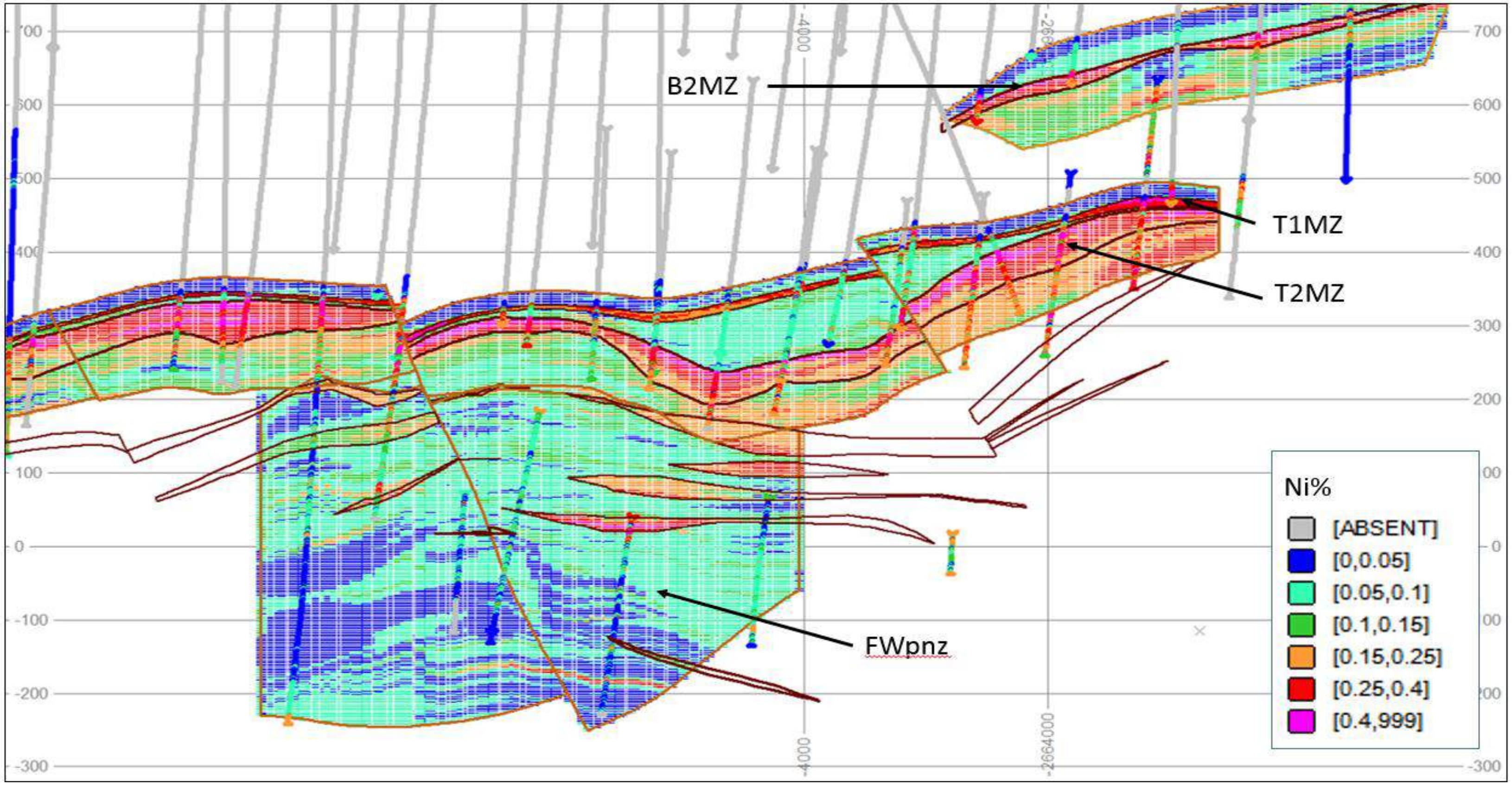


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 7 is shown on Figure 14.2.

Figure 14.16 Dip Section 2.0 Showing 3PE+Au (50 m Drill Hole Projection) Looking Northwest

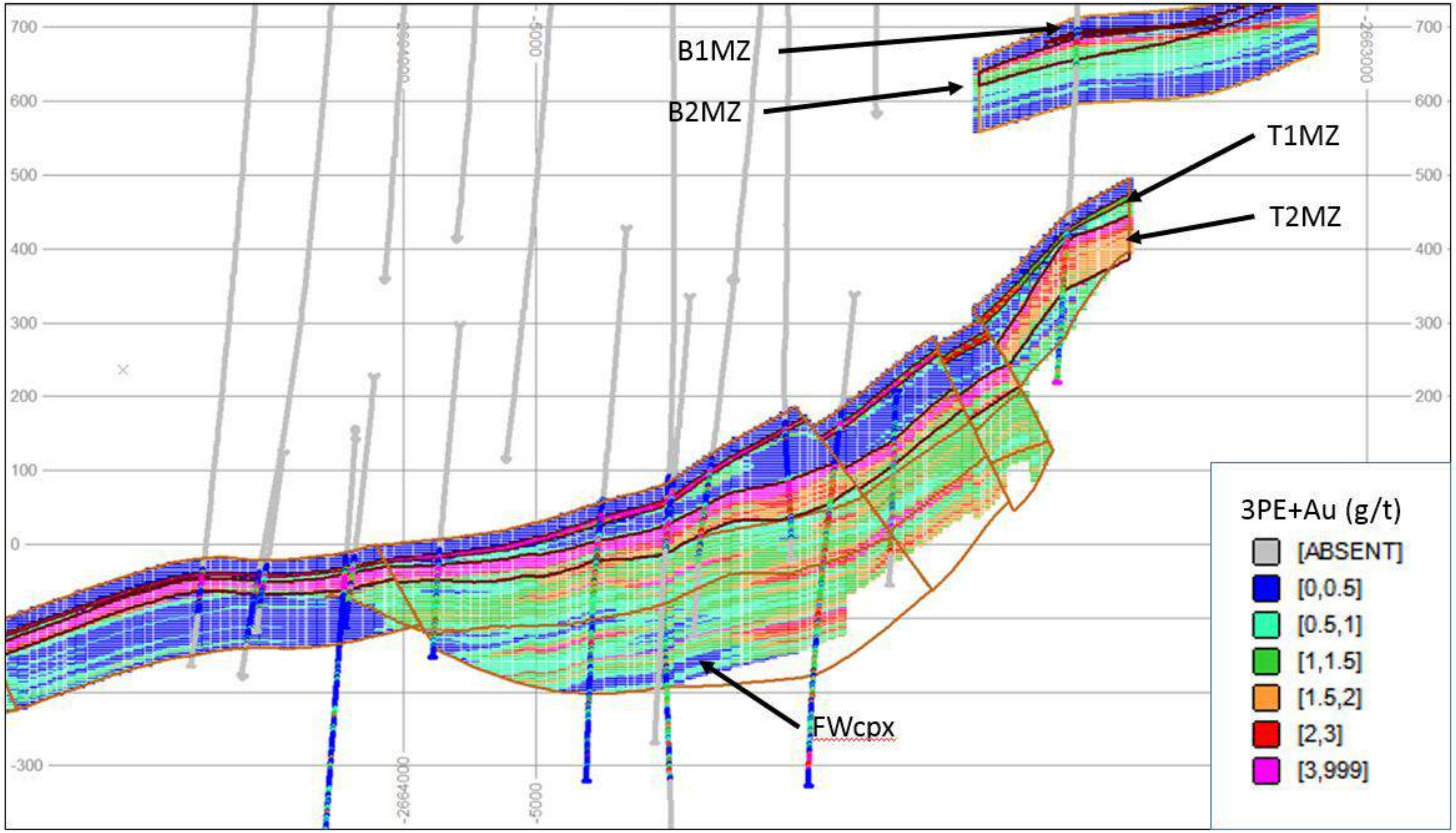


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 2 is shown on Figure 14.2.

Figure 14.17 Dip Section 2.0 Showing Ni% (50 m Drill Hole Projection) Looking Northwest

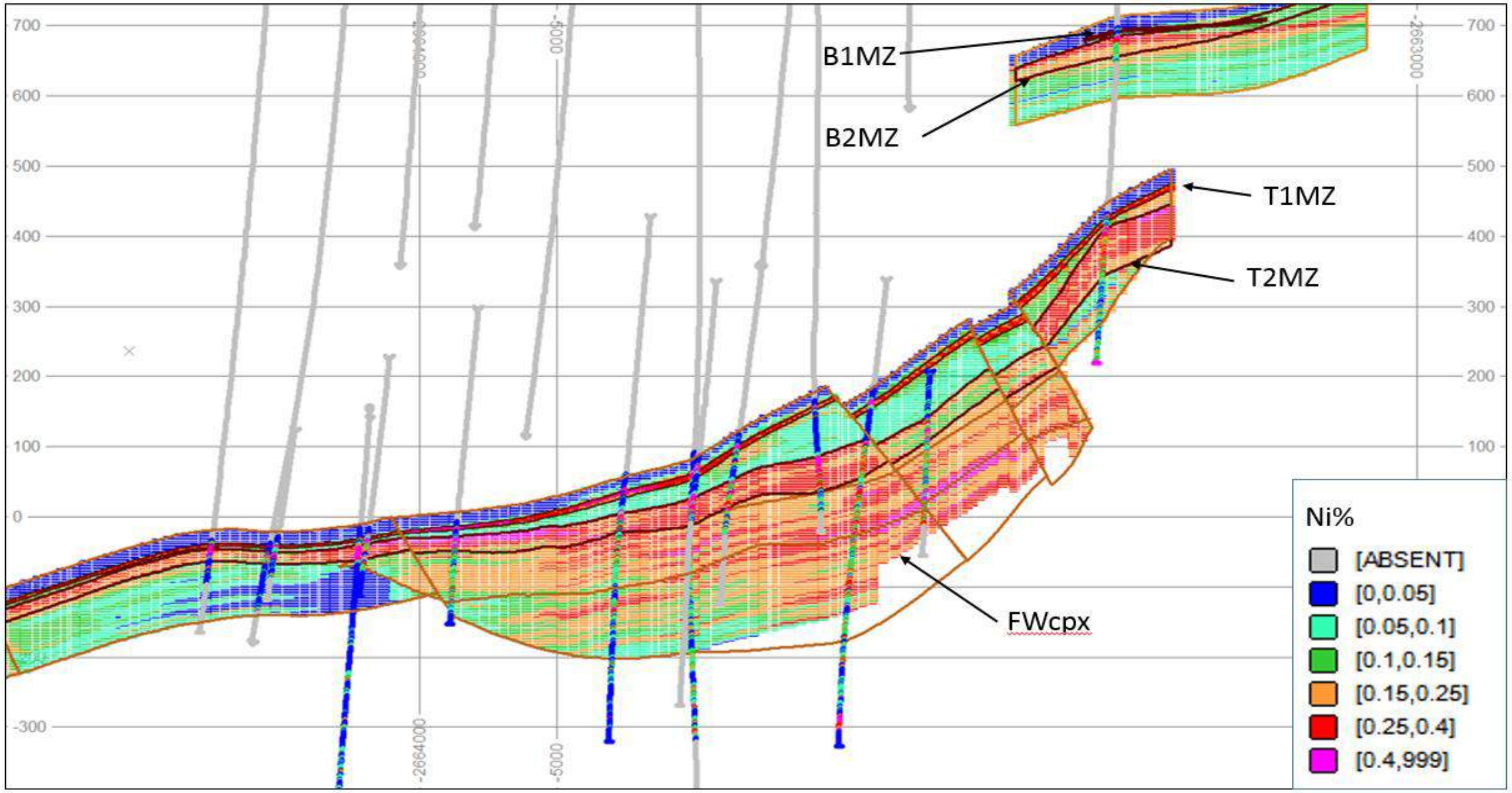


Figure prepared by Amec Foster Wheeler, 2016. Location of Dip 2 is shown on Figure 14.2.

14.2.13.2 Global Bias Check

A global bias check compared the Nearest Neighbor (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 mineralized 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

Box plots were completed for the sub-celled model for each element comparing the 1 m composites, NN, ID3 and OK estimates by mineralized unit. Block selection was restricted to those classified as Indicated. Figure 14.18 displays the box plots for Pt within the T2 2 g/t 3PE+Au.

The sub-celled and regularized models were also compared (Figure 14.19).

The box plots for the UMT-TCU model show good agreement between 1m 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.

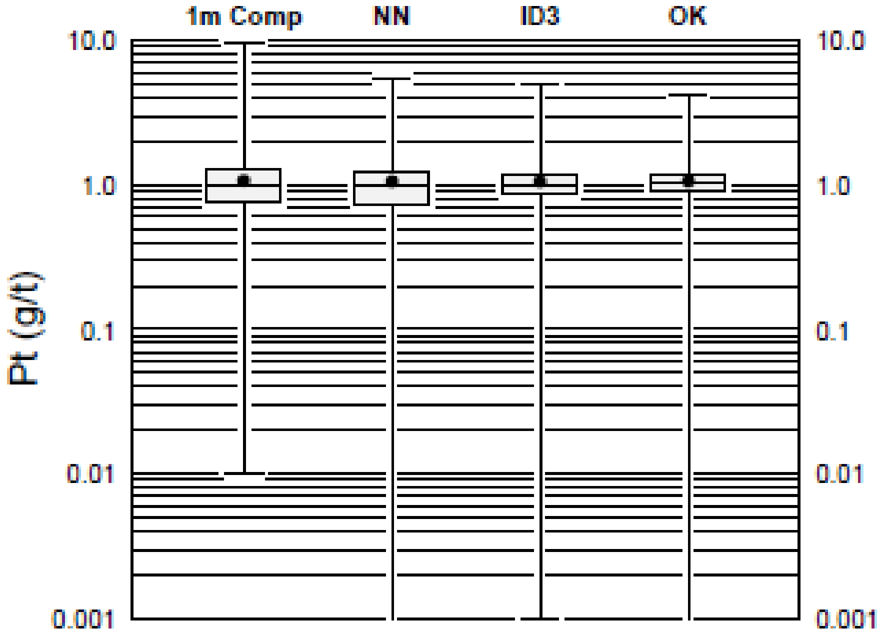
Overall, swath plots display reasonable comparisons between the ID3 estimates and their respective NN estimates; however, locally there are some differences, particularly in areas with limited drilling.

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%

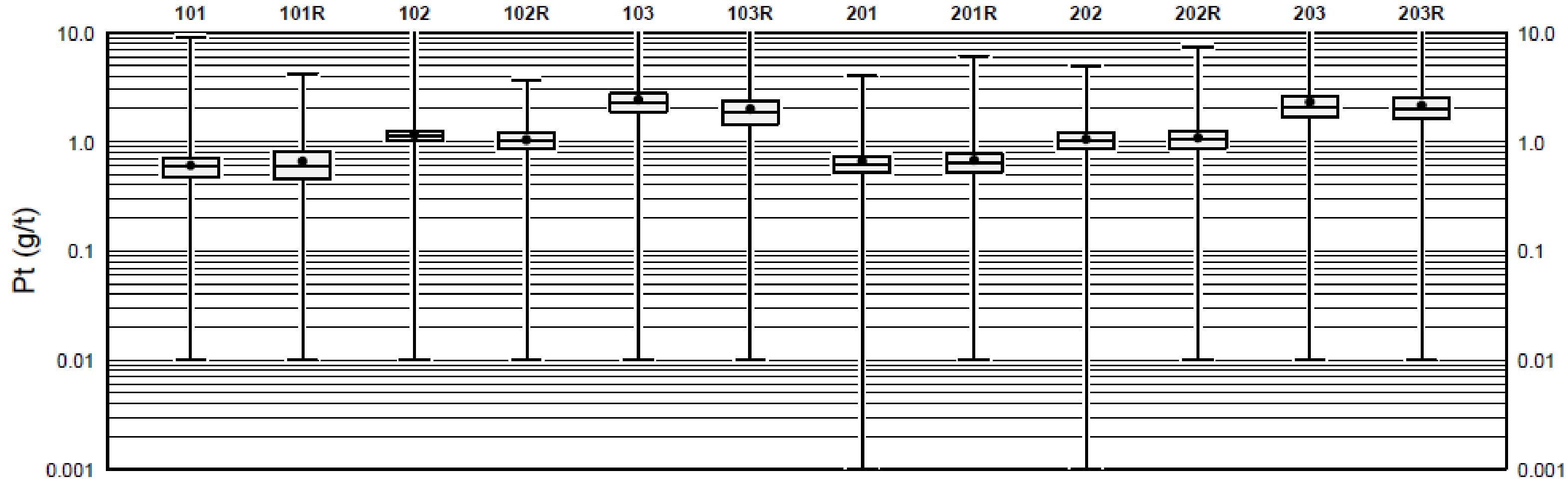
Figure 14.18 Box Plot of Pt for T2MZ 3PE+Au



	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 Regularized 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 regualrized 10 x 10 x 2 m blocks.

Figure 14.20 Platinum Swath Plot for T2MZ – 2 g/t 3PE+Au Shell; Regularized Model

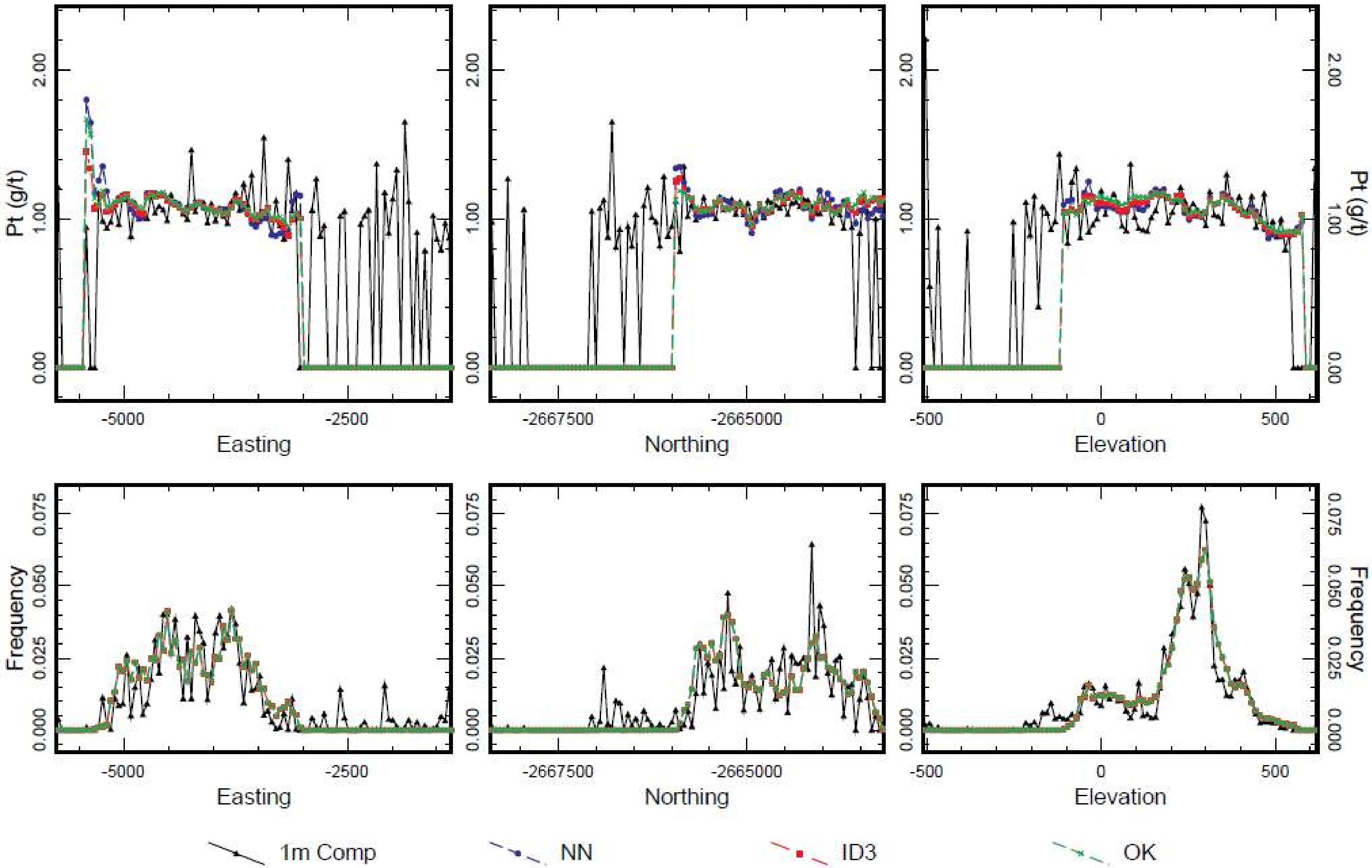


Figure prepared by Amec Foster Wheeler, 2016.

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 drill holes (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 contruct 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 Mineralized 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 (BT2MZ) to contrain the grade estimation. The nested grade shells were indentified from assay data using 1 g/t, 2 g/t and 3 g/t 3PE +Au cutoffs.

The grade-shell intercepts were coded into the drillhole database. The grade-shell drill hole intercepts were validated on dip and strike sections to ensure consistency. The grade shell drill hole intercepts were used to construct wireframes of the nested grade shells using stratigraphic surfacing functions in Leapfrog. Grade shell codes (GCODES) were used to code blocks within and outside the grade shells. The GCODES are summarised in Table 14.2.

14.3.5 Mineralisation Adjacent to the Bikkuri Mineralised Zones

There is scattered mineralisation locally adjacent to the B1MZ and B2MZ. Mineralisation adjacent to the Bikkuri mineralised zones may be included in future mine development, and a grade estimate is required for blocks within the Bikkuri model envelope.

14.3.6 Compositing and Exploratory Data Analysis (EDA) for UMT–BIK Model

The subset of drillholes used for the UMT-BIK resource was composited to 1 m length composites within the UMT-BIK model envelope. The compositing was controlled by the nested grade shells and the Bikkuri stratigraphic units.

EDA was completed using box plots, histograms, probability plots and contact profiles. Discontinuities in grade profiles near contacts suggested the grade shells and stratigraphic boundaries should be considered hard boundaries.

14.3.6.1 Rhodium Regressions

Rhodium analyses are only partially complete on the Bikkuri drillholes. Rhodium regressions for the Bikkuri drill data were used to address the missing rhodium data (See Section 14.2.6.1 and Table 14.3 and Table 14.4).

14.3.7 Block Model

The UMT-BIK block model includes two areas where the Bikkuri Reef has been interpreted (Figure 14.2). Blocks were oriented parallel to the national coordinate system. The block model used a parent block size of 10 m x 10 m x 2 m. Sub-celling was 10 m x 10 m x 0.5 m. The geological stratigraphic units and grade shells were coded to the blocks. After estimation, the final resource model blocks were regularised to 10 m x 10 m x 2 m block sizes.

14.3.8 Grade Estimation — UMT–BIK

14.3.8.1 B1MZ and B2MZ

Grade estimation in the B1MZ and B2MZ included block and composite matching by GCODE. To eliminate the effects of the structural blocks and variability in elevation, the center of individual stratigraphic units and mineralised zones were transformed to hang from the 1000 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 1000 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/ DH
	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 regularized to 10 m x 10 m x 2 m (no sub-cells) model blocks. The 10 m x 10 m x 2 m regularized 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	Maximum SG	Minimum 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 inFigure 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 mineralized 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%

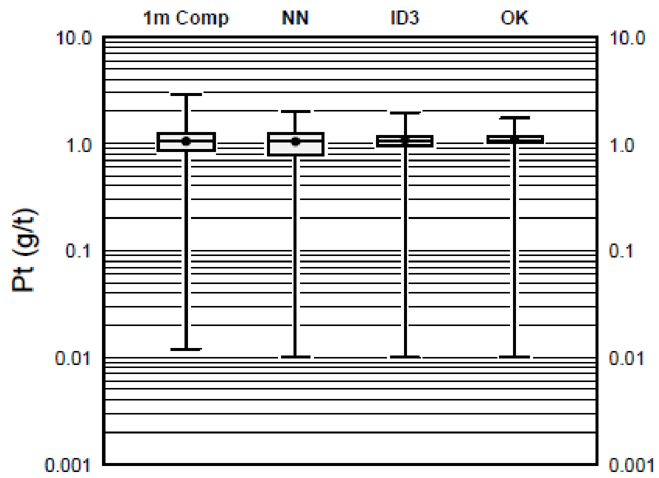
Note: Rows are shaded where the relative bias is outside (±5%)

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 14.21 displays 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 regularized models were also compared (Figure 14.22). The grades for the regularized model is commonly low due to the inclusion of low-grade blocks in the regularization process.

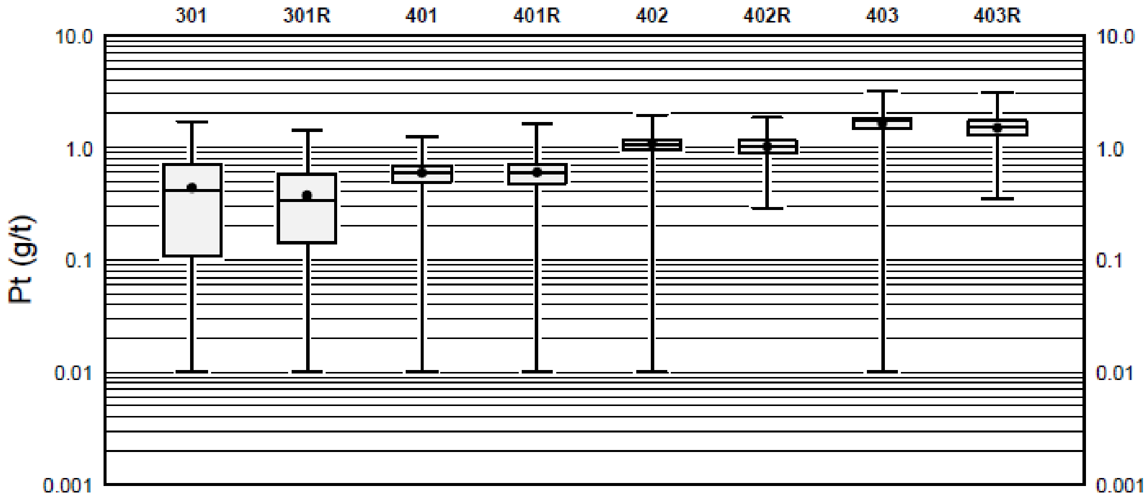
Figure 14.21 Box Plot Pt for B2MZ 2g/t 3PE+Au



	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-secitons 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 Blocks

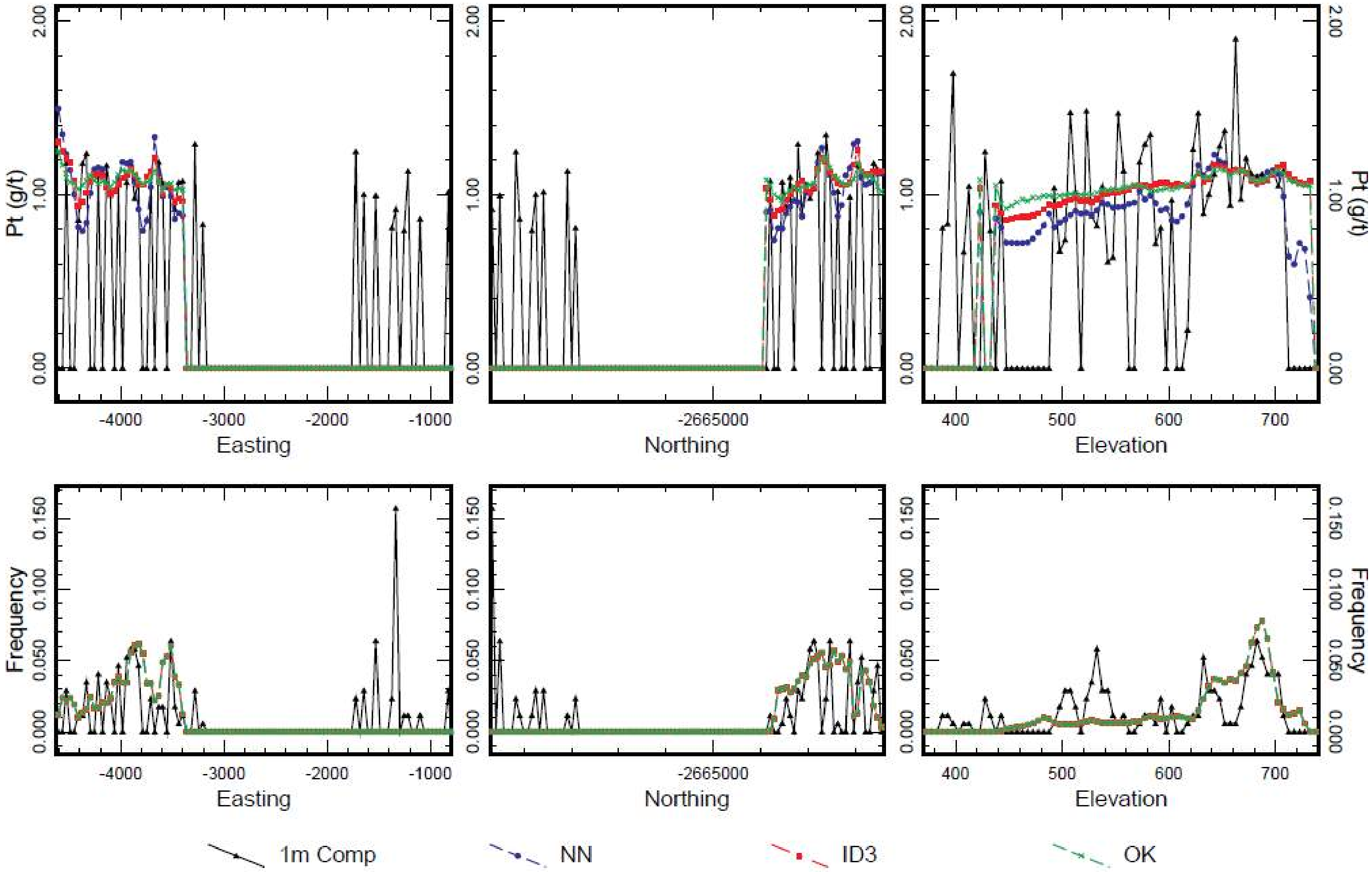


Figure prepared by Amec Foster Wheeler, 2016.

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 mineralization 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 fothe FWcpx and FWpnz domains. Additional footwall mineralisation is recognised, but the insufficient drillhole and sample data does not support resource modeling. 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 drill hole 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

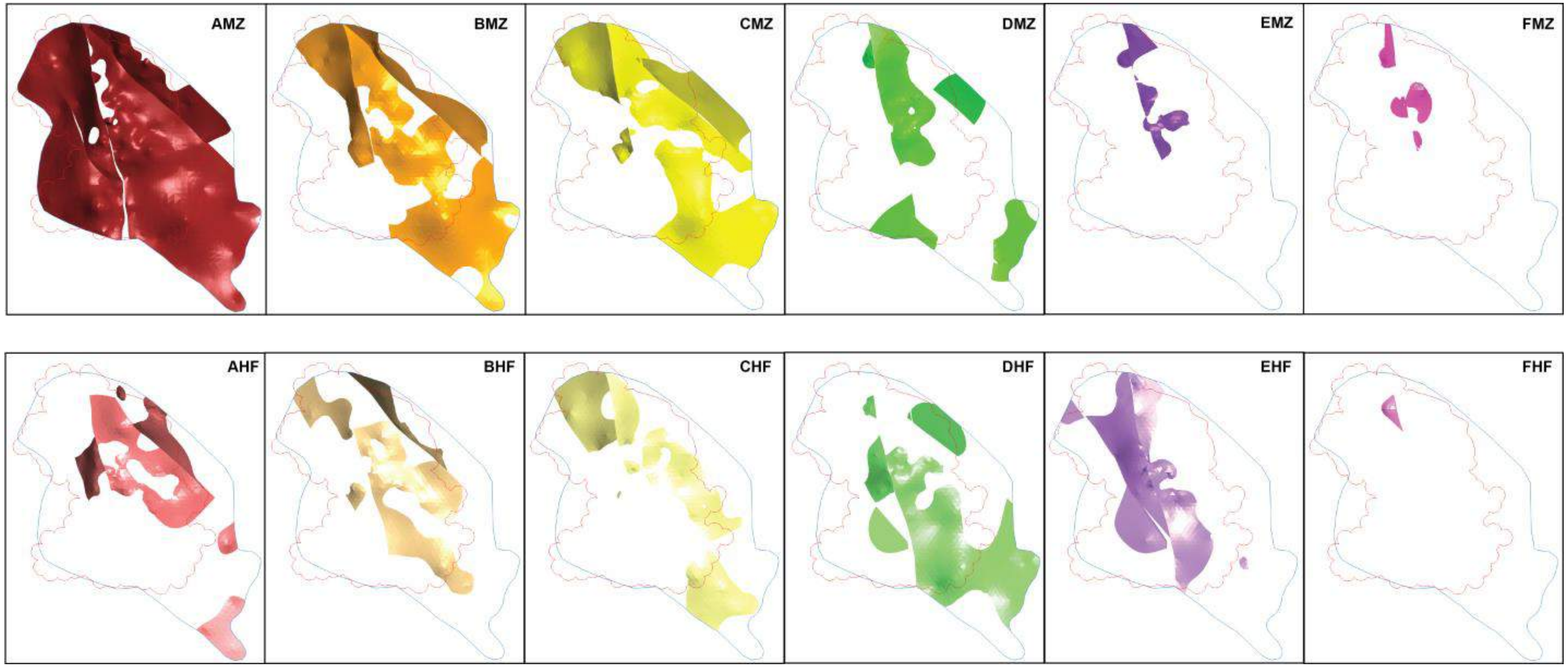
The PNZ is a unit of pyroxenite – norite composition that includes local unassimilated hornfels xenoliths. The mineralisation occurs predominantly as disseminated sulphides. Local massive sulphides are recognized at the contacts with the hornfels xenoliths. The 3PE+Au grades are commonly in the 1g/t 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 mineralization forms mineralized 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 decending order) have been identified in the FWpnz domain. Five minzones (BMZ, CMX, DMZ EMZ and FMZ in decending order) are identified between the hornfels xenoliths. The lateral extent of the correlated mineralization 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 Leafrog using the vein modelling functions. Model and composite coding is summarized 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 Statigaphic 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 center 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 1g/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 10m x 10m x 2m with subcelling to 5m x 5m x 0.5m (see Table 14.6) for better geological resolution. The PNZ model was permited to overwrite the TCU model below the lower boundary of the T2MZ.

Composites and model blocks were transformed to hang from the center of each lithology or MZ from the 1000 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 1g/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 15m and grade thresholds summarized in Table Table 14.19.

Table 14.20 Estimation Parameters for FW Model

Search Pass	Axis	Azimuth	Dip	Search Range	Min Samples	Max Sample	Max per Drill Hole
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-FWpnz Global Bias Check

MZ	Element	NN	ID3	Relative Difference
AMZ (311)	Pt	0.50	0.50	-0.25%
	Pd	0.55	0.55	0.49%
	Au	0.08	0.08	1.34%
	Rh	0.07	0.07	-0.49%
	Ni	0.19	0.19	1.68%
	Cu	0.11	0.11	0.71%
BMZ (312)	Pt	0.44	0.44	-0.12%
	Pd	0.58	0.56	-2.50%
	Au	0.09	0.09	-1.29%
	Rh	0.07	0.07	0.99%
	Ni	0.21	0.21	0.41%
	Cu	0.12	0.13	0.63%
CMZ (313)	Pt	0.45	0.47	4.81%
	Pd	0.65	0.66	2.47%
	Au	0.09	0.09	5.39%
	Rh	0.07	0.07	1.25%
	Ni	0.25	0.27	5.42%
	Cu	0.14	0.14	2.62%
DMZ (314)	Pt	0.60	0.59	-0.56%
	Pd	0.71	0.71	-0.02%
	Au	0.11	0.11	0.62%
	Rh	0.07	0.07	0.69%
	Ni	0.20	0.21	3.85%
	Cu	0.12	0.13	2.32%
EMZ (315)	Pt	0.18	0.19	2.41%
	Pd	0.11	0.11	2.01%
	Au	0.61	0.62	2.39%
	Rh	0.70	0.71	1.45%
	Ni	0.09	0.09	0.05%
	Cu	0.07	0.07	1.09%
FMZ (316)	Pt	0.18	0.19	3.34%
	Pd	0.12	0.13	2.27%
	Au	0.46	0.48	4.00%
	Rh	0.60	0.61	1.53%
	Ni	0.09	0.09	-0.01%
	Cu	0.07	0.07	-0.14%

Shaded rows indicate Relative Difference outside (±5)

14.4.8.1 Visual Ispection

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 indicates that grade continuity is best observed at a 1.0 to 1.5 g/t 3PE+Au cutoff). At a 2 g/t 3PE+Au cutoff, mineralization 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 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 regularized 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 mineralization has been modelled based on limited drill data, as not all of the UMT drill holes extended into the FW. For this reason, estimation of Mineral Resources has been restricted to the northwestern area of the Platreef Project where drill spacing is in the order of 100 m to 200 m. Similar mineralization has been seen in drill holes 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 cutoff. Discontinuous pods of mineralization at a 2.0 g/t 3PE+Au cutoff 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 mineralization that may permit mass mining methods at a lower cutoff (1 g/t 3PE+Au).

14.5 Final Platreef Mineral Resourcc 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 Rh, $1,500/troy ounce for Au, $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.

The process recoveries shown in Table 13.7 represent current metallurgical testwork and have approximately confirmed the recoveries used in Table 14.23 after adjustments are made for differences in head grades between the tables.

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.

Amec Foster Wheeler tested the Mineral Resources for reasonable prospects for eventual economic extraction. At a 2 g/t 3PE+Au cut-off grade, approximately 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. All of the blocks above a 1 g/t 3PE+Au cut-off generate an NSR of $10/t, meaning they will cover nearly all process, concentrate transport, and G&A costs.

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	1213	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	–	131	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 Grade (3PE+Au)t	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Cu (%)**	**Ni (%)**
3.0 g/t	2	1.67	1.45	0.34	0.09	3.55	0.22	0.40
2.0 g/t	7	1.30	1.16	0.28	0.07	2.81	0.19	0.35
1.0 g/t	31	0.73	0.71	0.16	0.05	1.65	0.14	0.24
Indicated Mineral Resources - Contained Metal								
Cut-off Grade (3PE+Au)t	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**
3.0 g/t		0.13	0.12	0.03	0.01	0.28	12	22
2.0 g/t		0.29	0.26	0.06	0.01	0.62	29	52
1.0 g/t		0.74	0.71	0.17	0.05	1.67	99	170
Inferred Mineral Resources -Tonnage and Grades								
Cut-off Grade (3PE+Au)t	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Cu (%)**	**Ni (%)**
3.0 g/t	8	1.59	1.52	0.36	0.09	3.55	0.20	0.37
2.0 g/t	27	1.23	1.17	0.25	0.07	2.72	0.16	0.30
1.0 g/t	112	0.75	0.76	0.15	0.05	1.72	0.14	0.24
Inferred Mineral Resources - Contained Metal								
Cut-off Grade (3PE+Au)t	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Cu (Mlbs)**	**Ni (Mlbs)**
3.0 g/t		0.41	0.39	0.09	0.02	0.92	35	65
2.0 g/t		1.09	1.03	0.22	0.06	2.40	100	184
1.0 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 Mineral Resources - Tonnage and Grades								
Cut-off Grade (3PE+Au)t	**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 Grade (3PE+Au)t	–	**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 Grade (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 Grade (3PE+Au)t	–	**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 mineralization that may permit mass mining methods at a lower cutoff.
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 Grade (3PE+Au)t	**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 Grade (3PE+Au)t	–	**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 cutoff. Discontinuous pods of mineralization 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 Grade (3PE+Au)t	**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	4.7	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 Grade (3PE+Au)t	–	**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 Grade (3PE+Au)t	**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	1431	0.88	0.94	0.17	0.07	2.05	0.13	0.25
Inferred Mineral Resources - Contained Metal								
Cut-off Grade (3PE+Au)t	–	**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 drill holes 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 drill holes 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 drill holes 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 drill holes 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

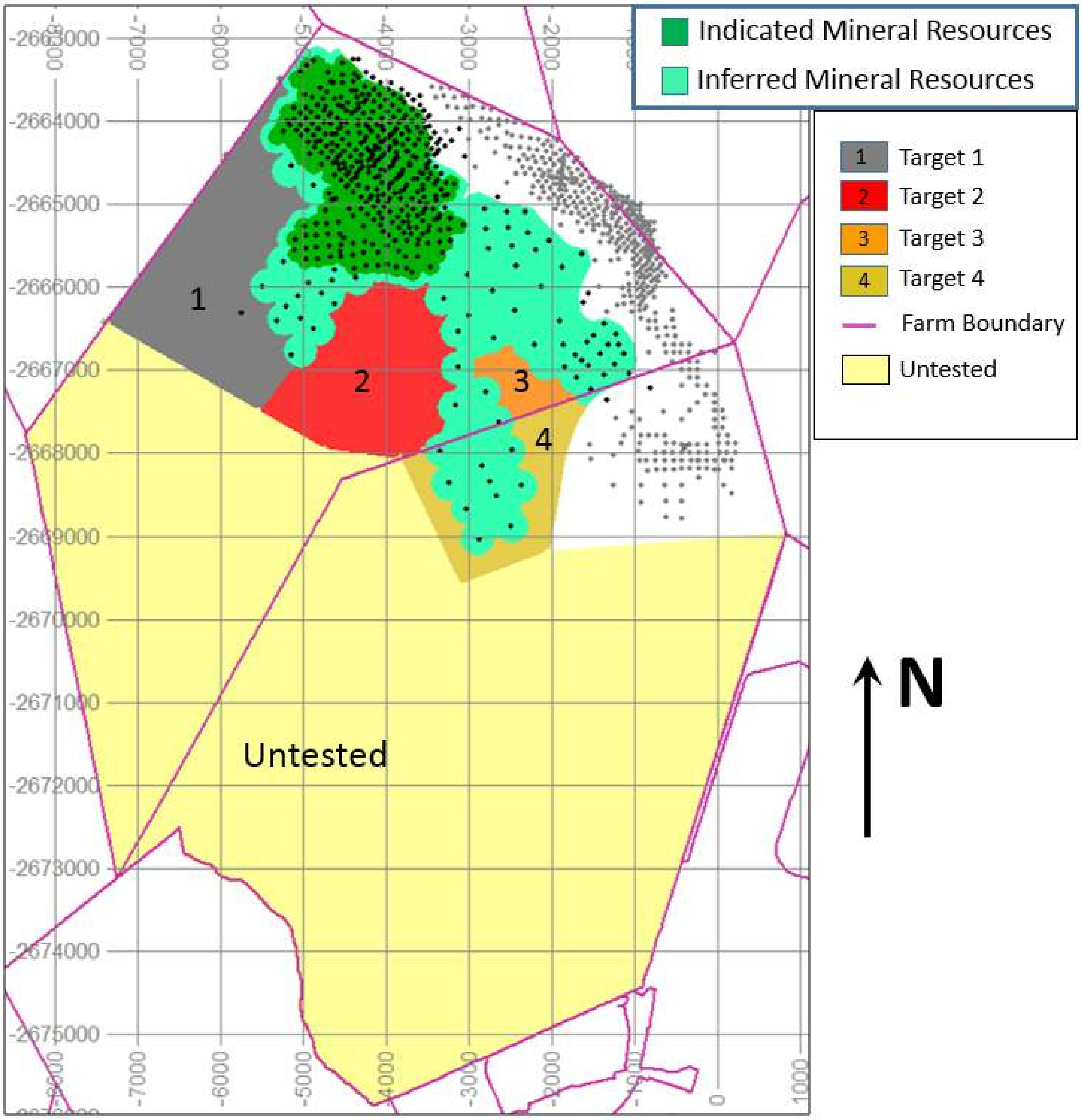


Figure prepared by Amec Foster Wheeler, 2016.

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 recognized a definitative 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 mineralized materal.

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

This section has not been changed from the Platreef 2014 PFS and remains the most currrent study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

The mineral reserve estimate for Platreef is based on the resource block model (file name pmbdt13a.bmf) that was provided to Stantec by Ivanhoe. Only Indicated resources from the block model were used for determination of the probable Platreef mineral reserve. The mineral resource block model also includes a net smelter return (NSR) variable. NSR is the dollar value of the metals recovered from a tonne of ore, less the cost for concentrate transport to the smelter, smelting and refining charges, and other deductions at the smelter. The NSR does not consider the cost for mining, milling, or general and administration (G&A). In order for mining of a resource block to be economical, the NSR value must be high enough to cover these costs. NSR calculation formulas and metal prices used in the block model were provided by Ivanhoe. The metal prices used in the NSR calculations are summarised in Table 15.1.

Table 15.1 Metal Prices Used in Net Smelter Return Calculations

Metal	Selling Price
Au (US$/oz)	$1,315.00
Pd (US$/oz)	$667.00
Pt (US$/oz)	$1,699.00
Rh (US$/oz)	$1,250.00
Cu (US$/lb)	$2.73
Ni (US$/lb)	$8.81

Mineral reserves were calculated from the resource model using a combination of generated grade shells for designing stopes, which were based on the economic NSR cut-off values and Stope Optimiser software from Alford Mining Systems. Three stoping methods (Longhole, Drift-and-Bench, 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 per tonne to $58.53 per tonne, excluding capital recovery and profit margin. For the production schedule and mineral reserve, a declining cut-off was chosen. A $100 NSR cut-off value was used in defining these reserves in order to increase the initial mill head grade and to shorten the payback period. An $80 NSR cut-off value was used later in the mine life, as the higher-grade reserves deplete and mining progresses further from the production shaft. 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. The minimum NSR cut-off grades estimated are provided in Table 15.2.

Table 15.2 Minimum Net Smelter Return Cut-off Grades

Mining Method	NSR Cut-off ($/tonne)
Longhole Stoping	$47.71
Drift-and-Bench Stoping	$52.80
Drift-and-Fill Stoping	$58.53

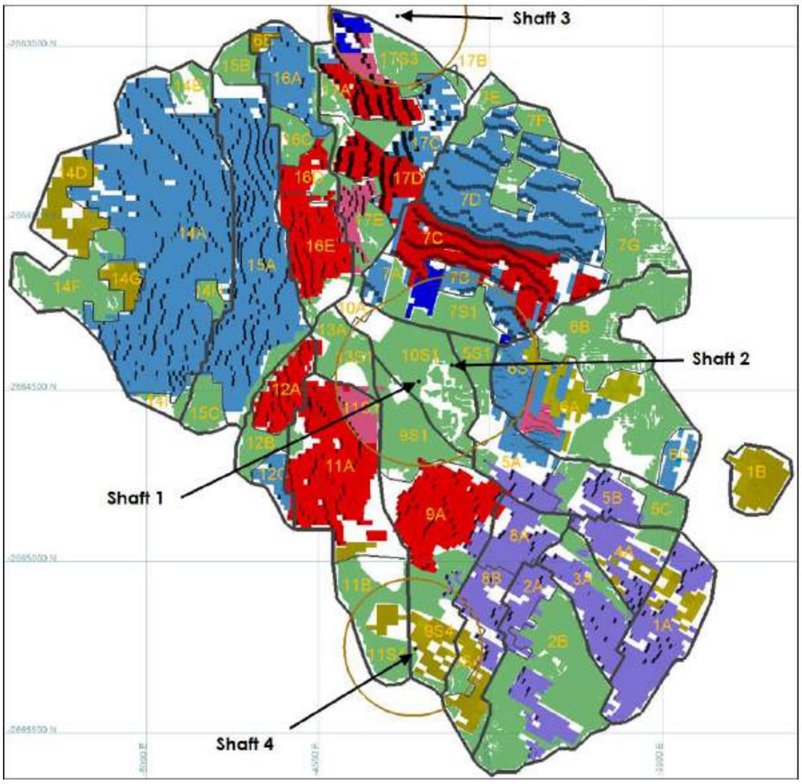
A definitive mine plan based on detailed stope layouts supports the mineral reserve. Stope Optimiser software was used to generate longhole stopes and bench stopes, while Vulcan software NSR grade shells were used to guide the design of drift-and-fill slices.

Due to irregularities in the geometry of the mineralised zones, not all cut-off grade material 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. Dilution and recovery factors that were applied to the resource are discussed in Section 16.2.5.

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. The SRK report divides the deposit into five major geotechnical zones, with recommendations for the best stope orientation within the 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, shown in Figure 15.1. This figure also shows the mineral reserve zones and sub-zones included in Stantec's mineral reserve estimates.

Figure 15.1 Platreef Mineral Reserve Zones



Probable Mineral Reserve

Red: > $100 longhole Stopes
Blue/violet: > $80 longhole stopes
Brown: > $80 bench
Green: > $80 cut and fill

These stope shapes were then used to query the block model and report tonnage and grades within the shapes. The tonnage and grades were then input into an Excel workbook to apply dilution and mining recovery factors on a stope-by-stope basis to all areas defined for use in production sequencing.

The variability of mining, metallurgy, infrastructure, permitting, and other factors 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 ($100 and $80) will provide protection from future negative impacts of these factors.

Table 15.3 and Table 15.4 show the total Probable Mineral Reserve for Platreef.

Table 15.3 Platreef Probable Mineral Reserve – Tonnage and Grades as at 8 January 2015

Method	Tonnage (Mt)	NSR ($/t)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)
Longhole Stopes	106	133.5	1.73	1.86	0.25	0.12	3.97	0.16	0.32
Drift-and-Fill	10	144.3	1.99	1.95	0.29	0.13	4.36	0.14	0.30
Drift-and-Bench	5	146.4	1.95	2.01	0.28	0.14	4.38	0.15	0.32
Total	**120**	**134.9**	**1.76**	**1.87**	**0.26**	**0.13**	**4.01**	**0.15**	**0.32**

1. Metal prices used in the reserve estimate are as follows: Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $1,250/oz, Cu: 2.73/lb, and Ni: $8.81/lb.
2. Tonnage and grade estimates include dilution and recovery allowances.
3. A declining NSR cut-off of $100/t–$80/t was used in the mineral reserve estimates.
4. Total may not add due to the rounding.
5. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)

Table 15.4 Platreef Probable Mineral Reserve – Contained Metal as at 8 January 2015

Method	Tonnage (Mt)	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Cu (Mlb)	Ni (Mlb)
Longhole Stopes	106	5.88	6.33	0.86	0.42	13.49	362	758
Drift-and-Fill	10	0.63	0.62	0.09	0.04	1.39	30	65
Drift-and-Bench	5	0.28	0.29	0.04	0.02	0.64	15	32
Total	**120**	**6.80**	**7.24**	**0.99**	**0.49**	**15.51**	**408**	**855**

1. Metal prices used in the reserve estimate are as follows: Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $1,250/oz, Cu: 2.73/lb, and Ni: $8.81/lb.
2. Tonnage and grade estimates include dilution and recovery allowances.
3. A declining NSR cut-off of $100/t–$80/t was used in the mineral reserve estimates.
4. Total may not add due to the rounding.
5. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)

15.1 Conclusion

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

This section has not been changed from the Platreef 2014 PFS and remains the most currrent study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

SRK Consulting (South Africa) (Pty) Ltd (SRK) was requested by Platreef Resources (Pty) Ltd (Ivanhoe) to undertake a PFS-level mining geotechnical investigation for its Platreef Project in the Limpopo Province of South Africa. A detailed geotechnical report was compiled for the PFS (SRK Report 458213/1). A summary of SRK's report is provided in this section.

The primary aim of the PFS-level investigation was to increase the confidence level of the current geotechnical database as well as to undertake numerical analyses, based on data from the actual mine site, to optimise the mine design going forward.

This section describes the nature and quality of the sub-surface conditions as represented by drilled core from boreholes over the Platreef area. The nature and quality of the ground conditions were derived from data assimilated from the geotechnical core logs, as well as from laboratory testing results. The design of the stopes and access development, as well as the associated support requirements are summarised.

The following work programme was carried out:

- An initial visit to the Platreef Project site was conducted from the 23–24 May 2013 by SRK representatives to review logging carried out by Ivanhoe's geotechnical staff and to plan the study programme.
- Subsequent QA/QC visits to the Platreef Project site were undertaken from 29–31 May 2013 and from 3–7 June 2013.
- A memorandum (Memo 1) discussing the findings of the first three QA/QC visits was submitted to Ivanhoe on 11 June 2013.
- Laboratory test results collated from programmes prior to the PFS were analysed and summarised in June 2013.
- A structural analysis and creation of structural domains took place in June 2013.
- Drilling of additional geotechnical holes commenced in June 2013.
- Data analysis and determination of Rock Mass Ratings (RMR) and Barton's Q' values were determined in July 2013.
- Empirical geotechnical stope design was carried out in July 2013.
- A memorandum (Memo 2) comprising preliminary mine geotechnical parameters was submitted to Ivanhoe on 10 July 2013, and revised twice upon receiving feedback.
- A memorandum (Memo 3) consisting of shaft reef extraction trade-off information was submitted to Ivanhoe on the 8 August 2013.
- Laboratory test results from PFS boreholes GT013 and GT014 were received from Ivanhoe in August 2013.

- Geotechnical borehole drilling was completed in November 2013, with the exception of GT007.
- PFS borehole logging data was received from Ivanhoe in November 2013.
- The final QA/QC visit to the Platreef Project site for the PFS was undertaken from 11–13 November 2013.
- A memorandum (Memo 4) comprising the findings from the last QA/QC visit was submitted to Ivanhoe on 14 November 2013.
- Drilling of GT007 was completed in December 2013.
- Addition of PFS logging data was adapted to the structural domains, RMR, and Q' values; and the stope design in January 2014.
- Detailed analyses and sensitivity analyses of the failure potential of the mine stopes were undertaken in January 2014.
- A memorandum (Memo 5) responding to the peer review by Mike Sandy on the Platreef Scoping Study Geotechnical Review Report was submitted to Ivanhoe on 30 January 2014.
- Compilation of the PFS report was completed in February 2014.

16.1.1 Geotechnical Investigation

Geological Information Provided

Geological surfaces were received from Ivanhoe in May and June 2013. Data was subdivided into stratigraphy zones, mineralised zones with a fault plane, and mineralised zones without a fault plane.

SRK was also supplied with a geological model by Ivanhoe. This orebody model is based on the block model received from Ivanhoe using a net smelter return (NSR) cut-off approach.

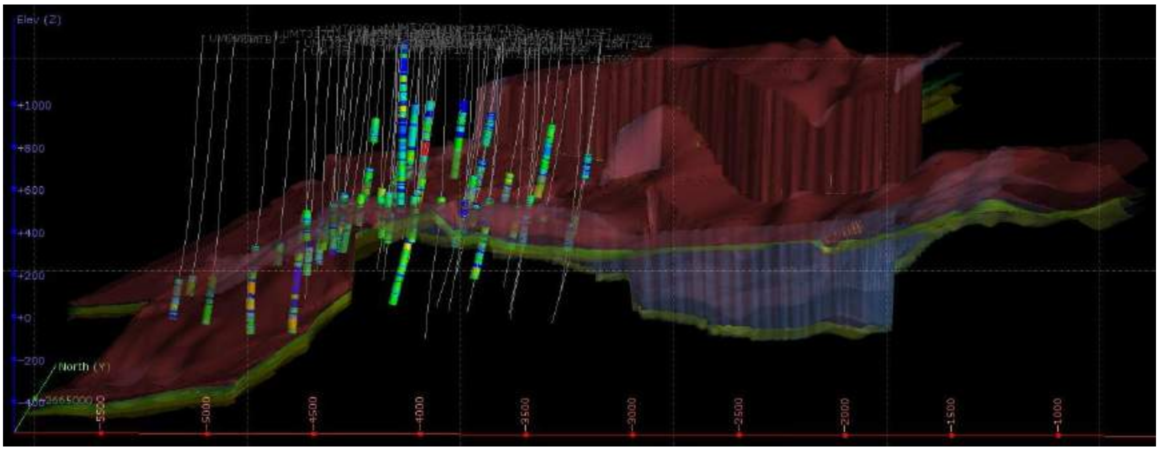
The geological drillhole data was also provided in comma delimited text (CSV) and GIS-compatible shape file formats. The survey coordinates of drillholes were also provided.

Geotechnical Data Collection

Geotechnical, geological, and geophysical holes drilled for previous programmes in the area were collated, and 59 of the most representative data were selected for inclusion in the analyses. In addition to these 59 drillholes, five PFS holes were drilled and included in the study.

These drillholes were logged by Ivanhoe, MSA Group (MSA), or SRK representatives for the various programmes. A QA/QC programme was adopted to ensure that the logging data remained consistent across all the drillholes. The block model was used to define the hanging wall, mineralisation, and footwall zones for the design. Drillholes that intersected the geological model were selected for further analysis (Figure 16.1 and Figure 16.2).

Figure 16.1 Platreef Drillholes Intersecting Geological Model



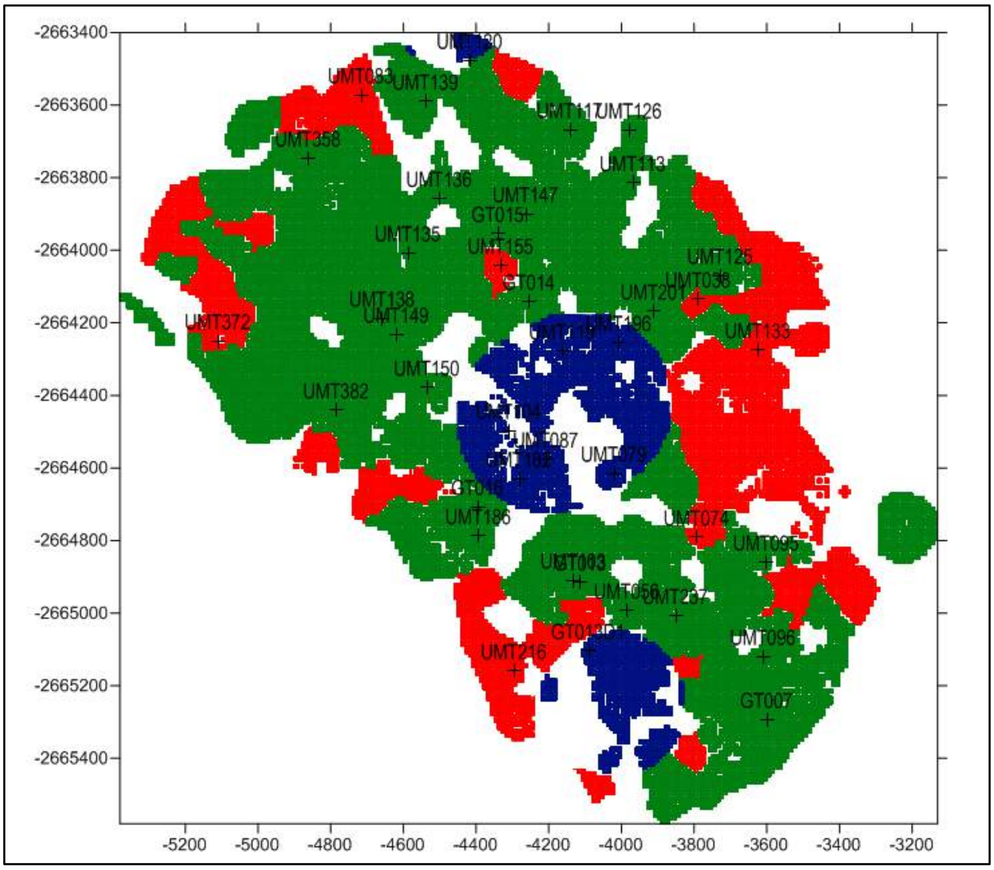
Geophysical drillholes considered for the structural analysis included those that contained reasonably accurate drill core orientation data that could be correlated with both the Acoustic Televiewer (ATV) data and local structural trends. Geophysical drillholes (orientated and ATV) considered for structural analysis are presented in Table 16.1.

Table 16.1 Geophysical Drillholes

GT004	UMT096	UMT252	GT013‡	GT008	UMT130	UMT361
GT004A	UMT103	UMT257	GT013D1‡	UMT038	UMT146	UMT371
GT005	UMT105	UMT357	GT014‡	UMT077	UMT149	UMT372
GT006	UMT113	UMT357D1	GT015D1‡	UMT087	UMT196	UMT382
GT007	UMT123	UMT358	GT016‡	UMT095	UMT216	GT012‡
UMT063	UMT066					

‡ PFS Drillholes

Figure 16.2 Drillholes Intersecting Geological Model



Laboratory Test Data

Laboratory testing results from the PFS, scoping study, shaft investigation, MEng (Master of Engineering testing by A. Cooper) and underground (UG Mine) testing regimes were collated and utilised, in addition to two drillholes for the PFS study (Table 16.2).

Table 16.2 Drillholes with Laboratory Testing Results

MENG	PFS	Scoping	Shaft #1	UG Mine
UMT078	GT013	UMT100	GT008	UMT075
UMT341D2	GT014	UMT109A		UMT382
		UMT123		UMT385
		UMT130		

These tests were separated into those that fell into the hanging wall, mineralisation, and footwall. The total number and type of geomechanical tests conducted are presented in Table 16.3.

Table 16.3 Laboratory Test Summary

Type of Laboratory Test	UCM	UCS	TCS	BTS
Total Number of Tests	81	108	120	104
Hanging wall	47	34	72	38
Mineralisation	21	5	21	27
Footwall	13	57	27	39

UCM Uniaxial Compressive Strength with Young's Modulus and Poisson's Ratio
UCS Uniaxial Compressive Strength
TCS Triaxial Compressive Strength
BTS Brazilian Indirect Tensile Strength Test

On the 23 May 2013 SRK commenced with the QA/QC visits to the Platreef Project site in Mokopane. The site visits were conducted by SRK personnel on the following dates:

- 23–24 May 2013.
- 29–31 May 2013
- 3–7 June 2013
- 11–13 November 2013

Overall, 46 geotechnically/geologically logged holes, 13 geophysical (ATV) holes and three additional holes that were logged at site were made available to SRK. Thirty of the 62 holes were selected for the QA/QC audit. The drillholes that were chosen for the audit are presented in Table 16.4 and Table 16.5.

Table 16.4 List of QA/QC Audited Geotechnically/Geologically Logged Drillholes

Geotechnical/Geological Drillholes					
UMT069	UMT125	UMT230	UMT358	UMT382	GT014
UMT079	UMT133	UMT244	UMT361	GT012	GT015
UMT080	UMT136	UMT357	UMT371	GT013	GT016
UMT117	UMT189	UMT357D1	UMT372	GT013D1	

Table 16.5 List of QA/QC Audited ATV Logged Drillholes

Geophysical Holes (ATV)		
UMT38	UMT96	UMT196
UMT56	UMT105	UMT216
UMT77	UMT113	UMT252
UMT87	UMT149	UMT095*
		UMT103*

* denotes that the core was also geotechnically logged

In SRK's opinion the quality of drillhole logging was generally good.

The findings of the QA/QC visits are summarised as follows:

- The joint set counts carried out by Ivanhoe's geotechnical staff correlates well with the quality control count and were within an acceptable range.
- The microscopic joint properties identified correlate well and are within an acceptable range.
- Serpentine was originally described as a non-softening fine material; however, it was later corrected to a softening fine material. The ratings for joints with serpentine infill were thus adjusted accordingly.
- The joint orientation data logged by Ivanhoe's sub-contractor (MSA) were well within an acceptable range.
- Geotechnical domains are allocated by different attributes; one such attribute is geology. As the geological interpretation has been revised by Platreef, it was found that boreholes geologically logged prior to 2013 have been sub-divided differently to those logged in 2013. However, this is expected to have a minor impact on the geotechnical rock mass classification.

Geophysical drillholes chosen for analysis were limited to those that contained data of a satisfactory quality. Data used in the scoping study was considered poor quality mainly because the core orientation was done on vertical holes. Holes that were too deep or too shallow were excluded. It was also observed that an unnatural systematic pattern was produced for the DIPS plots for PFS drillholes GT015D1 and GT016. SRK was unable to establish the reason for this, and the data from these boreholes was consequently not included in the analysis.

Rock Properties

This section describes the determination of rock properties from the collated laboratory testing programmes.

The uniaxial compressive test results were determined for the following purposes:

- Empirical stope design.
- Variation of the rock strength for the risk analysis.
- Determination of Hoek-Brown parameters for non-linear modelling to be conducted in the feasibility study.

The material properties were obtained from the available strength (unconfined compressive strength (UCS), Brazilian and triaxial) test results. The intact rock properties were estimated by fitting Hoek-Brown failure envelope to the laboratory test results.

The design rock properties determined for the 10 m hanging wall, mineralisation, and footwall are presented in Table 16.6. A summary of the elastic properties determined for the hanging wall, orebody and footwall are presented in Table 16.7.

Table 16.6 Design Rock Strength Properties

	Hanging wall	Mineralisation	Footwall
Intercept	36,301	27,596	34,289
Slope	2,237.8	1,901.1	2,404.1
UCS	191	166	185
UCS Std Dev	33	33	44
UCS Mean -	157	133	141
UCS Mean +	224	199	230
UTS	16	15	14
mi	11.7	11.4	13.0

Table 16.7 Intact Elastic Material Properties

	Density (t/m^3)			Young's Modulus (GPa)			Poisson's Ratio		
	H/W	**Min**	**F/W**	**H/W**	**Min**	**F/W**	**H/W**	**Min**	**F/W**
Mean	**2.94**	**2.92**	**2.98**	**98**	**61**	**123**	**0.30**	**0.33**	**0.30**
Standard Deviation	0.22	0.05	0.20	23	9	9	0.04	0.04	0.04
Mean –Std. Dev.	2.72	2.87	2.78	75	52	115	0.26	0.29	0.26
Mean +Std. Dev.	3.16	2.96	3.18	122	69	132	0.34	0.37	0.34
Minimum	2.52	2.82	2.63	52	54	106	0.22	0.27	0.27
Maximum	3.31	3.02	3.23	139	75	134	0.37	0.38	0.36
Samples	136	14	44	30	5	7	30	5	7

Rock Mass Classification

To classify the quality of the rock mass, use was made of two rock mass classification systems, (i) Laubscher's (1990) Mining Rock Mass Rating (RMR) Classification System, and (ii) Barton et al's (1974) Norwegian Geotechnical Institute's Q-System.

All geotechnical drillholes were logged using the SRK standard logging sheet, which is adapted to suit the determination of both Laubscher's RMR and Barton's Q values from the logging data. The Q-System was adopted to facilitate the derivation of Q' values for the stope design and the determination of development support recommendations.

Laubscher's RMR values were determined for the verification and validation of the Barton Q values and are not presented here. The Geological Strength Index (GSI) was also determined for the purposes of obtaining rock mass parameters for non-linear modelling, which will be conducted at the feasibility level of study, but is not presented here.

Barton's Q-System was utilised to facilitate the derivation of Q values for the rock mass. Q is obtained using the following expression:

Q = (RQD/Jn) x (Jr/Ja) x (Jw/SRF)

Where: RQD is the rock quality designation
Jn is the joint set number
Jr is the joint roughness number
Ja is the joint alteration number
Jw is the joint water reduction factor
SRF is the stress reduction factor

The Q' value, which excludes the effects of stress and water, represents the rock mass characteristics. Q' was determined for each geotechnical interval in each drillhole as follows:

Q' = (RQD/Jn) x (Jr/Ja)

RQD was estimated using both the original method by Deere and a method proposed by Palmstrom (1974). It was decided that Palmstrom's approach would be used in the analysis.

The joint set number (Jn) values ranged from 1–9, with 1 being massive, with few joints and 9 being three joint sets. The joint roughness (Jr) values varied between 0.5–3, with 0.5 being polished and 3 being rough. The joint alteration (Ja) values varied from 0.75–4, with 0.75 being no alteration and 4 having a clay coating.

The weak joints, have a Jr rating of 1 and a Ja rating of 4 and make up about 8% of the logged intervals.

To obtain an indication of the distribution rock mass conditions in the hanging wall, mineralisation, and footwall zones, it was necessary to use weighted averages and contouring. However, Q' values are expressed on a logarithmic scale and is therefore difficult to statistically analyse and display on contours. Therefore, the Q' value was converted to Bieniawski's RMR for the purpose of statistical analysis and contouring. The relationship between Bieniawski's RMR and Q is as follows:

RMR = 9 ln.Q + 44

Weighted average RMR values were determined for the hanging wall, mineralisation, and footwall zones in each of the drillholes. RMR contour plots, which indicate the spatial distribution of rock mass quality, are presented in Figure 16.3. These show that the weighted average RMR values determined are fairly consistent with the range of values falling within one class of 60–80 (good quality rock). Most values lie between 65–70. It was therefore not considered necessary to define geotechnical domains on the basis of rock mass quality.

The 20th percentile of the weighted average RMR values, calculated per drillhole per zone, and converted back to Q', was used for design purposes. It is too conservative to use minimum Q' values for the general design, and using mean values would imply that it only caters for 50% of the ground conditions encountered. Summaries of the Q' values for the hanging wall, mineralisation, and footwall are presented in Table 16.8.

Table 16.8 Rock Mass Quality (Q')

	Hanging wall			Min	Footwall		
	0–2 m	0–5 m	0–10 m		0–10 m	10–20 m	20–50 m
Mean	14.1	14.0	13.7	15.8	13.8	16.3	16.9
Minimum	4.8	3.7	2.9	5.4	5.4	4.6	6.3
Maximum	70.6	70.5	100.0	89.0	54.4	57.3	98.5
20th Percentile	*8.4*	*8.6*	*8.6*	*9.9*	*7.9*	*7.4*	*9.2*
No. of Drillholes	34	35	34	32	35	35	8.7

The standard deviations are very low (less than 8% of the mean), which confirms the lack of variability. The 20th percentile values do not differ significantly for the different hanging wall zones, which indicates that it is not critical which of these zones is used. It is apparent the footwall ratings improve slightly with greater depth in the footwall.

It should be noted that there are intervals that contain shear zones in the geotechnical database. It is believed that these are structural weaknesses and are therefore taken into account by considering the joint orientations in the structural analysis. These sheared intervals are too short to significantly influence the weighted average RMR. A summary of the shear zones is presented in Table 16.9. The risk of chromite stringers present in the hanging wall was also assessed; these do not seem to pose a great risk as the majority of stringers are present high up in the hanging wall. This will be looked at in more detail during the Feasibility Study.

Table 16.9 Summary of Shear Zones

Area of Interest	Hanging wall	Mineralisation	Footwall
Intervals with Shear Zones	7	33	36
Percentage of Total Geotechnical Intervals	11%	17%	9%
Minimum Length (m)	0.10	0.05	0.06

Figure 16.3 Rock Mass Rating (Q converted) Distribution in Plan

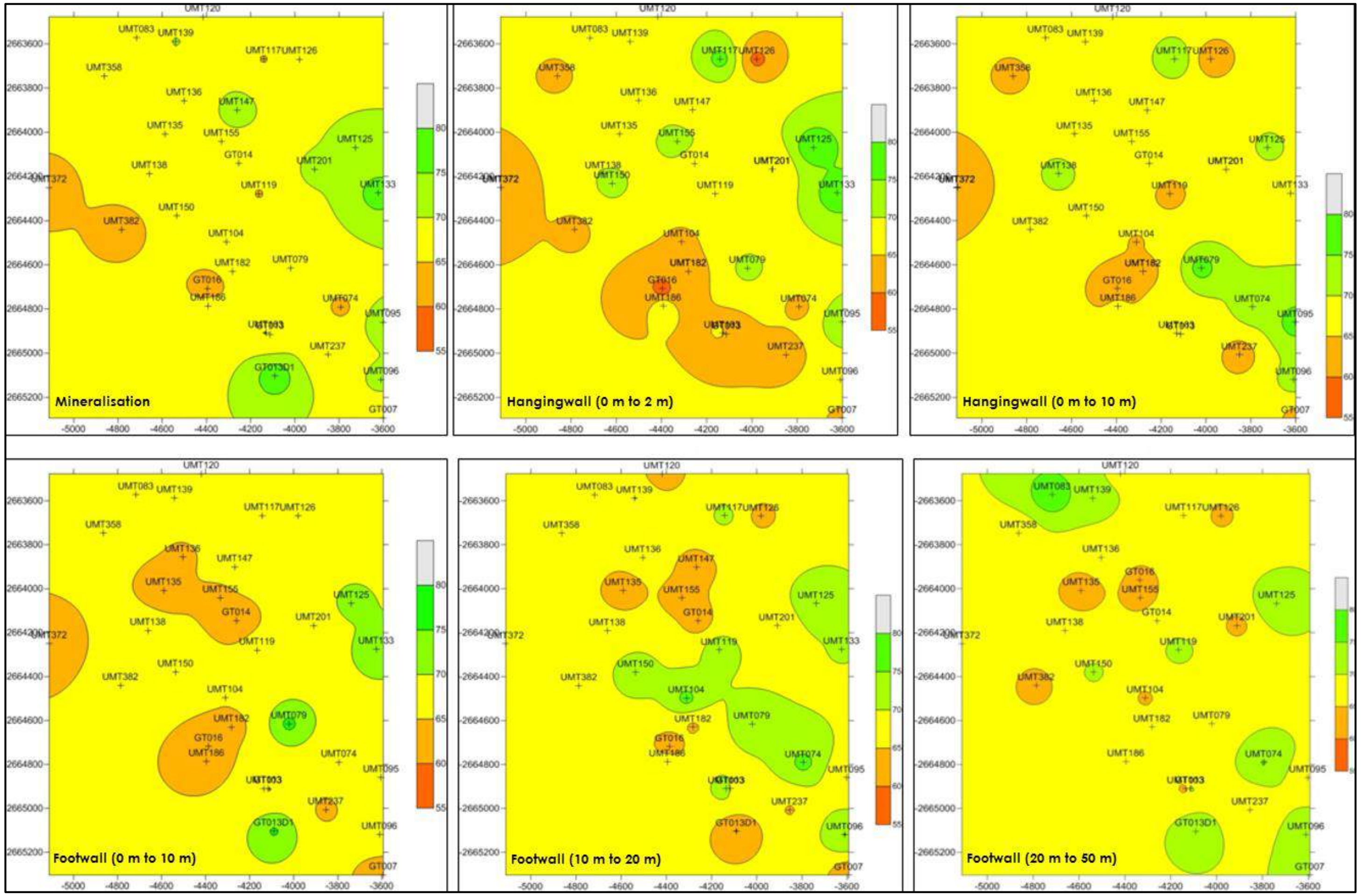


Figure by SRK 2014

Structural Analysis

The structural model provided by Ivanhoe was used to assist with dividing the area into structural domains. Joint orientation data from each drillhole was plotted individually on a stereographic projection.

Five structural domains have been identified for the Platreef Project. These are presented in Figure 16.4.

A summary of the domain joint orientations is presented in Table 16.10.

Table 16.10 Domain Joint Operations

Structural Domain	Mean Joint Orientations			
	Joint Set 1	Joint Set 2	Joint Set 3	Joint Set 4
1	40/085	49/172	86/304	
2	46/067	39/148		
3	23/054	51/066	63/104	47/235
4	16/075	50/066	64/163	
5	44/239	55/322		

Additonal joint data are required for the JBlock kinematic analysis. The standard deviation for joint orientation, true joint spacing and friction angle has been calculated for each joint set identified but are not present in this report.

Figure 16.4 Structural Domains

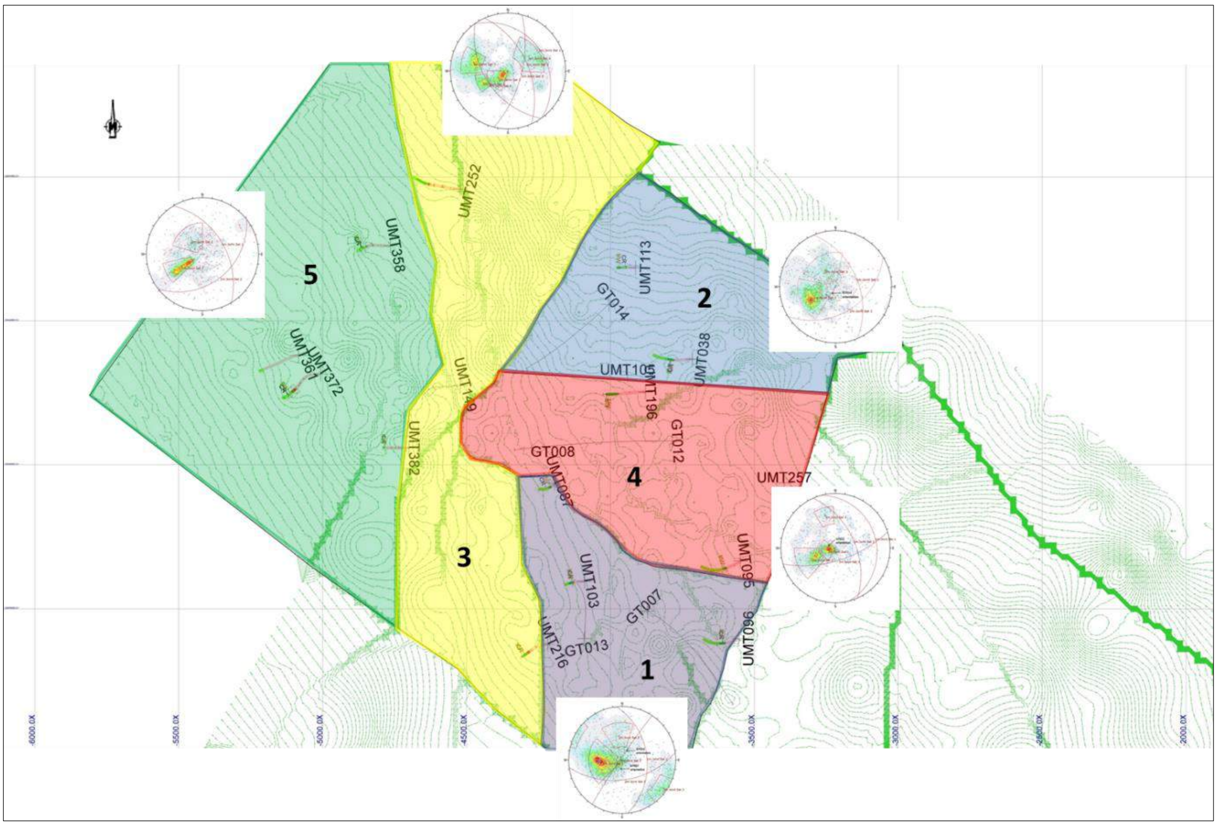


Figure by SRK 2014

16.1.2 Analysis and Design

Numerical Modelling of Stopes

The main objective of the numerical modelling was to investigate the maximum stress likely to develop around the stopes throughout the life of mine.

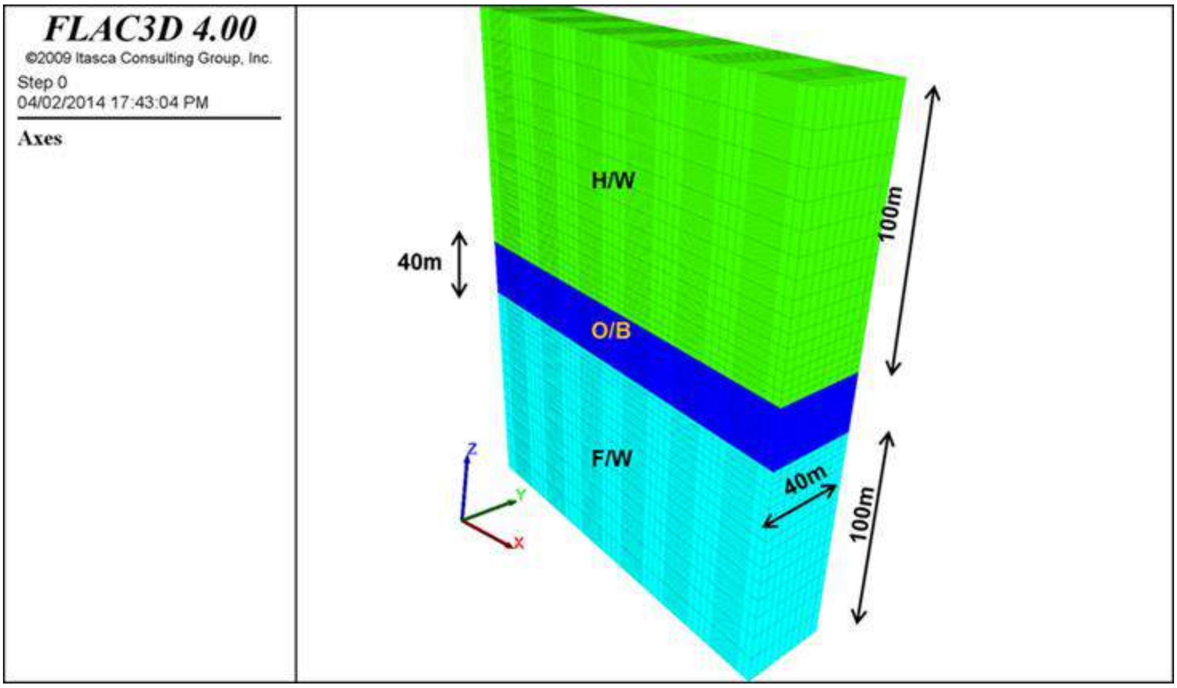
Based on data collated by Stacey & Wesseloo (1998), the stress conditions in many Bushveld Igneous Complex (BIC) operations are characterised by a high ratio of horizontal-to-vertical ground stresses (K ratio), particularly at shallow depth. However, at 1,000 m depth, it has been found that the K ratio in the BIC is more typically 1.0. The expected K ratio is therefore1.0, but since no stress measurements have been conducted, this is uncertain. A sensitivity analysis to K ratios during numerical modelling was considered essential to investigate the effect of the uncertain virgin stress.

Model Construction

Three-dimensional elastic numerical modelling was conducted. Figure 16.5 shows the conceptual geometry. The model geometry comprises the following components:

- 100 m hanging wall (H/W)
- 40 m and 25 m mineralisation (O/B), which represent the maximum and the average target stoping heights, respectively
- 100 m footwall (F/W)
- 40 m out-of-plane dimension (the model actually represents an infinite length).

Figure 16.5 FLAC3D Conceptual Model used to Estimate Evolution of Stress in the Stope Backs and Walls



Roller boundary conditions applied all-round the geometry sides and the base, which imply that the geometry represents effectively multiple instances of 12 infinitely long stopes. This assumption was considered valid since in most instances the length of the stopes is expected to be over three times greater than its cross section dimensions, which should result in negligible confinement from the abutment faces.

The principal stress directions were assumed to be parallel and perpendicular to the stope orientation. K_0 is oriented perpendicular to the stope orientation, while K_{90} is oriented parallel to the stope. The ratio of the two horizontal stresses was also varied during the analysis. Table 16.11 shows the field stress range considered in the sensitivity analysis.

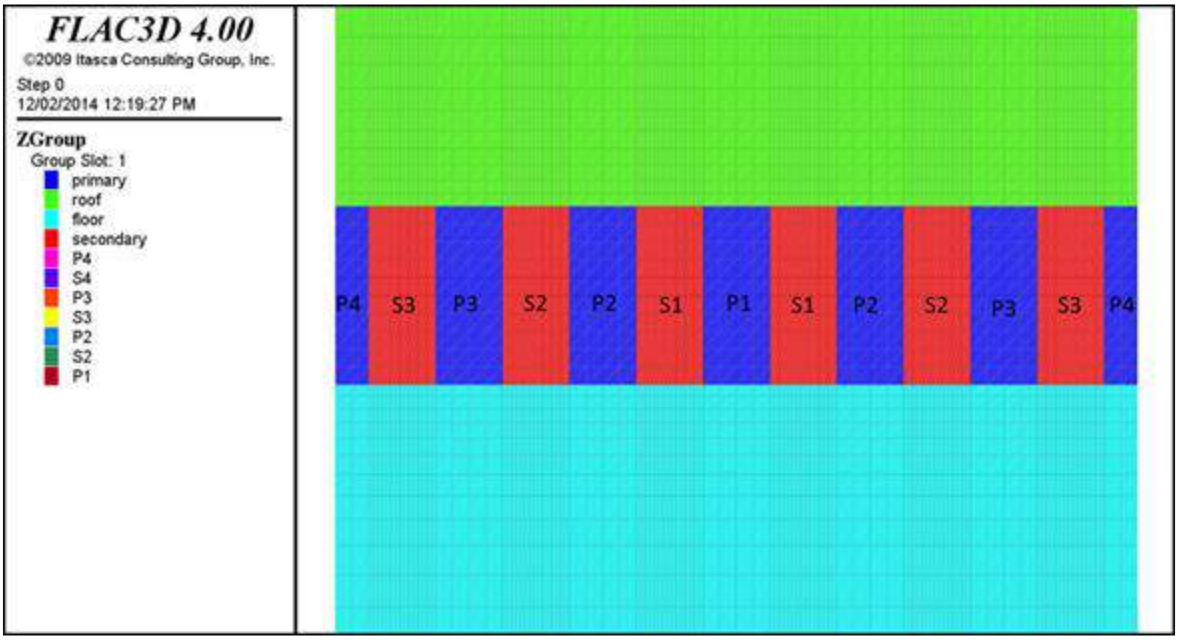
Table 16.11 Field Stress Sensitivity Analysis

Item	K_0	K_{90}
1	1.0	1.0
2	1.0	0.5
3	1.0	1.5
4	1.0	2.0
5	0.5	1.0
6	1.5	1.0
7	2.0	1.0

For simplicity, the mechanical elastic constitutive model in FLAC3D was utilised as a first attempt to quantify induced stress around the excavation, which is required for the stope empirical design. Non-linear analysis should be conducted during the FS.

A conceptual model sequence including six steps was implemented as presented in Figure 16.6 to show the change in stress as more and more stopes are mined. It represents the state of stress in primary and secondary stopes as mining progresses, but does not necessarily reflect a particular mining sequence. In this sequence, the stopes are mined from the middle of the model out, starting with primary stopes (blue). The symbol P1 represents the first primary stope while P2, P3 and P4 represent the second third and fourth primary stopes. S1 to S3 represents the sequence applied on the secondary stopes. Once all primary stopes have been completed (P4) then mining of the secondary stopes (red) commences. The model progresses from a single stope to an infinite number of mined stopes.

Figure 16.6 Mining Sequence



Numerical modelling results

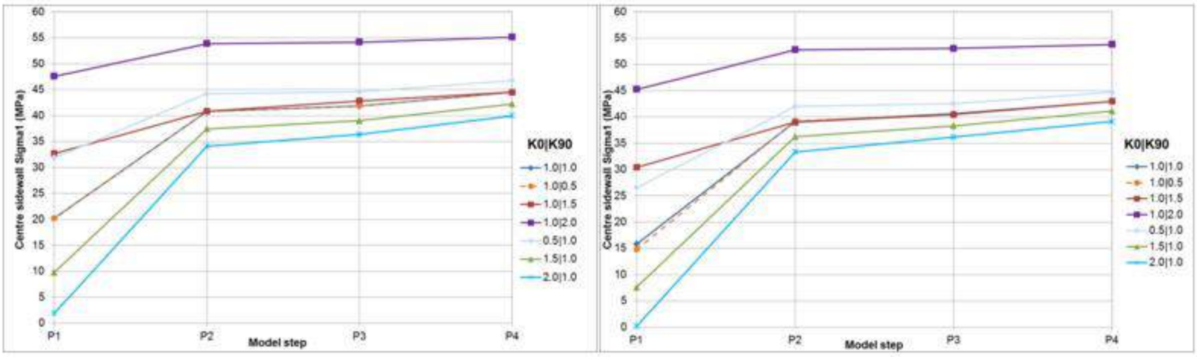
The maximum tangential stress values were queried at the centre of the stope walls and the backs. After each model step an average stress value was computed.

Figure 16.7 and Figure 16.8 show the progression of the maximum tangential stress, for walls and backs, obtained in the models for a variety of possible K-ratio values.

For walls, high stress increase is experienced during model step P2 when the first pillars are formed. The stress increase in subsequent steps is marginal. The highest stress occurs when the stress is oriented parallel to the stope. The 40 m high stopes have slightly lower stresses than the 25 m high stopes.

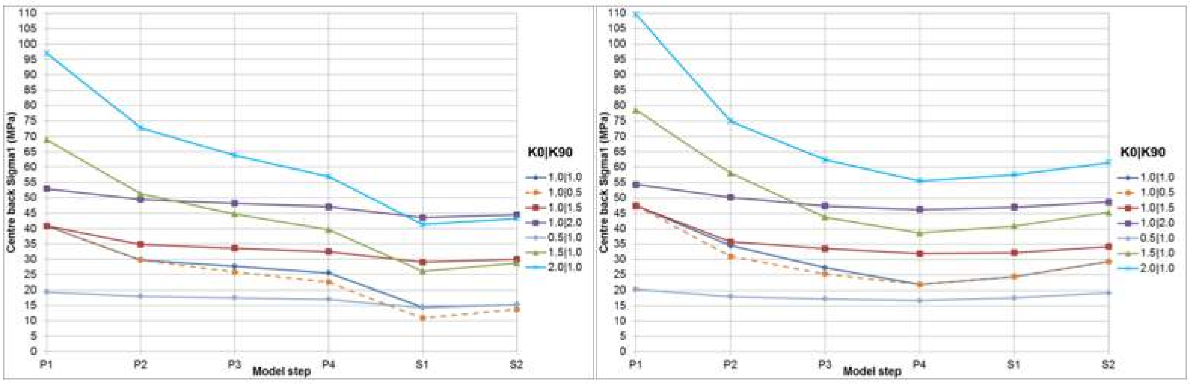
As for the backs, the highest maximum tangential stress values occur at the commencement of mining when the first stopes are mined. As mining progresses, there is a decrease in stress in the backs. A high K ratio will result in higher stress in the backs, particularly when the high horizontal stress is perpendicular to the stope access. The 40 m stopes have significantly higher stresses than the 25 m high stopes.

Figure 16.7 Progression of the Maximum Tangential Stress in the Walls for Various K-Ratio Values



a) 25 m stope height b) 40 m stope height

Figure 16.8 Progression of the Maximum Tangential Stress in the Backs for Various K-Ratio Values



a) 25 m stope height b) 40 m stope height

Empirical Stope Design

The stability of the stopes was assessed using Mathews/Potvin stability graph method and geotechnical borehole data. This method of assessing stope stability was developed by Mathews et al (1981) and modified by Potvin (1988), Potvin and Milne (1992) and further by Nickson (1992). In this method, the classification of the rock mass and of the excavation problem itself is accomplished through the use of the Modified Stability Number (N'), which is plotted against the hydraulic radius of the face of the stope for which the stability is being assessed. The stability number, N', is defined as:

$$N' = Q' \times A \times B \times C$$

Where: *Q' is the modified Q*

A is the Rock Stress Factor

B is the Joint orientation factor

C is the gravity factor

The 20th percentile Q' values used were 9.9 and 8.4 for the orebody and the hanging wall, respectively (see Table 16.8). Assuming that the borehole data is representative of the overall rock mass conditions, the stopes were designed to remain stable for 80% of geotechnical conditions that would probably be encountered. For conditions not accounted for in the design, where severe deterioration of the stope walls may be experienced, it is envisaged that remedial measures will be implemented by further limiting the length of the stope and an early placement of backfill.

The adjustment factor A, also referred to as strength factor, is directly related to the ratio of intact rock strength to induced compressive stress. This ratio decreases as the uniaxial compressive stress acting parallel to the free face of the stope approaches the UCS of the material so as to reflect instability due to rock yield. The mean values of intact rock strength for the OB and HW were taken from (Table 16.6). The stresses of 40 Mpa for the walls and 34 Mpa for the backs were derived from the modelling results for a k-ratio of 1.0. The A factor values were found to be 0.51 and 0.34 for the backs and walls, respectively.

Because of the inherent difference in geometry, the adjustment factors used were different for the walls (right and left) and the backs, and also varied with the stope orientations. A range of stope orientations from 0° to 360° was analysed to assist in optimising the mining direction. The B and C adjustment factors were computed for each orientation and each joint set in each structural domain. The critical joints were identified as the least favourable orientation.

The Joint Orientation adjustment factor, B is a measure of the relative orientation of the dominant joint set relative to the face of the stope for which stability is being assessed. For instance, joints which form a shallow oblique angle (10°-30°) with the face are the most likely to become unstable (B=0.2), while Joints perpendicular to the face are considered to have the least influence on the stability (B=1.0). The true angle between the stope faces and the joint sets were estimated.

The gravity adjustment factor, C, measures the impact of gravity on the stability of the face under consideration. For instance, overhanging stope faces (backs) or structural weaknesses which are oriented unfavourably with respect to gravity sliding have a maximum detrimental influence on stability.

For the gravity adjustment factor, C the most likely gravity driven mode of failure (gravity fall, slabbing and sliding) must be identified. For gravity fall and slabbing mechanisms, the C factor increases from 2 to 8, with increasing dip of the stope face, implying greater stability. For sliding mechanisms, the C factor increases with decreasing dip of joints.

For each situation, the modified stability number, N', was computed based on the relevant Q' and A, B and C equation. The allowable hydraulic radii (HR) were obtained from the Matthews/ Potvin stability graphs in Figure 16.9 for supported and unsupported faces. The resultant HR values for the stope backs were found to be particularly low especially in domain 4 with HR=2.9. Stantec have opted to have larger supported stopes rather than to backfill frequently. It is not practical to support stope walls, so these remain unsupported. Therefore, the design was carried out on the basis of supported stope backs and unsupported walls.

Table 16.12 and Table 16.13 summarise modified stability number N' and allowable hydraulic radii (HR) values, respectively, for different stope orientations and different geotechnical domains and a K ratio of 1.0. The HR values are coloured according to the associated risk levels. For instance, the colour blue represents low risk, while yellow and red represent medium and high risk, respectively. It can be seen that for most geotechnical domains, HR values approximating 8.0 on both walls can be achieved using optimally oriented stopes. However, for geotechnical domain 4, the optimal HR value is expected to be slightly lower (HR=6.8). Similarly, for the supported stope backs, the optimal HR values are mostly greater than 7 for all geotechnical domains.

Figure 16.9 Stability Graph for Unsupported and Supported Excavation

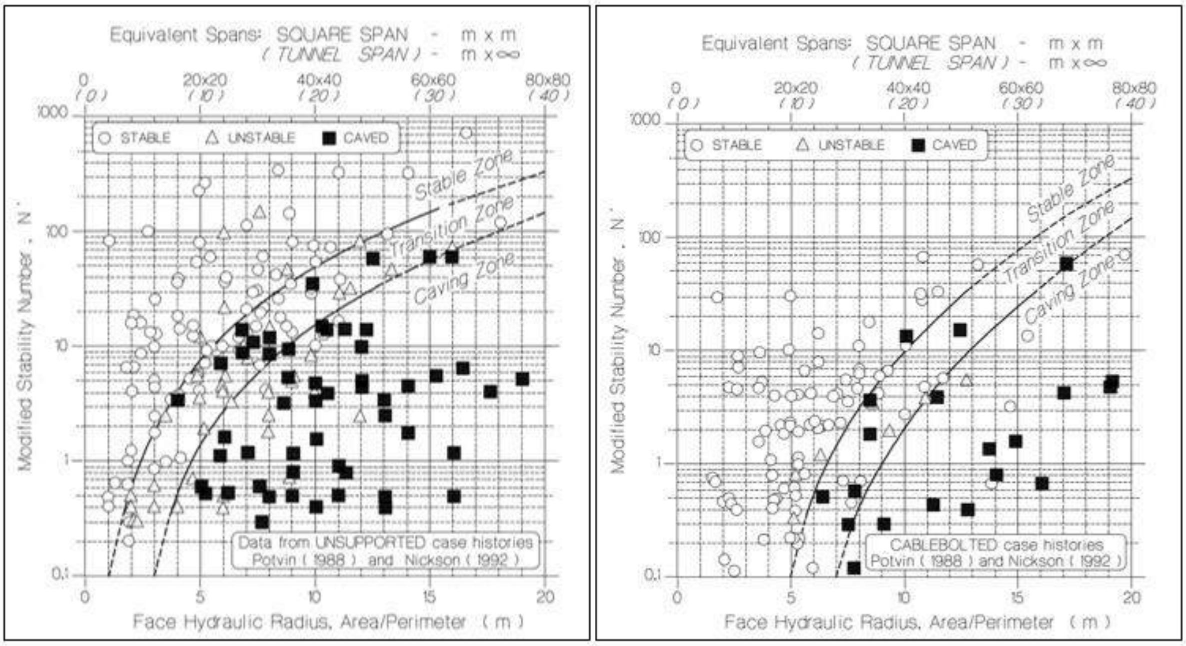


Table 16.12 Modified Stability Number (N') for Different Stope Orientations and Geotechnical Domains

		Stope Orientation (°)												
Domain	**Face**	**0**	**15**	**30**	**45**	**60**	**75**	**90**	**105**	**120**	**135**	**150**	**165**	**180**
1	Back	3	3	3	3	3	3	3	3	3	3	3	3	3
	L wall	14	16	20	16	11	8	8	11	16	22	24	26	27
	R wall	27	27	27	27	27	27	27	25	23	22	18	15	14
2	Back	4	4	4	4	4	4	4	4	4	4	4	4	4
	L wall	27	27	27	27	27	26	24	22	19	14	12	13	15
	R wall	15	19	12	8	7	8	12	19	23	25	27	27	27
3	Back	2	2	2	2	2	2	2	2	2	2	2	2	2
	L wall	3	3	4	8	15	22	25	27	27	27	27	27	27
	R wall	16	22	24	26	27	26	24	22	15	10	8	7	3
4	Back	2	2	2	2	2	2	2	2	2	2	2	2	2
	L wall	11	17	14	7	3	3	3	8	15	22	25	27	27
	R wall	27	27	27	27	27	26	24	22	15	11	8	9	11
5	Back	4	4	4	4	4	4	4	4	4	4	4	4	4
	L wall	27	27	27	27	27	25	23	22	16	13	12	13	17
	R wall	17	14	8	6	6	9	14	22	24	26	27	27	27

Table 16.13 Maximum Allowable Hydraulic Radii for Different Stope Orientations and Geotechnical Domains

		Stope Orientation (°)												
Domain	Face	0	15	30	45	60	75	90	105	120	135	150	165	180
1	Back	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2	8.2
	L wall	6.4	6.7	7.2	6.7	5.8	5.3	5.3	5.8	6.7	7.5	7.8	8.0	8.1
	R wall	8.1	8.1	8.1	8.1	8.1	8.1	8.1	7.9	7.7	7.5	6.9	6.5	6.4
2	Back	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4	8.4
	L wall	8.1	8.1	8.1	8.1	8.1	8.0	7.8	7.6	7.1	6.4	6.1	6.2	6.6
	R wall	6.6	7.1	6.1	5.3	5.0	5.3	6.1	7.1	7.6	7.9	8.1	8.1	8.1
3	Back	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1
	L wall	3.8	3.7	4.0	5.2	6.6	7.6	7.9	8.1	8.1	8.1	8.1	8.1	8.1
	R wall	6.8	7.5	7.7	8.0	8.1	7.9	7.7	7.5	6.5	5.6	5.1	5.1	3.8
4	Back	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1
	L wall	5.9	6.8	6.4	5.0	3.8	3.7	3.8	5.2	6.6	7.6	7.9	8.1	8.1
	R wall	8.1	8.1	8.1	8.1	8.1	8.0	7.7	7.5	6.6	5.8	5.3	5.3	5.9
5	Back	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5
	L wall	8.1	8.1	8.1	8.1	8.1	7.9	7.7	7.5	6.7	6.1	5.9	6.2	6.8
	R wall	6.8	6.3	5.3	4.6	4.7	5.4	6.4	7.5	7.7	8.0	8.1	8.1	8.1

Low risk	
Medium risk	
High risk	

A sensitivity analysis was conducted for the range of rock strengths and possible K ratios. The stope orientations in this analysis are based on the Stantec mine design for each geotechnical domain, which approximate the optimum stope orientations. The maximum allowable hydraulic radius has then been determined for the different scenarios and the results are presented in Table 16.14. It should be noted this method is simply used to present the potential hazard and should not be considered as alternate designs.

Although there is a significant change in the maximum allowable stope back hydraulic radius, it does not drop below the design value of 6.0 and the potential hazard remains low. This is due to the use of stope support. This is a very positive result, since the stability of the back is considered more critical.

The stress increase in the walls for higher K ratios in the walls is less significant than in the backs. However, the wall design is currently aggressive, so the higher stress values represent a significant increase in hazard (from low to moderate risk). This may result in increased overbreak due to spalling of the stope walls. However, this can be managed through visual monitoring of stope conditions and if the deterioration is excessive, the stope can be stopped and filled.

This highlights the importance of stress measurements and the development of strategies to manage this risk. It is also recommended that 3D non-linear analyses are conducted during the FS to further investigate stope behaviour.

Table 16.14 Hydraulic Radii for Primary Stope at Mining Step P2 for 40 m High Stope

Field stress state ($K_0 \mid K_{90}$)		1.0|1.0			1.0|0.5			1.0|1.5			1.0|2.0			0.5|1.0			1.5|1.0			2.0|1.0		
Mean UCS ± 1 Std Dev		Mean-	Mean	Mean +	Mean-	Mean	Mean +	Mean-	Mean	Mean +	Mean-	Mean	Mean +	Mean-	Mean	Mean +	Mean-	Mean	Mean +	Mean-	Mean	Mean +
Domain	Back UCS (MPa)	157	191	224	157	191	224	157	191	224	157	191	224	157	191	224	157	191	224	157	191	224
	Orebody UCS (MPa)	133	166	199	133	166	199	133	166	199	133	166	199	133	166	199	133	166	199	133	166	199
1	Back	7.8	8.1	8.5	8.0	8.4	8.7	7.7	8.1	8.4	6.9	7.3	7.6	9.1	9.3	9.3	6.7	7.1	7.4	6.1	6.5	6.9
	L wall	6.1	6.8	7.4	6.0	6.8	7.4	6.1	6.8	7.4	5.1	5.8	6.4	5.8	6.6	7.2	6.3	7.1	7.7	6.6	7.3	8.0
	R wall	6.9	7.8	8.5	6.9	7.7	8.4	6.9	7.8	8.5	5.8	6.6	7.3	6.6	7.5	8.2	7.2	8.1	8.8	7.5	8.4	9.1
2	Back	8.0	8.4	8.7	8.2	8.6	8.9	7.9	8.3	8.6	7.1	7.5	7.8	9.3	9.6	9.6	6.8	7.3	7.6	6.2	6.7	7.1
	L wall	7.3	8.2	9.0	7.3	8.2	8.9	7.3	8.2	9.0	6.1	7.0	7.7	7.0	7.9	8.7	7.6	8.5	9.3	7.9	8.9	9.6
	R wall	5.5	6.1	6.7	5.4	6.1	6.7	5.5	6.1	6.7	4.6	5.2	5.8	5.3	5.9	6.5	5.7	6.4	6.9	5.9	6.6	7.2
3	Back	6.8	7.1	7.4	7.0	7.3	7.6	6.7	7.0	7.3	6.0	6.4	6.7	7.9	8.1	8.1	5.8	6.2	6.5	5.3	5.7	6.0
	L wall	7.1	7.9	8.7	7.1	7.9	8.7	7.1	8.0	8.7	5.9	6.8	7.5	6.8	7.7	8.4	7.4	8.3	9.0	7.7	8.6	9.3
	R wall	7.0	7.8	8.5	6.9	7.8	8.5	7.0	7.8	8.5	5.8	6.7	7.4	6.7	7.5	8.2	7.3	8.1	8.8	7.6	8.4	9.2
4	Back	6.8	7.1	7.4	7.0	7.3	7.6	6.7	7.0	7.3	6.0	6.4	6.7	7.9	8.1	8.1	5.8	6.2	6.5	5.3	5.7	6.0
	L wall	6.1	6.9	7.5	6.1	6.8	7.5	6.1	6.9	7.5	5.1	5.9	6.5	5.9	6.6	7.2	6.4	7.1	7.8	6.6	7.4	8.1
	R wall	7.3	8.2	9.0	7.3	8.2	8.9	7.3	8.2	9.0	6.1	7.0	7.7	7.0	7.9	8.7	7.6	8.5	9.3	7.9	8.9	9.6
5	Back	8.0	8.4	8.8	8.3	8.7	9.0	8.0	8.4	8.7	7.1	7.6	7.9	9.4	9.7	9.7	6.9	7.3	7.7	6.3	6.8	7.1
	L wall	6.7	7.6	8.2	6.7	7.5	8.2	6.7	7.6	8.3	5.6	6.4	7.1	6.5	7.3	8.0	7.0	7.8	8.5	7.3	8.2	8.9
	R wall	6.7	7.6	8.3	6.7	7.5	8.2	6.8	7.6	8.3	5.6	6.5	7.1	6.5	7.3	8.0	7.0	7.9	8.6	7.3	8.2	8.9

Low risk	
Medium risk	
High risk	

Support Evaluation

The design of the components of the stope back and drift support systems was based on Barton's Q' support chart (2002). The support recommendations obtained from the chart are based on different combinations of rock quality (Q) and Equivalent Span, which is the ratio of the span to the Excavation Support Ratio (ESR).

The ESR is a factor used by Barton to account for different degrees of allowable instability based on excavation service life and usage. The span, ESR and equivalent for each type of excavation are listed in Table 16.15. For the span of intersections, in particular, the length of the diagonal was used considering 5 m wide tunnels.

Table 16.15 Span, ESR and Equivalent Span

Excavation	Span, S (m)	ESR	Equivalent Span=S/ESR (m)
Stopes	15	3	5
Drifts	5	3	≈2
Main haulages and declines	5	1.6	3.1
Medium/short term tunnels	5	2	2.5
Large excavations	Up to 20	1.6	12.5
Intersections	7	2	3.5

To determine the rock tunnelling quality index, Q, the Q' values in Table 16.8 were multiplied by the joint water reduction factor (J_w) and divided by the stress reduction factor, SRF. A groundwater study conducted by Golder indicated that water has little effect, therefore J_w was considered equal to one.

There are two methods of estimating SRF for Barton's Q system. In the first method, SRF is determined based on the major weakness zones if present in the area. Shear zones were observed in the borehole core and the frequency of occurrence is addressed in Table 16.9. This indicates that SRF values of 5 will occasionally be experienced and very occasionally up to 10. An improved structural model will help to indicate these zones in the next phase of study.

In the second method, SRF is a function of the ratio of the uniaxial compressive strength (σ_c) of the rock to the initial in situ major principal stress. The risk of stress induced damage through intact rock, and ultimately rock burst, increases with higher stress. At 1000 m depth, the vertical stress will be about 30 Mpa and the horizontal stress could be anywhere between 15 Mpa and 60 Mpa, although 30 Mpa is considered most likely. The SRF will range between 0.6 and 14, for the range of rock strengths (Table 16.6).

Figure 16.10 shows how Q values and equivalent span pertaining to stope and drifts, respectively, plot on Barton's support design chart. Weld mesh with a 100 mm aperture spacing is recommended for short term excavations in place of shotcrete.

In addition, cable bolts will be required for stope back support. The recommended spacing and length were obtained empirically based on recommendations made by Hutchinson and Diederichs (1996), which is illustrated in Figure 16.11. The empirical method assists in designing cable bolts on the basis of the hydraulic radius (HR) and the stability number (N'). Considering a stope back HR=6 m and N' of about 4, 2.5 m x 2.5 m spacing and 6 m long cable bolts are recommended.

Figure 16.10 Stope back Support Design Based on the Q' Support Design Chart (after Hutchinson and Diederichs, 1996)

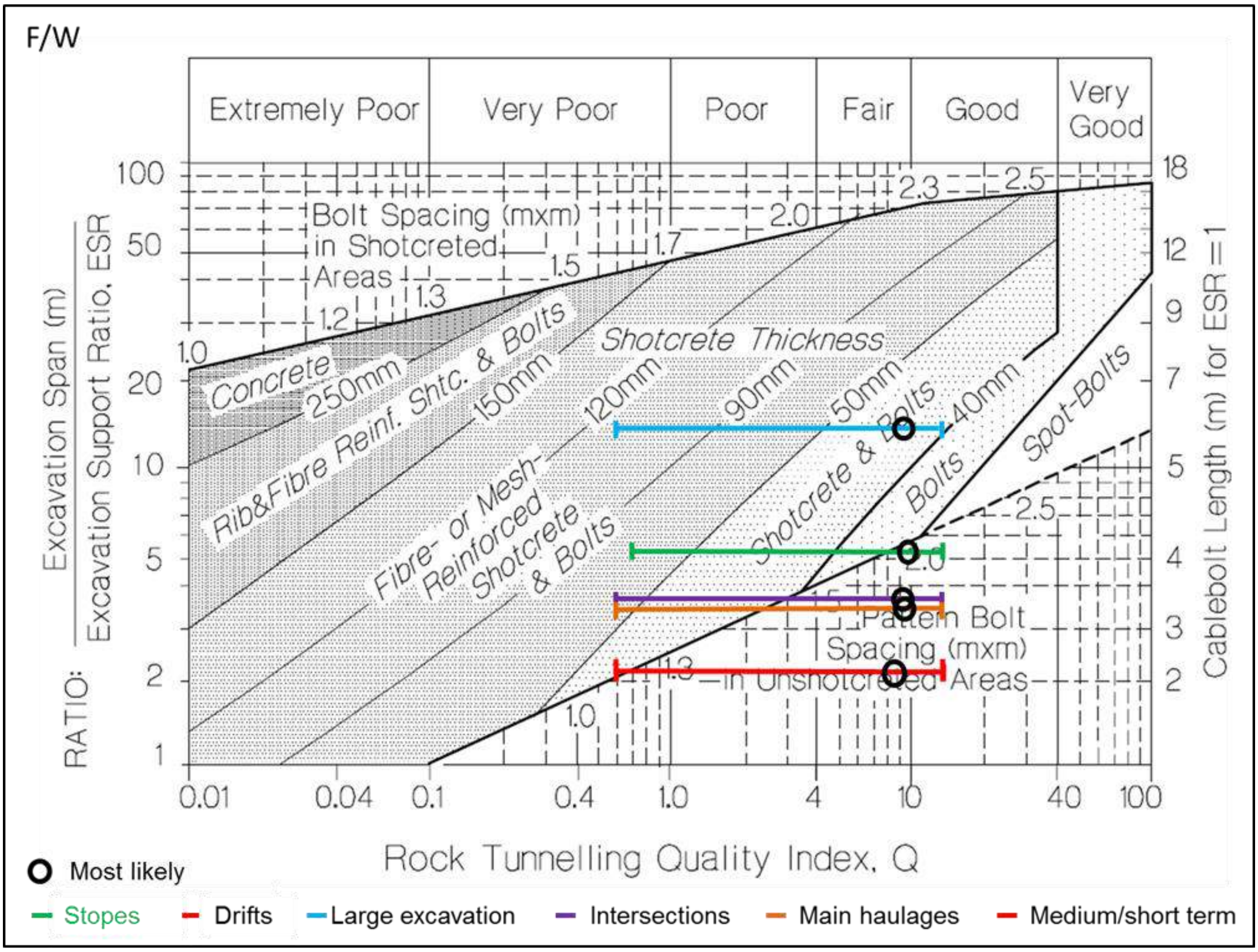
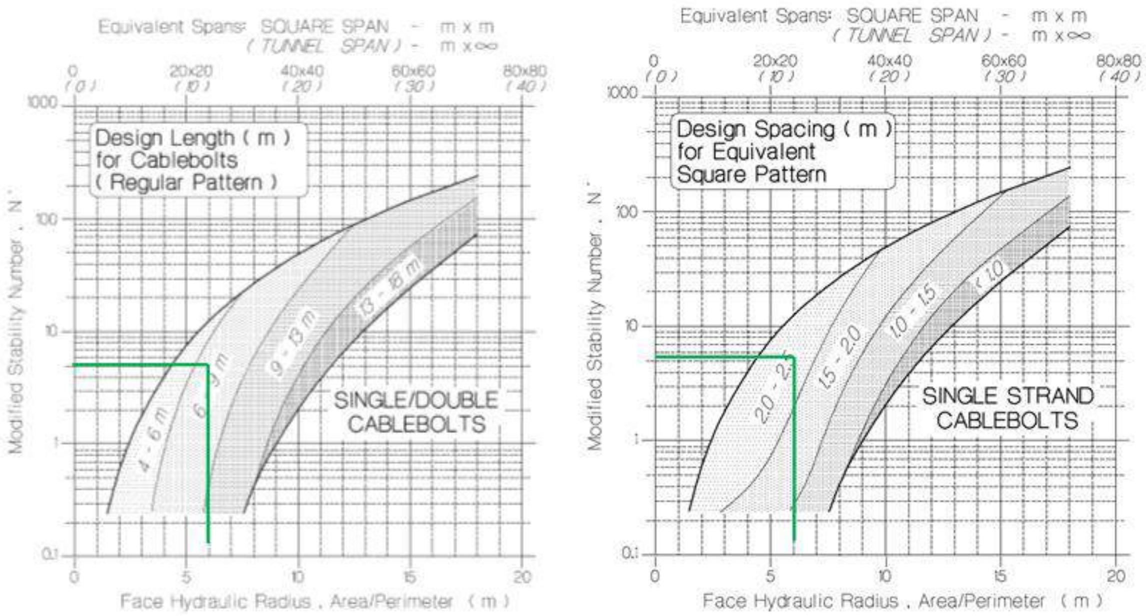


Figure 16.11 Cable Bolt Length and Spacing (after Hutchinson and Diederichs, 1996)



Stope Kinematic Analysis

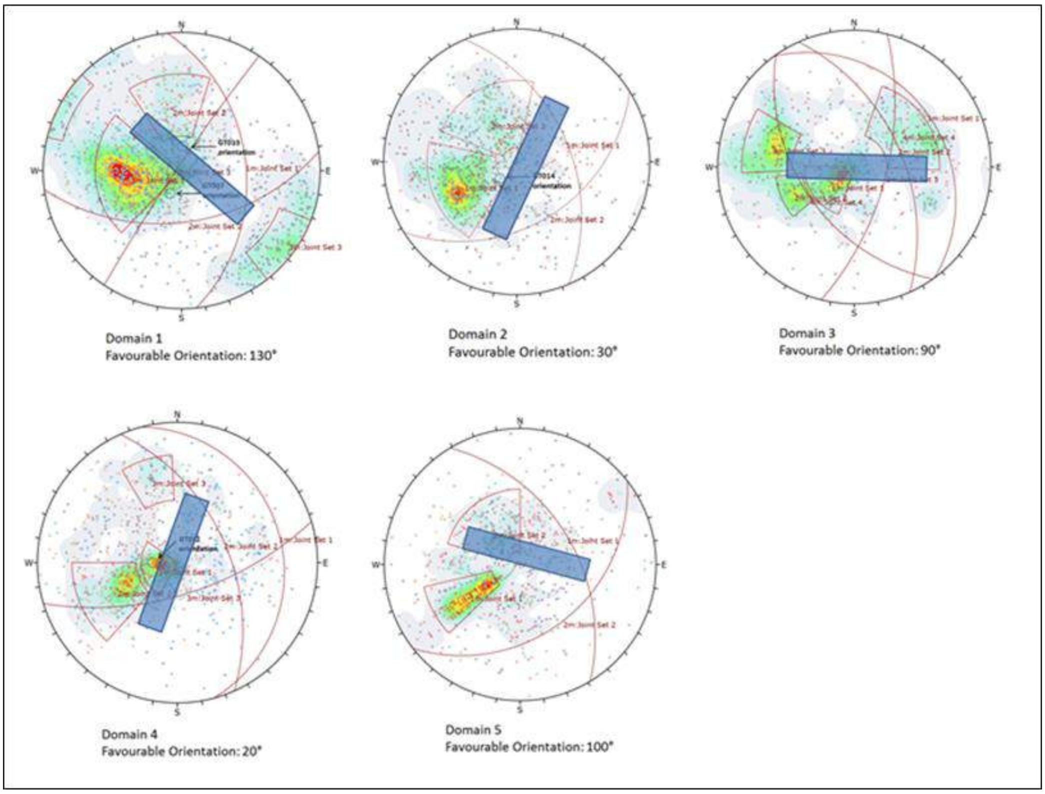
A statistical block analysis of the failure potential of the back area and side walls of the mining stopes was carried out using the program JBlock. JBlock is designed to create and analyse geometric blocks or wedges. The program has the capacity to simulate a large number of keyblocks as a function of joint set characteristics from which to derive a statistical failure distribution. The influence of various support types and support patterns on failure potential can also be simulated.

The first step is to simulate blocks, based on the stope wall or back orientation and the joint set characteristics. This will differ for each geotechnical domain and for different stope wall orientations. More than 100 000 blocks are typically simulated in each case. The second step is to simulate rock falls. A block limit equilibrium analysis is performed on each block, which are placed in random positions within the walls or back of the stope. The stope support in the back is taken into consideration. During the rock fall simulation the exposed surface area is recorded, which allows the number of simulated rock falls to be normalised. The average linear overbreak can be estimated by dividing the total volume of rockfalls by the exposed surface area.

The failure potential of blocks has been simulated for the back, left wall and right wall of the mining stopes, orientated as per mine design, in each structural domain (Figure 16.12). Joint input parameters include mean joint orientations (Table 16.10) and their standard deviation, joint spacing and joint friction angles. The following support scenarios were simulated:

- Left stope wall without support
- Right stope wall without support
- Back with mesh and tendon support

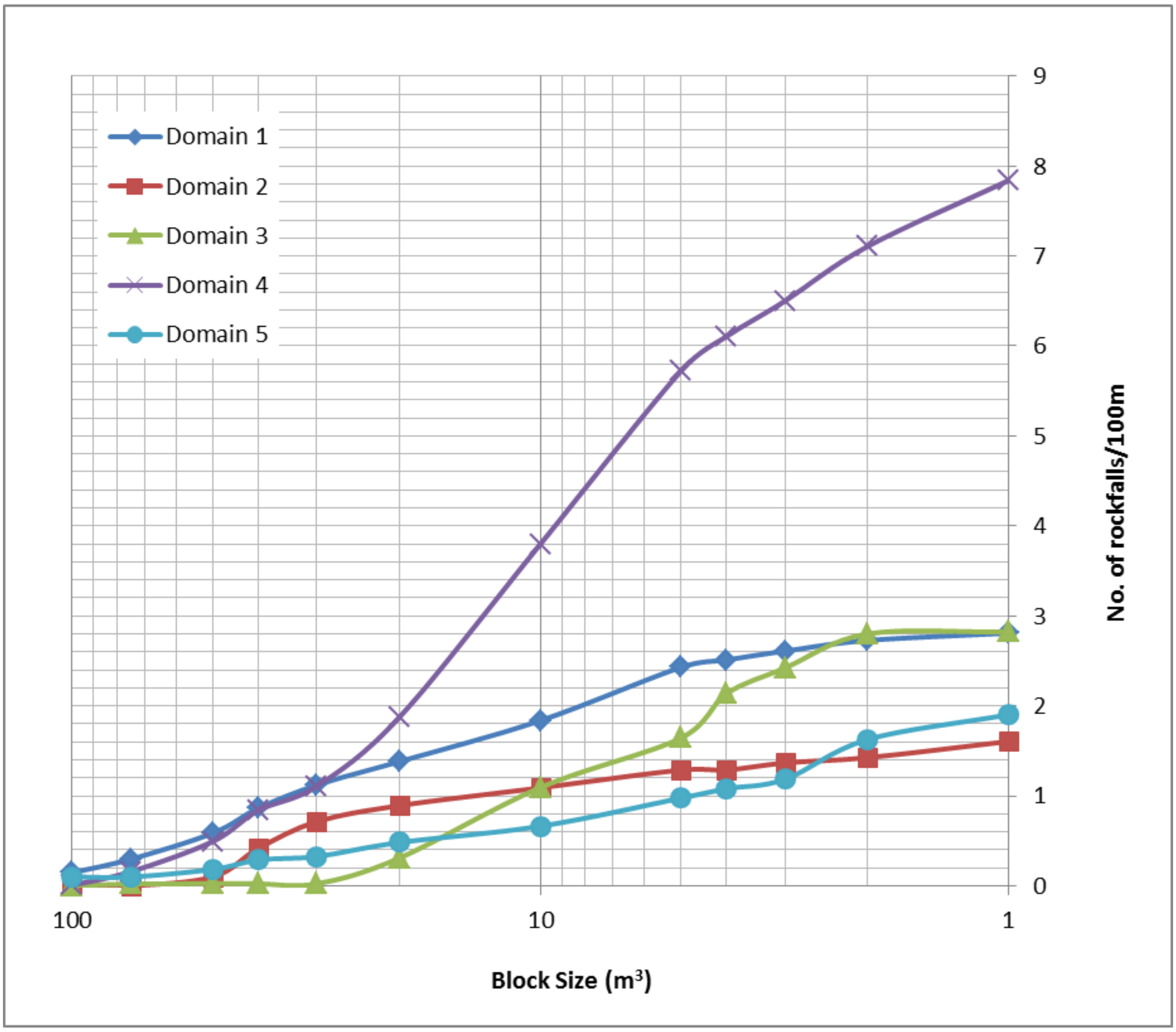
Figure 16.12 Design Stope Orientations per Structural Domain



The results indicate that the geologically controlled overbreak will be less than 1%, is low and unlikely to influence the overall project.

The number of rockfalls normalised to rock falls per 100 m advance using the total area exposed and the relevant dimension (stope height for walls and stope width for backs) is presented for each structural domain in Figure 16.13. This provides an indication of the frequency of occurrence of large rockfalls. The risk of damaging incidents due to large rock falls is more significant, particularly in Domain 4 and 1, but this is considered manageable.

Figure 16.13 No. of Rock Falls per 100 m Advance for Each Structural Domain



16.1.3 Stope Layout, Sequence and Support Requirements

Recommendations for the stope layout, sequence and support requirements were provided in the preliminary geotechnical design parameters report (SRK Project Memo 458213 M2).

Stope Dimensions and Orientation

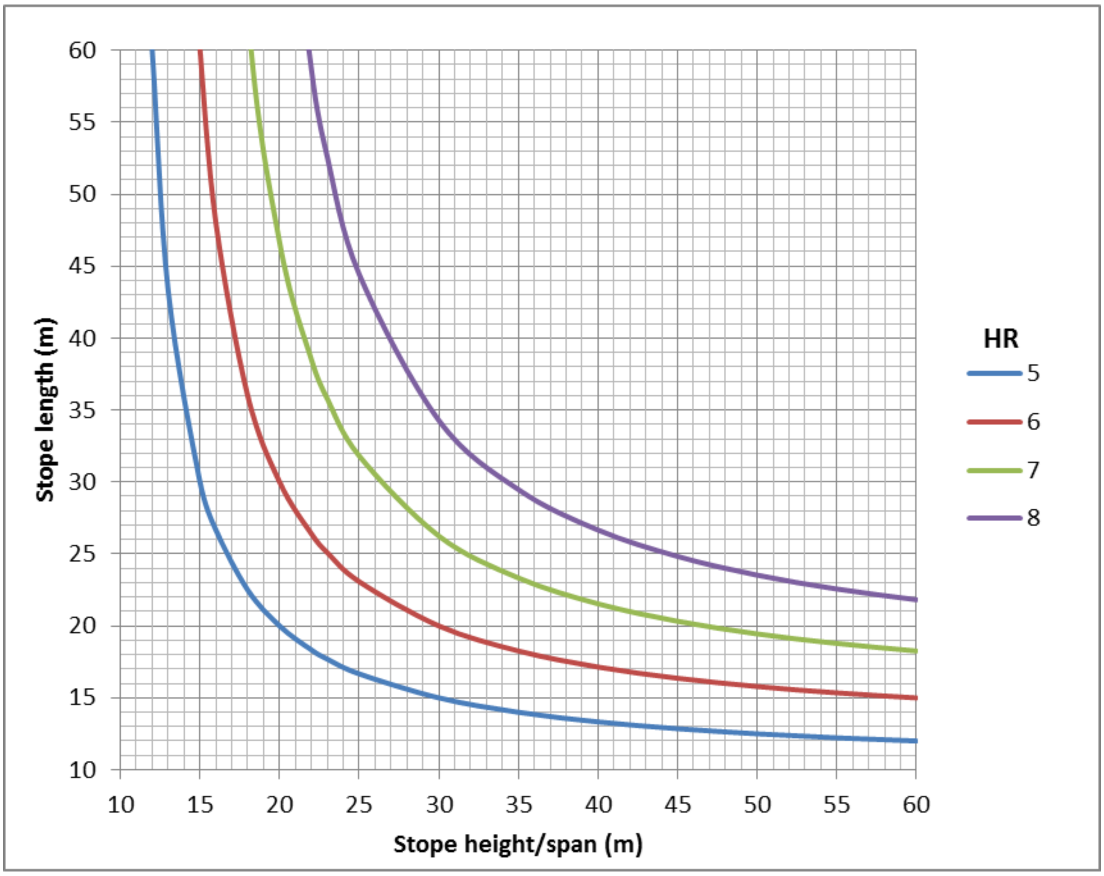
The stope dimensions can be determined from the chart presented in Figure 16.14.

Back: Max HR = 6 m

Walls: Max HR = 8 m

It should be noted that the hydraulic radius for the walls in the optimum orientation (Table 16.13) is sometimes lower than 8. This indicates that some instability of the walls can be expected. However, the stopes are non-entry and the hydraulic radius of the stope walls will be gradually increased as the stope retreats. It will be possible to monitor the condition of the walls as the stope retreats and larger hydraulic radii may actually be achieved. If the conditions of the walls deteriorate, then the stope can be stopped and backfill placed. If there is some sloughing off the walls, secondary blasting may be required to remove this ore. In extreme cases, there will be minor ore losses. The failures will effectively reduce the volume of ore in the secondary stopes, which will be replaced by backfill. It will be possible to implement a management system to minimise ore losses and disruptions to production.

Figure 16.14 Stope Dimensions



The optimum stope orientations are presented in Table 16.16, which are determined from Table 16.13.

Table 16.16 Optimum Stope Orientations

Domain	Optimum orientation
1	130°
2	15°
3	90°
4	15°
5	100°

Mining Sequence

Two mining methods are being considered. Where the orebody is greater than 18 m, Longhole stoping with post backfilling will be used, while drift-and-fill with benching will be applied in the narrower areas.

Longhole Stoping with post backfilling

The recommendations for longhole stoping are as follows:

- Longhole stoping will be applied when the orebody is between 18 m and 40 m thick.
- The stope cross-sectional dimensions considered at this stage are 15 m wide x height of the target stoping zone.
- The unfilled stope length should be limited, so that the design hydraulic radius of 8 for the walls is not exceeded.
- As far as possible, the stopes should be aligned to minimise the risk of failure. Where this is not possible and the hazard is medium to high, the rock falls may result in increased costs and slightly lower productivity.
- The stopes must be non-entry. The broken ore should be removed with remote LHDs.
- At 1000 m depth, minor rock bursts are anticipated, but this risk can be managed. It is not expected that it will be necessary to implement a complex centre-out stoping sequence to manage rock bursts.
- A primary- secondary stoping sequence will be more productive and will not require a long build up time.
- Adequate time should be allowed for the construction of stope bulkheads, filling of stopes and curing of backfill to the required strength
- Adjacent secondary stopes should not be mined simultaneously and backfill should be placed prior to mining the next secondary stope.
- Secondary stope development can be carried out at any stage.

Drift, bench and fill

The recommendations for drift, bench and fill are as follows:

- The width of the drifts should be limited to 5.5 m to limit the length of support to 2.4 m in the crown and walls.
- Wider drifts can be considered, but longer and stronger bolts will be required to support the crown. The crown should be supported with mesh or shotcrete to ensure the safety of personnel. Where the drift or bench height exceeds 7 m, personnel access should be restricted.
- The walls of benches do not need to be supported, provided that these are restricted access and remote LHDs are used to remove the broken ore.
- The sequence should incorporate at least primary, secondary and tertiary drifts. This is to ensure that slender backfill ribs are not formed. Where the overall unfilled excavation height exceeds 12 m, a 4 drift/bench sequence may be required.

16.1.4 Stope Support

Table 16.17 presents the type of support proposed for the drift and stope back and Table 16.18 summarises the support specifications.

Table 16.17 Support for Drifts and Stope

Support type	Drift	Stope Back
Split Sets	X	X
Cable Bolts		X
Weld Mesh	X	X

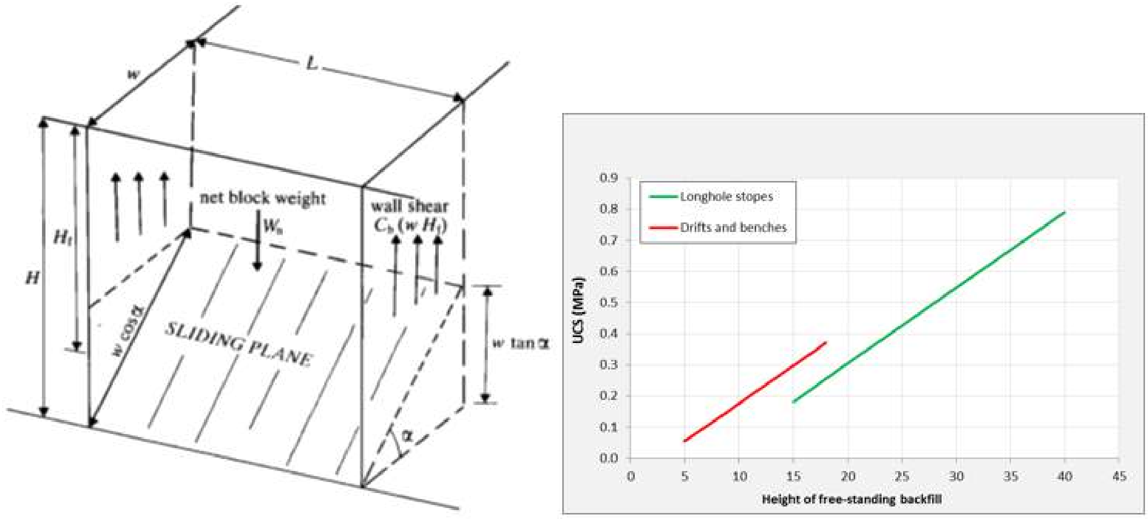
Table 16.18 Drift and Stope Support Specifications

Split Sets	1.5 m x 1.5 m spacing, 2.4 m length 4.5 - 8.2 ton, 46mm outer diameter of tube (SS-46 model)
Cable Bolts	6 m Long, 2.5 m spacing 25 t
Weld Mesh	100 x 100 mm aperture spacing, 5.6 mm thick strands

Backfill Requirements

Preliminary fill strength requirements have been determined using Mitchell's analytical backfill model for static analysis of fill stability (Figure 16.15). When a backfill wall is exposed, it will be necessary to use sufficient binder to ensure that the backfill is strong enough to be free-standing (primary). The secondary stopes and final drift/benches in a given sequence will not be exposed on one side and do not need to be designed to be free-standing. However, the backfill will require a minimum binder content of about 2% to prevent subsequent liquefaction.

Figure 16.15 Backfill Strength Requirements Determined Using Mitchell, 1983



Bulkheads

Bulkheads will need to be designed and constructed at the stope entrances. There are many types of bulkheads in use, but reinforced shotcrete arch bulkheads are recommended, since they can be constructed relatively quickly.

Preliminary Bulkhead design specifications are presented in Figure 16.16 and Table 16.19, which are based on bulkhead specifications used by similar underground mining operation.

A simple yield line analysis was conducted to verify the required thickness for 5 m wide stope drifts. Factors of safety of 1.3 and 1.9 were considered for the base of the bulkhead and for the centre. The shotcrete uniaxial compressive strength and the thickness were downgraded to account for the fact that in practice the curing time is more likely to be less than 7 days. It is also important to point out that the method is somehow conservative since it does not account for the additional resistance forces due to the arched shape of the bulkhead.

Detailed design of the bulkheads will be required prior to implementation, which should include numerical modelling to quantify the membrane (arching) forces that are mobilised at the bulkhead edges. Monitoring of pressure build up in the stope and at the bulkhead will be critical when the first stopes are filled. Routine pressure monitoring of pressure build up behind the bulkhead is essential.

Figure 16.16 Construction of Fibrecrete Bulkhead (after Andrews et. al. 2010)

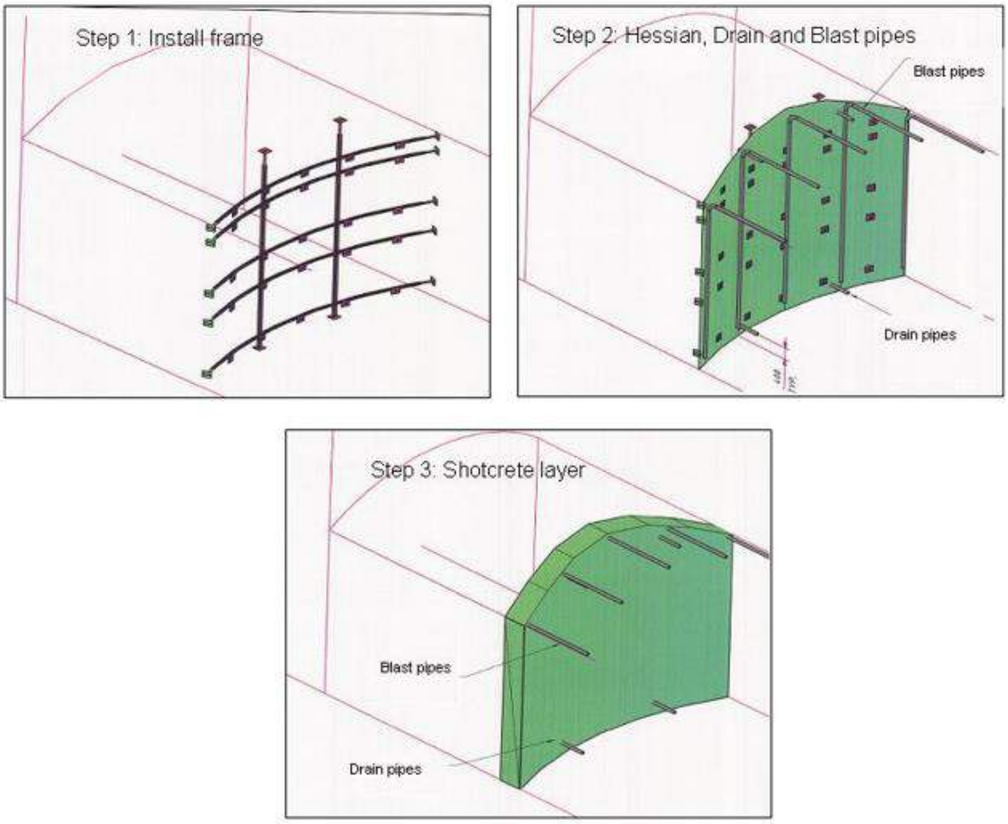


Table 16.19 Fibre Reinforced Shotcrete Bulkheads

Specifications	
Maximum stope entrance dimensions	5.5 m x 5.5 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	2 days
Plug height	8 m
Plug binder content	5 %
Plug curing time	48 hours
Materials	
Arched wall frame kit	
Mesh 100 x 100 x 5 mm	
Hessian	
Fibre reinforced shotcrete	
Poly propylene drainage pipes	
Geotextile fabric for drain pipes	
Earth pressure cell	
Piezometer	

16.1.5 Development Layout and Support Requirements

The following rules should be applied for development layouts:

- Large, important excavations should be developed in good quality rock, in areas that are not influenced by abutment stresses.
- Access development should avoid high stress abutments as far as possible and additional support will need to be installed if tunnels are likely to experience high stress during their life (Detailed numerical modelling of the mining sequence will be required in the FS to determine the stress levels).
- Excavations should be separated by 3x the combined width to avoid stress interaction. Where this is not possible, it will be necessary to model the stress interaction and determine the additional support requirements.
- As far as possible, major geological structures should be avoided. Where geological structures occur, the excavation should be developed at a large angle (>45°) to the strike of the structure.

- Large altered zones may occur in the footwall as identified in the Shaft 1 borehole. These are expected to be infrequent and could cause major delays.

This highlights the importance of improving the structural geological model using the seismic survey during the FS.

Development Rock Support

The development support requirements and specifications listed in Table 16.20 and Table 16.21.

Table 16.20 Development Support

	Main haulages and Declines	Medium/Short term tunnels	Large Excavations	Intersections (additional support required)
Split Sets		X		
Resin Rebar	X		X	
Cable Bolts			X	X
Weld Mesh		X		
Shotcrete	X		X	

5 % of the tunnels may require additional support in the form of cable anchors and straps for poor ground conditions.

Table 16.21 Support Specifications

Split Sets	1.5 x 1.5 m spacing, 2.4 m length 4.5 - 8.2 ton, 46mm outer diameter of tube (SS-46 model)
Resin Rebar	1.5 x1.5 m spacing, 2.4 m length 25–28 mm hole diameter, 20 mm bolt diameter (14 tonne)
Cable Bolts	length (half excavation span), spacing (half the length of cable bolt) 38 ton (5 cable bolts required for intersections)
Weld Mesh	100 x 100 mm aperture spacing, 5.6 mm thick strands
Shotcrete	25 Mpa fibre reinforced shotcrete (28 day strength), 50 mm thick

16.1.6 Protection of Shafts

Shaft Pillar

A trade off study was conducted to investigate the options of leaving a shaft pillar and early extraction of shaft reef and was presented in SRK Project Memo 458213 M3. This study concluded that it was lower risk to leave a shaft pillar and that the upside to early shaft reef extraction was not significant. The shaft pillar must be large enough that the stress levels in the centre of the pillar do not cause damage to the shaft and critical excavations (field stress < 0.5 UCS). The shaft pillar radius should be 250 m for the dual shaft system and 200 m for the ventilation shafts.

However, it should be noted that if the geological losses prove to be lower than anticipated and a higher overall extraction is achieved, the shaft pillar will ultimately become highly stressed and problems will be experienced. If this is the case it will need to be managed through updated numerical modelling and monitoring.

The shaft pillar can be extracted at the end of the life of the mine. It is expected that at least 50% of the shaft pillar can be extracted safely at the end of the life of mine.

Rock Support

Detailed geotechnical investigations were carried out for shafts 1 and 2 (SRK Reports 450790/2 and 458213/2). It was found that the rock mass was of suitable quality for sinking to a depth of approximately 1100 m in Shaft 1 and 2. It is expected that these rock mass conditions are representative of the typical rock mass conditions for the other shafts. The primary support during sinking should comprise split sets and mesh to protect personnel at the base of the shaft. The secondary, long term support will be provided by the 300 mm shaft lining.

Below 1106 m in Shaft 1, an extremely poor quality fractured hornfels, extends to a depth of at least 1226 m. It is believed that the poor rock mass conditions are due to a very large xenolith. It is not recommended that the shaft should be sunk through this material. If it is necessary to extend the shaft through this material, an intensive reinforcement and ground consolidation support system will be required to ensure long term stability of the shaft.

16.1.7 Conclusions

The geotechnical investigation and design for the Platreef PFS has been completed. The geotechnical investigation was based on 50 boreholes for rock mass characterisation and 21 boreholes for structural analysis. Laboratory rock strength testing was also conducted to determine rock properties. Stoping and access development design parameters have been derived and a risk assessment was carried out. A summary of the findings is presented below:

- There is very little variability in the quality of the rock mass:
  - The 20 percentile Q' values used for stope wall and back design were 9.9 and 8.4, which represents fair ground (Barton) or good ground (Bieniawski);
  - The 20 percentile Q' values used for the design of development support ranged from 7.4 to 9.2, which represents fair ground (Barton) or good ground (Bieniawski);
  - There are small weak zones within the hanging wall, orebody and footwall. These are sheared and altered. However, they are too small to significantly affect the rock mass classification. They will have a structural effect and this is taken into consideration in the structural analysis.
- Five structural domains were created and remained consistent with the additional of the PFS boreholes are GT007. The stopes in the Stantec mine design have been optimally orientated within each domain.
- Rock properties and variability have been determined for the hanging wall, orebody and footwall from a comprehensive set of laboratory test data:
  - The mean hanging wall, orebody and footwall strengths are 191 Mpa, 166 Mpa and 185 Mpa;
  - Rock mass properties and their variability have been determined for non-linear modelling to be conducted during the FS.
- Numerical modelling was conducted to investigate the effects of stress in the back and walls as mining progresses. The updated model is more representative and considers other possible stress regimes:
  - The horizontal to vertical stress (K) ratio is expected to be 1.0, based on stress measurements at 1 000 m depth in the bushveld, however, higher K ratios of up to 2.0 have been measured at shallow depth;
  - High horizontal stress perpendicular to the stope axis will result in significantly higher stresses in the stope back;
  - High horizontal stress parallel to the stope walls will result in moderate increase in the stope wall.
- Detailed stope recommendations were provided.
  - The design is based on the expected stress regime (K = 1) and the mean rock strengths, taking the effects of stope height and mining progression into account;
  - The stope hydraulic radii for design of the walls and backs (supported) are 8 m and 6 m;
  - The stopes must be supported with split sets, mesh and 6 m long cable anchors;
  - Stope orientation should be optimised within each structural domain;

- Structural overbreak is not expected to be significant (less than 1%) for the design stope orientations;
- Large rock falls in stopes are expected and will be more frequent in domain 4 and 1, due to the combinations of joints;
- Higher stresses in the stope back is not expected to affect stability, because the stope back is supported;
- Higher stresses in the stope walls will increase the risk of instability, because the walls are not supported and the design is marginal;
- These risks can be managed through visual stope monitoring during mining and early placement of backfill if conditions deteriorate excessively. This could influence stope availability during mining. A more detailed analysis will be required in the FS to better understand the stope wall behaviour.
- Preliminary binder strength requirements were provided, which range from 0.05 Mpa to 0.8 Mpa.
- Preliminary bulkhead specifications were provided, which includes material and monitoring requirements.

* Detailed recommendations for access development layout and support were provided:
  - High stress abutments and major geological structures should be avoided as far as possible. Additional support will be required where this is not possible;
  - Generic development support recommendations were provided for short/medium term excavations, main haulages and intersections;
  - Approximately 5% of tunnels will require additional support due to geological weaknesses. Further work will be required during the FS to indicate where these weak zones occur.
  - The proposed support systems will cater for adverse stress conditions
* Recommendations for shaft protection were provided:
  - A trade off study was conducted to investigate the options of leaving a shaft pillar or early shaft reef extraction. It was concluded that the upside benefits to early shaft reef extraction are not significant and do not outweigh the risks;
  - The main shaft complex pillar must have a radius of 250 m and the additional shaft pillars should have a radius of 200 m;
  - It is expected that at least 50% of the shaft pillar could be extracted at the end of the mine life;
  - Split sets and mesh are required for primary support, while the concrete lining provides secondary support;
  - In Shaft 1, between 1106 m and 1226 m depth an extremely poor quality fractured hornfels occurs.

16.1.8 Recommendations for the Feasibility Study

The following work is recommended for the FS to further improve confidence in the geotechnical design:

- The structural geology should be updated, incorporating the latest structural data and the results of the seismic survey.
- Additional geotechnical boreholes should be drilled. They must either be oriented or an ATV log should be conducted or both.
- Rock mass characterisation of geotechnical boreholes should be conducted.
- Additional representative samples should be selected for laboratory testing to fill in gaps in the database. The laboratory testing should include UCS, TCS and BTS.
- Acoustic Emission (AE) or Deformation Rate Analysis (DRA) Stress measurements should be carried out to determine the virgin stress field. Overcoring stress measurements should be carried out in underground access development, prior to stoping.
- The geotechnical structural domain analysis should be updated based on the new structural geology information and additional boreholes.
- 3D conceptual non-linear modelling of stopes should be conducted to better understand the stability of the stope backs and walls.
- 3D elastic modelling of the overall mine layout and sequence should be carried out to:
  - Ensure that critical excavations are not adversely affected by stress;
  - Optimise or verify the shaft pillar dimensions;
  - Investigate the potential for seismicity on major geological structures.
- The stope design and sequence should be optimised based on the numerical analyses.
- The layout of access development and critical excavations should be optimised based on the 3D elastic stress analysis and the new structural geology model.
- The geotechnical risk assessment should be updated with the new information and analyses.

16.2 Mining

16.2.1 Introduction

This section has not been changed from the Platreef 2014 PFS and remains the most currrent study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

Mining zones included in the current Platreef mine plans occur at depths ranging from approximately 700 m to 1,600 m. Access to the mine will be via multiple vertical shafts. Mining will be performed using highly productive mechanised methods and paste backfill will be utilised to fill open stopes. When available, excess waste rock will be used as backfill where cemented backfill is not required.

In the Platreef 2014 PFS Stantec had primary responsibility for mine planning and design for the underground mine and certain related infrastructure and for associated capital and operating cost estimates.

All pertinent technical and economic data related to the mining of the resource was provided by Ivanhoe, including updated labour and materials costs. All dollar amounts throughout the report are expressed in 2014 US Dollars (US$).

16.2.2 Mining Block Model

The Mineral Reserve estimate for Platreef was based on the Mineral Resource reported in the Platreef 2014 PFS. Only Indicated Resources have been used for determination of the Probable Mineral Reserve.

The resource block model for selective mining within and adjacent to the TCU was used to prepare a mining block model (file name: pmbdt13a.bmf) for mine planning. The mining block model was prepared by updating each block in the resource block model to include Preliminary estimates of price and metallurgical parameters to calculate the Net Smelter Return (NSR). 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. This same Mineral Resource and mining block model was used for the Platreef 2014 PEA.

The NSR was calculated based on the parameters provided by Ivanhoe in Base Data Template 13 (BDT13) dated 22 March 2013. The BDT13 metal prices and realisation assumptions are shown in Table 16.22 and Table 16.23. The BDT13 metal recovery to concentrate was based on a fixed tail grade (Table 16.24) and a constant mass pull of 3.23%.

Table 16.22 BDT13 Metal Prices

Metal	Units	Price
Pt	$/oz	1,699
Pd	$/oz	667
Au	$/oz	1,315
Rh	$/oz	1,250
Ni	$/lb	8.81
Cu	$/lb	2.73

Table 16.23 BDT13 Realisation Assumptions

Payable Metal	%	82%
Transport	$/t Conc	22.00
Royalty	%	5.00

Table 16.24 BDT13 Fixed Tail Grade

Domain	Pt g/t	Pd g/t	Au g/t	Rh g/t	4PGE g/t	Ni %	Cu %
T1	0.30	0.25	0.10	0.007	0.63	0.09	0.03
T2U	0.32	0.28	0.11	0.007	0.71	0.11	0.03
T2L	0.32	0.28	0.11	0.007	0.75	0.11	0.03

The distribution of tonnes and grade in the mineral resource block model by resource classification is shown in Table 16.25.

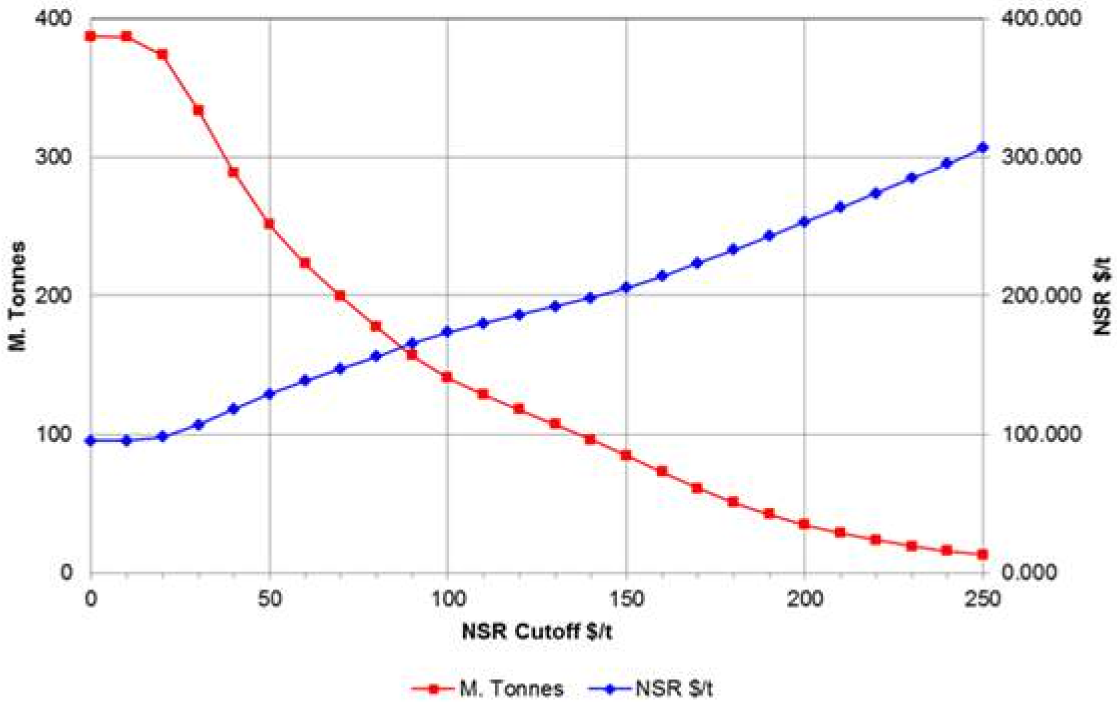
Table 16.25 Distribution of Tonnes in Block Model

Metal	Indicated	Inferred
Mt	387	1,054
Cu (%)	0.14	0.13
Ni (%)	0.28	0.26
Pt (g/t	1.28	0.96
Pd (g/t)	1.34	1.02
Au (g/t)	0.21	0.18
Rh (g/t)	0.09	0.07
3PE+Au (g/t)	2.92	2.23

For the Platreef 2014 PFS, only the indicated resource is used. Inferred metal values are zeroed out in the block model for all reserve calculations.

The graph in Figure 16.17 shows the tonnes and average NSR values for a full range of NSR cut-off grades for all indicated and inferred resources included in the mineral resource block model.

Figure 16.17 Platreef Available Resources



16.2.3 Mining Method Selection

The key criteria considered in the selection of the mining method for Platreef include the following.

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

The following mining methods are proposed for use at Platreef.

- Longhole Stoping
- Drift-and-Bench Stoping
- Drift-and-Fill (mechanised cut-and-fill)

Longhole stoping will be used where the thickness of the ore zones exceeds 18 m. Drift-and-bench stoping will be used in zones of intermediate thickness (between 10 m and 18 m thick). Drift-and-fill will be used in thinner portions (less than 10 m thick). All three methods use paste backfill or excess waste rock where cemented backfill is not required to fill open stopes. Prior to mill start-up and the availability of paste fill, a cemented rock fill (CRF) facility will provide cemented backfill during the initial years of mine production and will be available for future use should the need arise.

16.2.4 Economic Cut-off Grades

Prior to beginning stope design work, NSR cut-off values were evaluated by Stantec. The evaluation used updated mining costs estimated by Stantec during the 2013 scoping study update as well as updated processing and G&A costs provided by Ivanhoe.

Table 16.26 summarises mining, processing, and G&A costs for longhole stoping.

Table 16.26 NSR Cut-off Value Calculation – Longhole Stoping

Item	Unit Cost ($/t)
Mining	$11.83
Backfill	$6.77
Haulage	$0.16
Indirects	$5.63
Mine Air Cooling	$1.00
Power	$2.31
Water	$0.62
Undefined Allowance	$2.44
Subtotal Mining Cost	**$30.76**
Processing	$13.32
G&A	$3.63
Total Cost	**$47.71**

Table 16.27 summarizes mining, processing, and G&A costs for drift-and-bench stoping.

Table 16.27 NSR Cut-off Value Calculation – Drift-and-Bench Stoping

Item	Unit Cost ($/t)
Mining	$14.09
Backfill	$8.04
Haulage	$0.19
Indirects	$6.70
Mine Air Cooling	$1.00
Power	$2.31
Water	$0.62
Undefined Allowance	$2.90
Subtotal Mining Cost	**$35.85**
Processing	$13.32
G&A	$3.63
Total Cost	**$52.80**

Table 16.28 summarizes mining, processing, and G&A costs for drift-and-fill stoping.

Table 16.28 NSR Cut-off Value Calculation – Drift-and-Fill Stoping

Item	Unit Cost ($/t)
Mining	$18.10
Backfill	$8.04
Haulage	$0.19
Indirects	$7.90
Mine Air Cooling	$1.00
Power	$2.31
Water	$0.62
Undefined Allowance	$3.42
Subtotal Mining Cost	**$41.58**
Processing	$13.32
G&A	$3.63
Total Cost	**$58.53**

Based on the costs summarised in the tables above, the minimum NSR cut-off grades are estimated as provided in Table 16.29.

Table 16.29 Minimum NSR Cut-off Grades

Mining Method	NSR Cut-off ($/t)
Longhole Stoping	$47.71
Drift-and-Bench Stoping	$52.80
Drift-and-Fill Stoping	$58.53

These NSR cut-offs do not include allowances for capital recovery or profit margin.

Stantec used a declining cut-off grade approach in preparing the production schedule for the PFS. Early production is concentrated on higher grade resources located at shallower depths and in relatively close proximity to the production shaft. A $100/t NSR cut-off value was used in defining these resources in order to increase the initial mill head grade and to shorten the payback period. An $80/t NSR cut-off value was used later in the mine life as the higher grade resources deplete and mining progresses further from the production shaft. Lowering the cut-off grade ensured that adequate resources are available to satisfy Ivanhoe's requirement for the 30-year mine life after mill start-up.

16.2.5 Dilution and Recovery Factors

Due to irregularities in the geometry of the mineralised zones, not all cut-off grade material can be mined without incurring some dilution. In addition, small wedges of cut-off grade material can be left un-mined outside the limits of the planned stopes. Due to inefficiencies in final mining recovery from the stopes, small amounts of mineralised material are lost during final stope clean out and additional losses may occur in transit from the stopes to the mill. A mining recovery factor is applied to the diluted resources to account for these losses.

Several stope triangulations created by the Stope Optimiser were selected from various parts of the deposit for the purpose of compiling a representative sample group for dilution assessment. An assumption was made that overbreak will occur along the stope floor, back, and ends. Overbreak on the side walls (ribs) of the stope was not considered in the primary stopes since this overbreak mainly comes at the expense of the adjacent stopes. For each stope triangulation selected, a second, larger overbreak triangulation was created in Vulcan 3D software program. Dilution percentage and grade were then back calculated using the volumes and grades estimated in Vulcan for the original and overbreak triangulations.

The same procedure was used, for the secondary stopes except an allowance was made for overbreak into the backfill along each rib of the stope. The backfill is assumed to have zero grade. Dilution percentage and grade was again back-calculated using the volumes and grades estimated in Vulcan for the original and adjusted overbreak triangulations.

Dilution and recovery factors estimated for Platreef are shown in Table 16.30. These dilution and recovery factors are applied on a regional basis in preparing mineral reserve estimates and on a stope-by-stope basis in detailed production scheduling for the Platreef Project.

Table 16.30 Dilution and Mining Recovery Factors

Mining Method	% Dilution	Dilution Grade Factor	% Recovery
Longhole Stoping	8.5%	60%	95%
Drift-and-Bench Stoping	9.9%	60%	95%
Drift-and-Fill	5%	14%	95%

16.2.6 Mine Access Designs

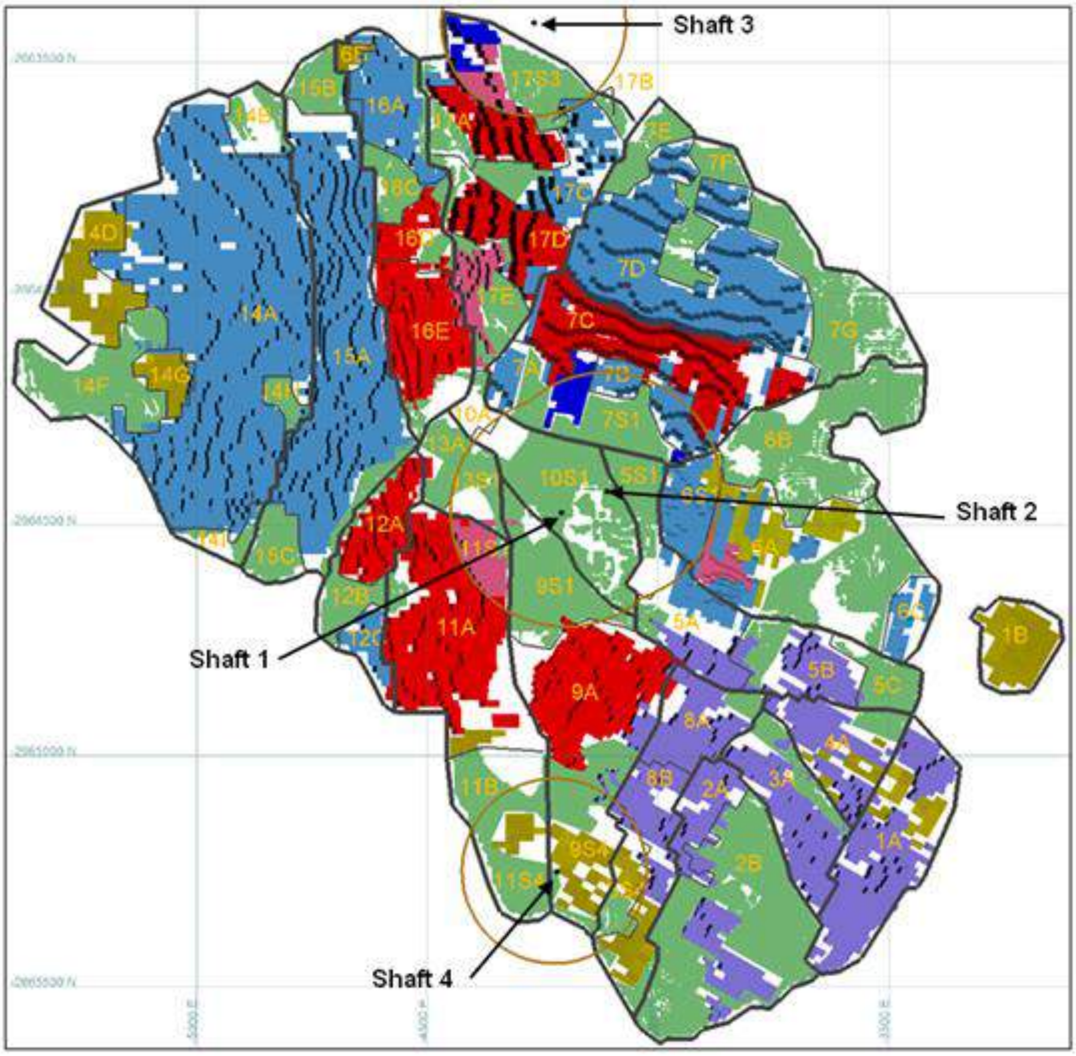
16.2.6.1 Shafts

Four vertical shafts are planned to provide access and ventilation to the mine. Shaft functions and design parameters are summarised in Table 16.31. Shaft locations are shown in plan view in Figure 16.18.

Table 16.31 Shaft Functions and Design Parameters

Shaft	Function	Diameter (m)	Depth (m)
1	Bulk sample / escape / ventilation intake	7.25	975
2	Production – 4Mtpa / service / ventilation intake	10.0	1,100
3	Ventilation Exhaust	7.25	975
4	Ventilation Exhaust	6.0	750

Figure 16.18 Shaft Locations



Primary access to the mine will be by a 1,100 m deep, 10 m diameter production shaft (Shaft No. 2). Secondary access to the mine will be via a 975 m deep, 7.25 m diameter ventilation / bulk sampling shaft (Shaft No. 1). During mine production, both Shaft Nos. 1 and 2 serve as ventilation intakes. Two ventilation exhaust shafts (Shaft Nos. 3 and 4) are planned. Shaft No. 3 will be a 975 m deep, with a 7.25 m diameter, and will be located near the northern edge of the mining area. Shaft No. 4 will be 750 m deep, with a 6.0 m diameter. During mine production, Shaft Nos. 3 and 4 serve as ventilation exhausts.

Shaft No. 1

Shaft No. 1 will initially be developed to collect a bulk sample of the underground ore for pilot testing. Sinking will continue below the ore zone to a depth of 975 m. The shaft will be equipped with a temporary loading station and will be used as a platform for early mine development prior to completion of sinking and commissioning the production shaft (Shaft No. 2).

Shaft No. 2

Shaft No. 2 will serve as the production shaft and primary access to the mine. The shaft diameter will be 10.0 m and the total depth will be 1,100 m. It will be equipped with skips, a service cage and an auxiliary cage. The current design incorporates 6.5 Mtpa hoisting capacity.

Personnel, equipment, and materials will be transported into and out of the mine via the service cage at Shaft No. 2. The cage will be sized to allow hoisting of fixed and mobile equipment and components with minimal disassembly.

Shaft No. 3

Shaft No. 3 will serve as a dedicated ventilation exhaust during mine production. It will also be used as a platform for early mine development for the northern portion of the mine. It will be located near the northern edge of the mining area.

Shaft No. 4

Shaft No. 4 will serve as a dedicated ventilation exhaust. It will be located near the southern edge of the mining area. The shaft diameter will be 6.0 m and the total depth will be 750 m. It will also be equipped with an emergency hoist and a 16 person capacity cage for emergency egress.

16.2.6.2 Main Access Levels

Three main access levels are established as primary haulage levels. They are targeted for initial development since they will provide early ventilation requirements for the ramps accessing the orebody from these levels. These haulage levels are designed flat and most of the ore passes terminate on or near one of these levels.

16.2.6.3 Access Ramps

Access ramps are designed to connect the haulage levels with the mining sublevels and to other infrastructure.

16.2.6.4 Mining Sublevels

The mining sublevels are normally accessed from the ramp system and driven in the footwall of the deposit approximately 25 m from the mineralised zone. The vertical spacing (15 m or 25 m) varies depending upon the height of the adjacent stope blocks. Typically, vent raises were incorporated at the terminus of these drifts to allow for adequate and efficient ventilation during production.

16.2.6.5 Ventilation Raises

A 3 m diameter exhaust raise will be raisebored south of Shaft No. 1 to provide a dedicated exhaust for underground shops and fueling stations. Additional internal ventilation raises will be developed throughout the mine to provide ventilation to the mining areas and to supplement the primary ventilation distribution system.

16.2.7 Mine Backfill

All proposed Platreef stoping methods require the use of backfill. Cemented backfill is used to fill open stopes and drift-and-fill cuts prior to mining adjacent stopes or cuts. Development waste may also be used for backfilling secondary stopes, when available. Mine backfill design and final cost information is based on recommendations provided by Golder and Associates.

For the first two years of production, and prior to the start-up of the mill and paste backfill plant, cemented rock fill (CRF) is used as the fill system and maintained as a backup system during later production. The waste rock required for the CRF backfill is sized and stored on surface in stock piles. These stockpiles are loaded into respective fine and coarse hoppers that discharge onto metering belts. The belts load the rock onto an aggregate conveyor, which delivers the waste rock underground via an 800 m long × 1.5 m diameter bored raise to an aggregate hopper. An apron feeder reclaims the rock from the raise and transports it to the mixer via another conveyor.

The CRF plant includes a colloidal mixing plant on surface, which combines fresh water, cement, and a chemical hydration retardant into a slurry. The slurry is then transported underground via a borehole and discharged into an agitated tank. The slurry is stored until pumped into a twin shaft continuous mixer containing the crushed and screened waste rock. When the desired formula is selected and continuous batching initiated, the system batches automatically on demand.

The CRF from the plant is discharged into haul trucks on the 750 Level and transported to the designated production stope.

The majority of the cemented paste fill (CPF) preparation system is located on surface, with underground components comprised only of the pipeline distribution system to the stopes. The paste plant process is designed to prepare a backfill product made from mill tailings alone and no additional components, such as sand or aggregate, are included in the process.

The 50 Bar positive displacement pumps deliver paste to the collars of two in-service boreholes that dip at 70°. In total there are four boreholes: two that service the northern area of the mine and two that service the southern area. Redundancy is built in by providing a second standby borehole to each area.

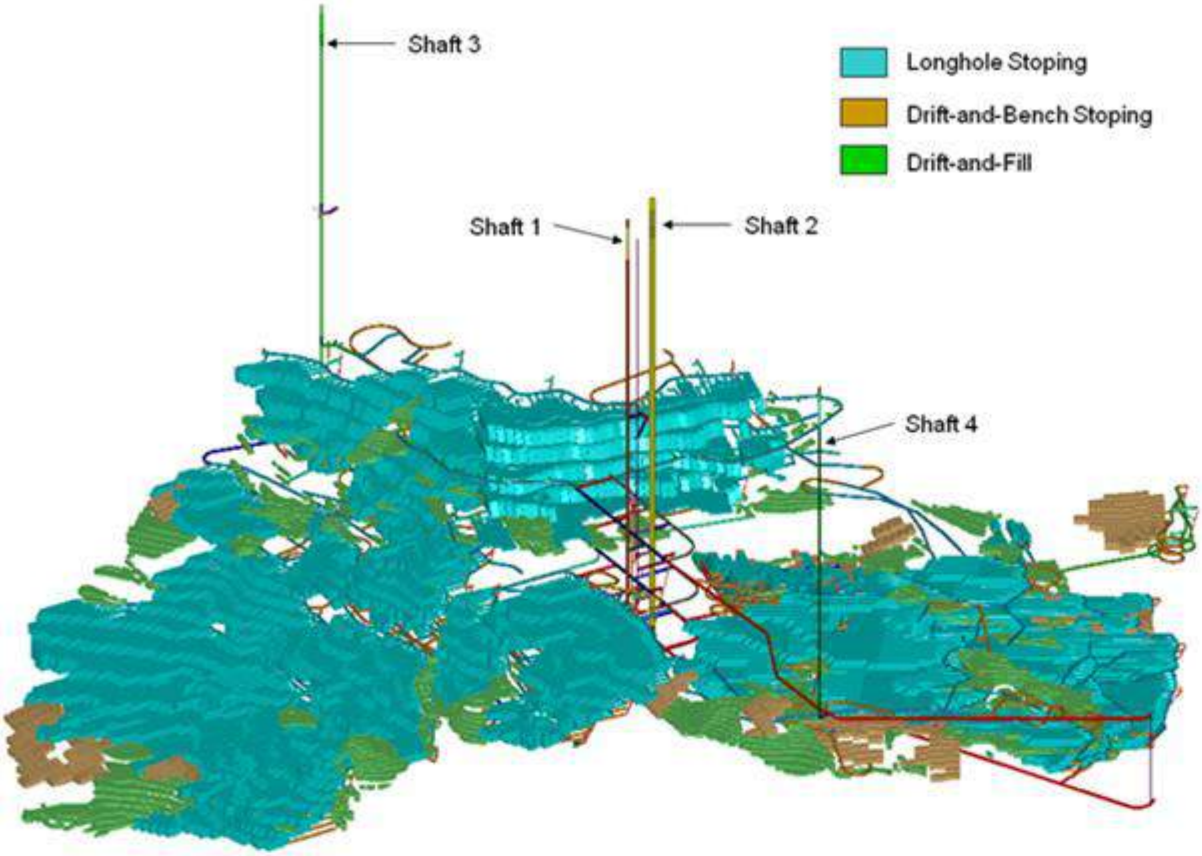
The main paste distribution lines underground are 203 mm Schedule 80 with Victaulic couplings. The main distribution lines feed 203 mm branch lines to supply paste to the stopes.

Essentially, the paste preparation process is split into two separate and somewhat independent preparation streams and underground distribution systems.

16.2.8 Mining Methods

Mining at Platreef will be performed using a combination of mining methods. Initially, ore zones with vertical thicknesses greater than or equal to 18 m will be mined using the longhole stoping method, while thinner ore zones will be mined using mechanised drift-and-fill or drift-and-bench methods. All methods require the use of backfill. An overview of the mining areas by method is in Figure 16.19.

Figure 16.19 Elevated View of Mining Areas by Method

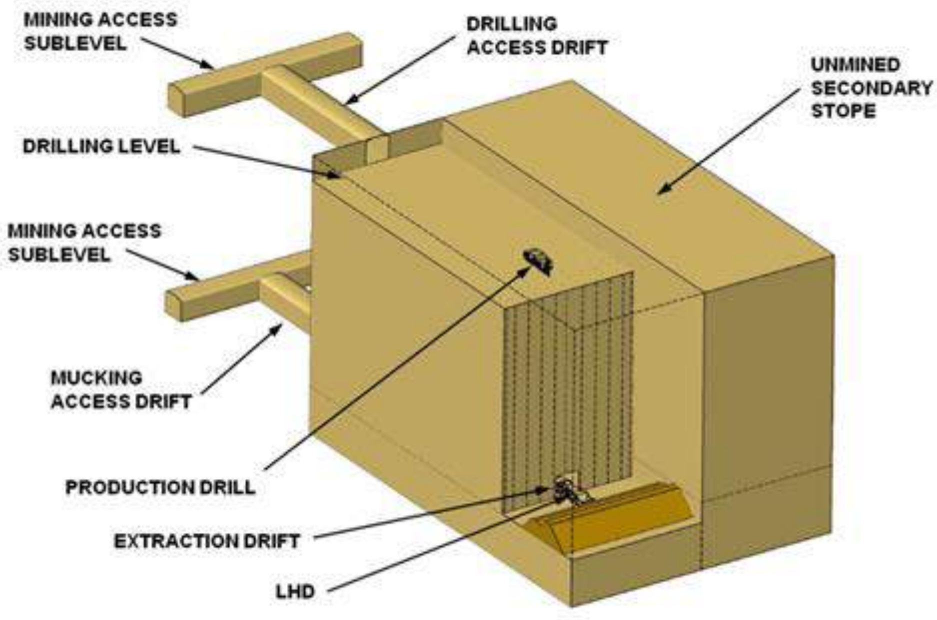


16.2.8.1 Longhole Stoping

The primary mining method selected for the Platreef project is transverse longhole stoping using paste backfill for post-mining support. The transverse longhole stoping method is a bulk mining method that provides good ore recovery and minimal dilution.

A typical transverse longhole stope layout is presented in Figure 16.20 . A drill drift will be developed along the top of the stope and an extraction drift will be driven along the bottom of the stope. The ore will be blasted on retreat from the far end of the stope to the entrance. Ore will then be extracted from the stope via the lower extraction drift using remote-controlled LHDs. After mining of the stope is completed, the open stope will be filled with paste backfill. The backfill will be allowed to cure before extraction of the adjacent stope blocks.

Figure 16.20 Typical Transverse Longhole Stope Layout



Stantec's designs assume a stope width of 15 m for all transverse longhole stopes. It is also assumed that a minimum vertical height of 18 m is required for longhole stopes to ensure the stability of the ground between the drill drift and the extraction drift. Stope lengths will vary depending on the thickness of the ore zone.

Stantec's stope designs were prepared using the assumption that longhole stopes are oriented transversely (perpendicular to) the strike of the mineralised zones. Mining access sublevels (sublevels) will be driven subparallel to the strike of the ore zones in the footwall of the deposit. Drilling and mucking access drifts will then be driven from the sublevels to the top and bottom of the stope. The sublevels and access drifts will be driven at 5 m wide x 5 m high.

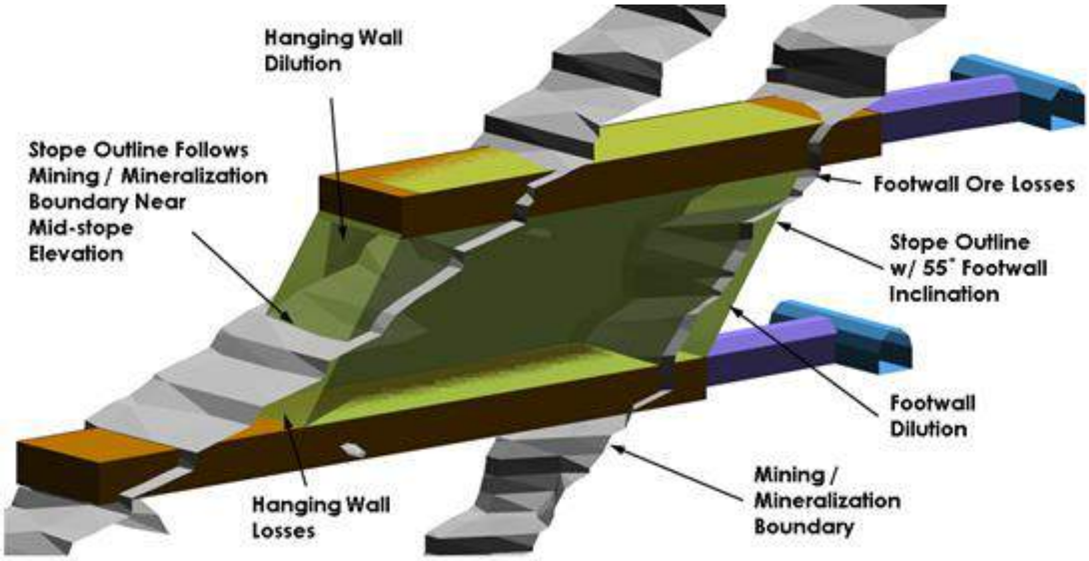
The drilling level at the top of the stope will be developed at 15 m wide x 5 m high. The extraction drift along the stope bottom will be developed at 15 m wide x 5 m high. The later assumption was made in order to simplify the stope design process.

Stope drilling requirements were determined using Orica Mining Software. After mining of the stope is completed, it will be backfilled with cemented fill. CRF will be used for the first one to two years of production, but paste backfill will be used after the mill and paste backfill plant are commissioned. Waste rock will be used as backfill in secondary stopes whenever possible.

Longhole stopes will be mined using a primary and secondary extraction sequence. The primary stopes will be mined and backfilled first, followed by the mining and backfilling of the secondary stopes.

The dips of the ore zones at Platreef vary from nearly flat to as much as 45° to 50°. The stope end walls on the footwall (entrance) side of the stope are designed with a minimum inclination of 55° in order to assure proper flow of ore along the footwall to the extraction level at the stope bottom. Significant dilution and ore losses will occur on the footwall since the dip of the ore zone will almost always be less than 55°. The hanging wall end will be drilled as flat as possible in order to minimize dilution and ore losses. Figure 16.21 illustrates these points.

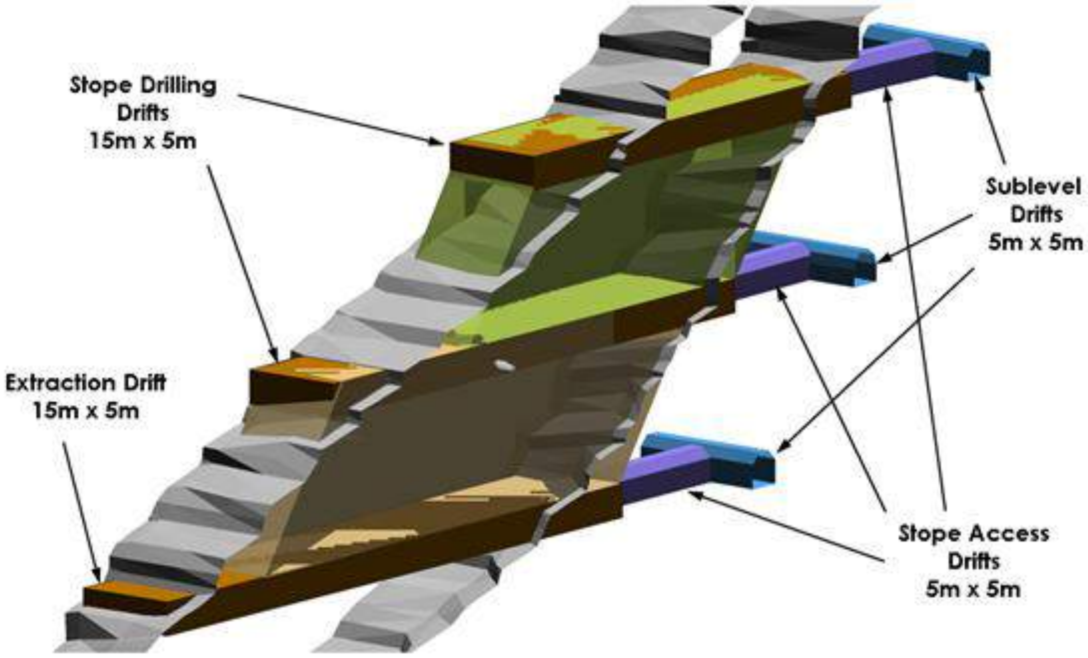
Figure 16.21 Planned Dilution and Ore Losses on Stope Ends



The overall length of individual stopes will vary with the width of the ore zones. The maximum length left open at one time for an individual stope will be determined using the stope height and hydraulic radius criteria provided by SRK. If the stope is longer than this maximum open length, the stope will be broken into shorter segments. Starting at the end of the stope, each stope segment will be mined and backfilled prior to mining the next segment. Mining will continue in a retreating fashion toward the deposit footwall until the remainder of the stope is extracted.

Figure 16.22 shows an example transverse longhole stope configuration. In such cases, where one stope is located immediately above another, the lower stope is mined first and backfilled to the floor elevation of the drilling drift. When the upper stope is mined, the drilling drift for the lower stope then becomes the mucking drift for the upper stope.

Figure 16.22 Example Stope Configuration in Moderately Dipping Ore Zones



Stope sequencing requires that the top stope is mined and filled prior to mining the lower stope (secondary) adjacent to the first (primary). This is because the drill level for the lower secondary cannot be excavated until the Extraction Level for the upper primary is filled.

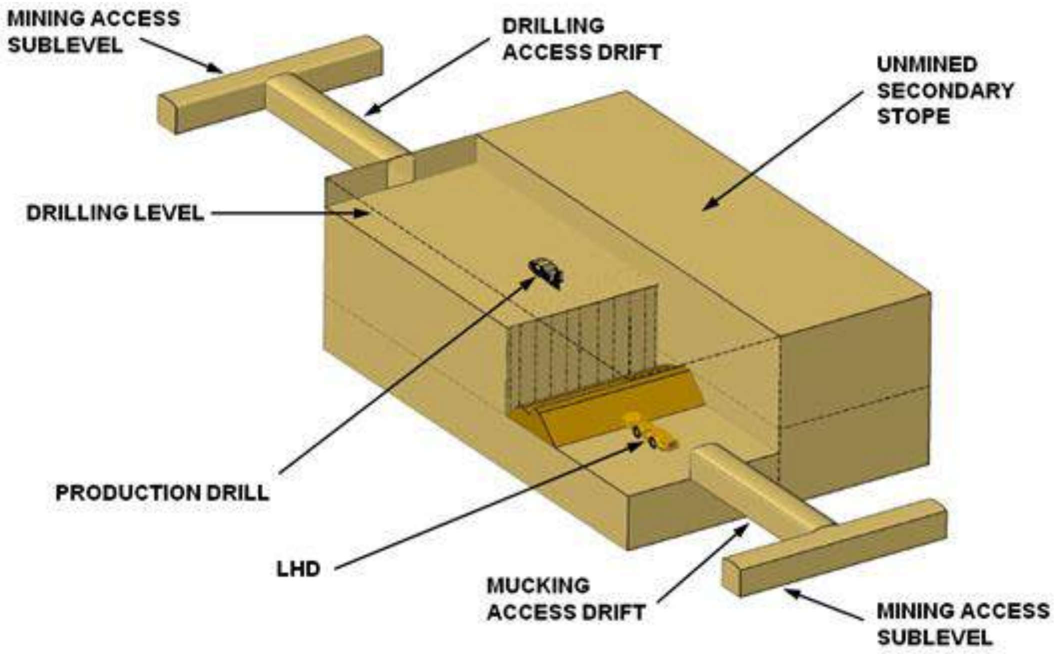
16.2.8.2 Drift-and-Bench Stoping

Drift-and-bench stoping is a variation of the longhole stoping method. It is used in areas where the ore zone is thick enough to allow the use of a production drill but does not have sufficient vertical thickness to allow for development of full length drill and extraction drifts with a stable bench in between.

Drift-and-bench stoping is similar to transverse longhole stoping in that a drill drift is developed at 15 m wide x 5 m high at the top of the stope. Ore is extracted from the stope via a lower extraction drift using remote-controlled LHDs. After mining of the stope is completed, the open stope will be backfilled with cemented paste fill, CRF or in the case of secondary stopes, excess waste rock when available.

A typical drift-and-bench stope layout is presented in Figure 16.23.

Figure 16.23 Typical Primary Drift-and-Bench Stope



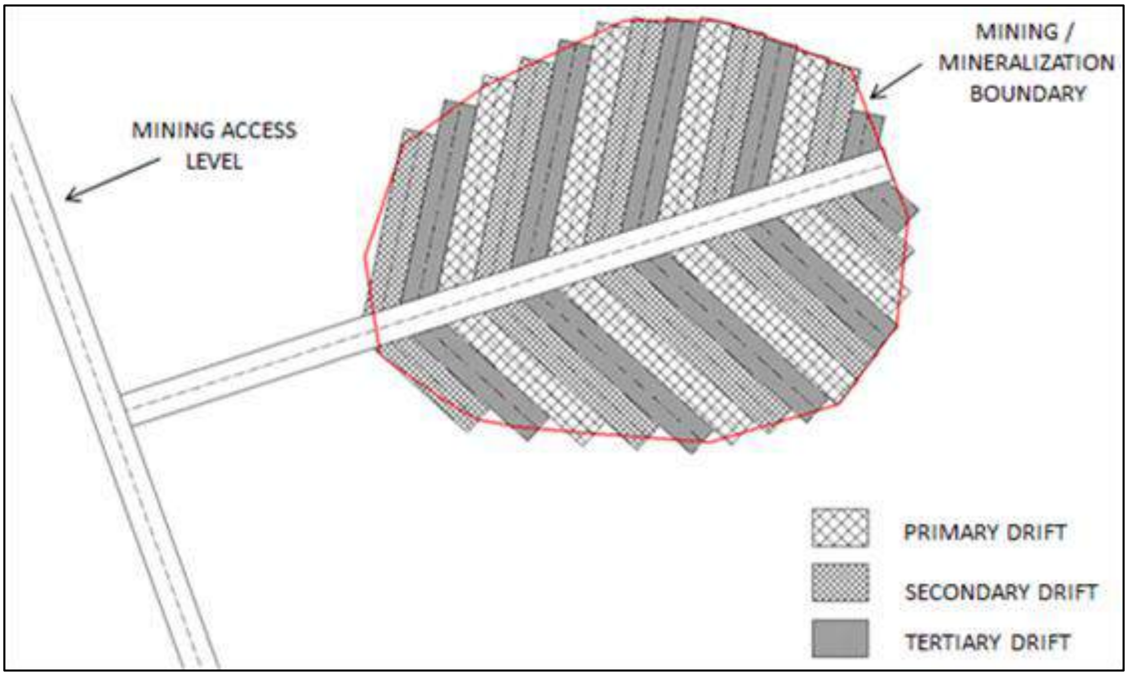
16.2.8.3 Drift-and-Fill Mining

Drift-and-fill mining is a variation of the cut-and-fill mining method. For the purposes of this report, the term "drift-and-fill" is used.

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. Mining progresses in this manner in a chevron pattern until the entire slice of ore is depleted. Where ground conditions permit, mining can be performed using a primary secondary or primary secondary-tertiary sequence, enabling access to multiple mining faces at all times and allowing greater productivity from an individual ore slice.

Drift-and-fill techniques will be used in ore zones where the vertical thickness ranges from 4 m to 10 m. Figure 16.18 shows a typical drift-and-fill layout.

Figure 16.18 Typical Drift-and-Fill Level Layout



Drift-and-fill mining is a flexible mining method that allows near complete recovery of the ore zones. Mining of the ore is completed with the same equipment used for mine development and dilution from waste external to the ore zone is minimal. Negatively, productivity is relatively low due to the small blast sizes, which results in small tonnages per cycle and a higher mining cost per tonne. Good control of drilling and blasting is also necessary to minimize dilution from backfill. When compared with other mining methods, operating costs for drift-and-fill are relatively high due to the lower productivity.

16.2.9 Mine Design Parameters

The mine design parameters are based on geotechnical recommendations provided by SRK. The geotechnical information is utilized for determining basic drive dimensions, stope width and height parameters, drill-and-blasting designs, and ground support requirements. The conceptual infrastructure designs are based on similar infrastructure designs from other projects with comparable production requirements. For mine backfill, Golder and Associates provide recommendations for the plant designs and capital and operating costs.

Identification of the mineable portions of the Platreef Resource was accomplished using the Vulcan three-dimensional underground design software package and the Stope Optimiser version 1.0 software from Maptek Pty Ltd and Alford Mining Systems as well as Stantec's proprietary Vertical Miner software tool. The method was to first use the Stope Optimiser software to generate 25 m high and 15 m high longhole stopes in Vulcan. Stope Optimiser and Vulcan were then used to make bench stopes of intermediate height between 10 m and 18 m high. Finally, Stantec's Vertical Miner software tool and Vulcan were used to create 5 m high horizontal drift-and-fill slices in the thinnest zones. Table 16-26 lists the key Stope Optimiser parameters used for the Platreef project.

Table 16.32 Key Stope Optimiser Parameters

Parameter	Longhole Stopes	Bench-and-Fill Stopes
Optimisation Direction	Horizontal along Stope (mining direction)	Vertical (optimizing thickness mined)
Stope Profile (Zones 7 and 17)	25 m high x 15 m wide	30 m long x 15 m wide
Stope Profile (other zones)	15 m high x 15 m wide	30 m long x 15 m wide
NSR Cut-off	\$100 and \$80	\$80
Maximum Waste Fraction	30%	30%
Stope Minimum Length	15 m	–
Stope Minimum Height	–	10 m
Minimum Dip Angle	55°	85°
Maximum Dip Angle	90°	95°
Maximum Strike Angle	±5° to ±10°	±5° to ±10°

Drift-and-bench stopes are planned in areas where an ore layer is too thin for longhole stoping but is thicker than two stacked 5m drift-and-fill lifts. Drift-and-bench stopes were generated using the Stope Optimiser at an \$80 NSR cut-off. For drift and-bench stopes, the grid of stope profiles is horizontal and the stopes were optimised in the vertical direction for optimum stope elevation and height using the same types of parameters that were used for the longhole stopes, as shown in Table 16.32.

Drift-and-fill horizontal slices, 5 m thick, were generated in areas where the ore layers are too thin to be mined by either longhole or drift-and-bench stoping. For the final resource, the drift-and-fill parameters are \$80 NSR cut-off, maximum 33% internal dilution, and minimum thickness of 5 m.

The mineable shapes generated in Vulcan were transferred to Studio 5DP software by CAE. Studio 5DP and CAE's EPS software was used for scheduling.

16.2.10 Mine Equipment Requirements

16.2.10.1 Equipment Criteria

Criteria considered in equipment selection included suitability, equipment standardization, and cost. The equipment selection process was iterative and aimed at obtaining the optimum equipment required to achieve the planned development and production quantities and rates.

The equipment requirements for the Platreef project are split into two categories, fixed equipment and mobile equipment. The equipment requirements for each category cover the major components for the operation. The following are the design criteria for sizing, selecting, and quantifying fixed and mobile equipment.

- Mining Method
- Mine Production Rate 4 Mtpa
- Ventilation Requirements
- Mine Design Criteria

Costs for mobile and fixed equipment are based on the following criteria.

- Mobile equipment quantities, purchases, and rebuild schedules are per the Platreef LOM plan.
- Contractor will provide own equipment at market rental rates.
- Haulage truck prices from DRA and Stantec data.
- Recent costs for utility equipment from DRA and Stantec data.
- Fixed equipment prices are derived from vendor quotations, DRA-provided information, and Stantec data.

16.2.10.2 Mobile Equipment

The mobile equipment required for lateral development includes drill jumbos, LHDs, haul trucks, and ground support equipment.

The mobile equipment selected for the Platreef project was based on the criteria mentioned previously and information from similar projects. The mobile equipment quantities are estimated at a Prefeasibility-level and are based on historic performance rates and production and development crew requirements. Operating mobile equipment requirements are estimated for the projected LOM production and development schedules.

The following three situations distinguish how the quantity of mobile equipment to be purchased was determined for the capital cost estimates.

1. Where a fleet of equipment (e.g., jumbo drills, underground loaders, haul trucks) is required, additional units are purchased to provide standby coverage associated with expected availability. Mechanical availability of 85% was assumed for all mobile equipment.

2. In situations with utility vehicles where one unit or less is required, one is purchased.

3. In situations where more than two units are required, three are purchased.

Table 16.33 provides a list of selected equipment; maximum quantity required LOM and does not include replacement equipment. The overall quantities fluctuate over the LOM to match the production and development schedule requirements at any given time.

Table 16.33 Mobile Equipment (Maximum Operating Quantities)

Mobile Equipment	Total Units
Drill Jumbo – Two Boom	8
Drill Jumbo – Single Boom	7
Drill Jumbo – Single Boom (cable bolting) Cable Bolter	2
Production Electric / Hydraulic Drill	3
LHD – 5.4 m3 Ejector Bucket, without Remote Package – Development	5
LHD – 8.4 m3 with Remote Package – Production	13
Haul Truck	13
Shotcrete Jumbo	3
Explosives Truck / Jumbo	5
Scissor Lift Truck	4
Personnel Carrier	35
Underground Road Grader	2
Supply Vehicles	5
Lube Truck	2
Fuel Truck	2
Flatbed Truck	2
Water Spraying	1

16.2.10.3 Fixed Equipment

A list of major fixed equipment by category for Platreef is presented in Table 16.34. This list includes fixed equipment for shaft and hoisting, a compressor plant, water handling, electrical, material handling, ventilation, an underground shop, fuel bays, safety, and miscellaneous.

Table 16.34 Fixed Mine Equipment (4 Mtpa)

Shafts and Hoists	Material Handling
Production Hoist Service Hoist Auxiliary Hoists (2) Head Frames (2) Hoist Wire Ropes, Sheaves, and Rope Attachments Loading Pocket Production Skips Dump Chutes Service Cage Auxiliary Cages (2)	Jaw Crushers (2) Primary Crushing Overhead Crane Transfer Conveyor to skip loading pockets Metal Detectors Belt Magnets Belt Scales Stationary Rock Breakers
Compressor Plant	**Ventilation**
Air Compressors Air Pre-filter Colling Water Pump Purge Tank, Filter, and Water Monitor Heat Exchanger & Cooling Fan Refrigerated Dryers Receiver Tanks Receiver Auto Drains	Primary Ventilation Fans – 2,000 HP Development/Production Fans – 100 HP Development/Production Fans – 50 HP Air Doors Shop Air Doors – Roll-up
Water Handling	**Safety and Miscellaneous**
Main Dewatering Pumps Clarifying Sump Mud Pumps Service Water Pumps Development Pumps	Engineering Equipment (Survey, Lasers, etc.) Maintenance Shop Equipment Shop Monorail Crane – 25T Shop Monorail Crane –10T Jib Crane – 5T Fire Extinguisher First Aid Kit Miscellaneous First Aid Supplies Mine Rescue Equipment for 15 Persons Cap Lamps & Chargers Self-Rescuers Miscellaneous Sanitary Supplies and Units Refuge Chambers
Electrical	**Underground Shop**
Main Substations U/G Substation Mine Load Centres Main Vent Fan Substations Work Shop Substation Compressor Substation Crusher Substation Leaky Feeder Radio System Data and Control (SCADA)	Shop Fixed Equipment and Tools **Fuel Bay** Fuel Bay Fixed Equipment and Storage Tanks

16.2.11 Personnel

Personnel requirements were developed to support planned development, construction, and operations requirements for the mine. Only personnel directly linked to the operation of the mine are included in this study. Personnel shared with other activities on the Platreef Project like accounting, training, personnel management, environmental, permitting, housing, security, ambulance, etc. are not included. Personnel requirements are not determined for the following factored personnel in this study.

- Owner's Project Team
- EPCM Team

16.2.11.1 Criteria

Direct and indirect labour requirements were established to suit the selected mining method, support systems, and general mine requirements during mine development, construction, and operations. Personnel requirements are based on an operating schedule of 8 hours per shift and three shifts per day. Contractor shaft crews will work 300 days per year and contractor non-shaft crews will work 360 days per year. Owner capital work and production are accomplished in 8 hr shifts per day and 360 days per year. Non-staff hourly personnel will work an extra day a week approximately every other week with a daily scheduled quarter hour for a hot shift change at the service shaft bottom at the end of each shift. An additional 5% unscheduled overtime allowance was added for a typical work week of 49.5 hours.

It is understood by Stantec that the training of the workforce is an extremely important requirement for the success of the Platreef Project. Ivanhoe is developing a site wide training program. Initial training is not included in the mine estimate and assumes a trained workforce that requires only periodic refresher courses.

16.2.11.2 Classification Descriptions

Personnel are presented as direct and indirect staffing for both contractor and owner teams. Staffing is further classified as hourly and staff in an effort to allocate costs for both classifications.

16.2.11.3 Owner's Project Team

The Owner's Project Team oversees the work performed by the contractor and coordinated by the EPCM contractor. The Owner's Project Team comprises owner staff and hourly employees and is assumed to contain the senior personnel responsible for the smooth operation of the mine following the completion of contractor activities. All production activities will be performed by owner personnel.

16.2.11.4 Contractor's Team

Contractor staff and hourly workers will be retained to perform short duration projects requiring special skills. This work includes, but is not limited to, shaft-related surface construction, primary and secondary mine development prior to Year 2021, shaft construction, underground construction during the preproduction period, delineation drilling, and raiseboring.

16.2.11.5 Personnel Allocations

Additional allocations of personnel are included in the overall staffing. They are the following.

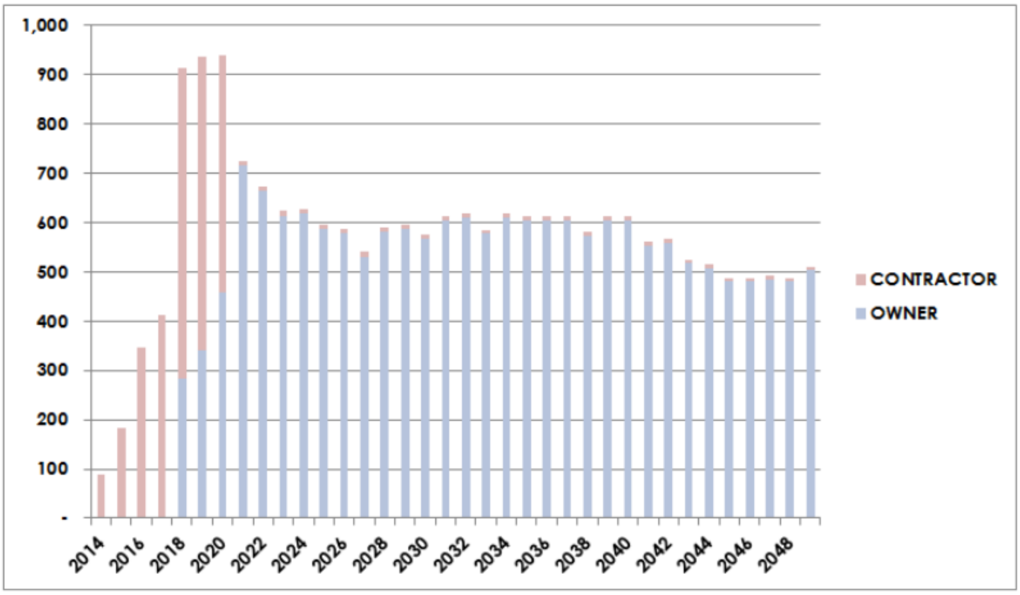
Payroll Manpower – This category represents the daily personnel and the personnel that are on rotation but need to be included in the overall personnel count in an effort to identify initial purchase numbers of mine lamps, mine lamp chargers, and self-contained self-rescuers required to support the total labour requirements. This includes direct supervisors as well as the crews they manage.

Vacation, Sickness, Absenteeism, and Training Allocation – This allocation represents a "miner's pool" that would be required on site to cover hourly labour during times when individuals are on vacation, sick, absent, or in training. This amounts to approximately 15% of the annual hourly personnel requirements.

16.2.11.6 Staffing Schedule

The annual contractor and owner underground payroll personnel requirements are presented in Figure 16.24.

Figure 16.24 Mining Annual Contractor and Owner Personnel Requirements



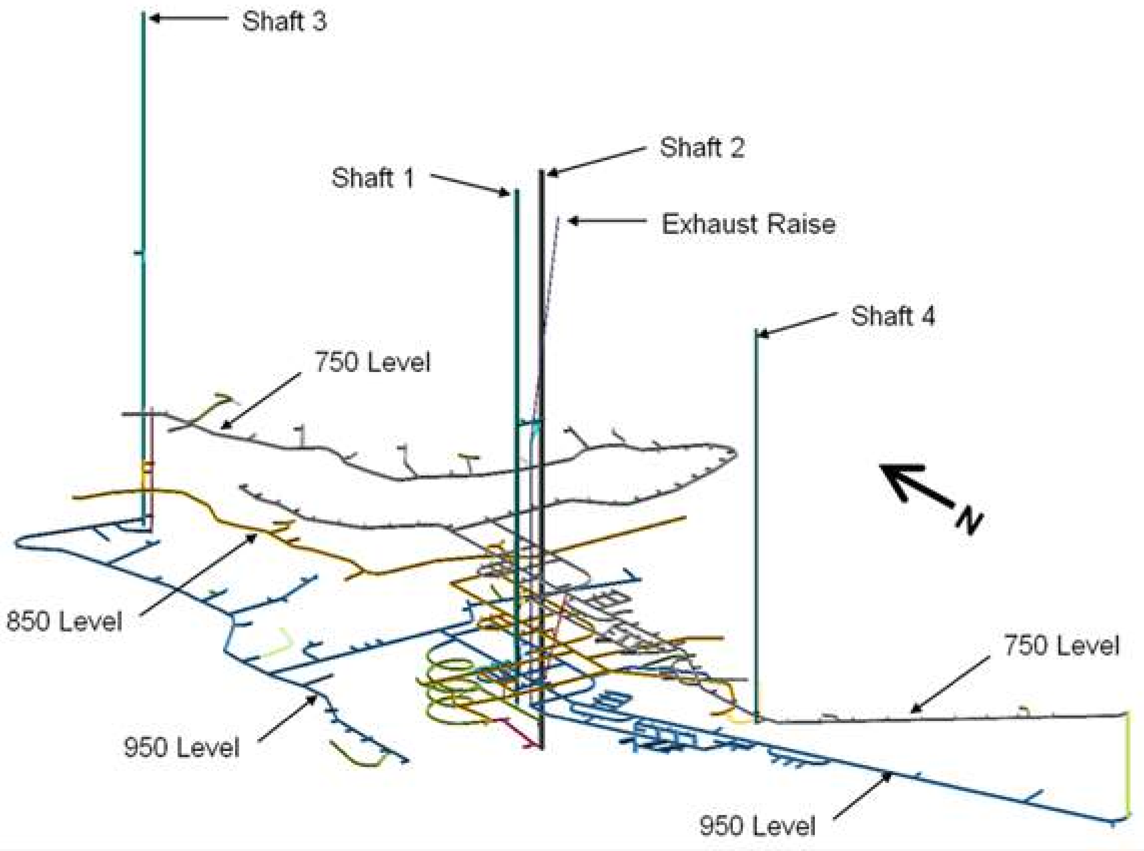
16.2.12 Mine Development Plan and Schedule

Critical activities in developing the underground mine and ramping up to full production include the following items.

- Sinking and commissioning of the four shafts.
- Completion of required early ore characterization work and delineation drilling activities.
- Establishment of sufficient ventilation for early mine development activities.
- Development and construction of critical surface and underground infrastructure.
- Development of critical access drifts and ventilation airways.

Figure 16.25 shows an elevated view of the locations of the shafts and main access levels.

Figure 16.25 Mine Access Layout

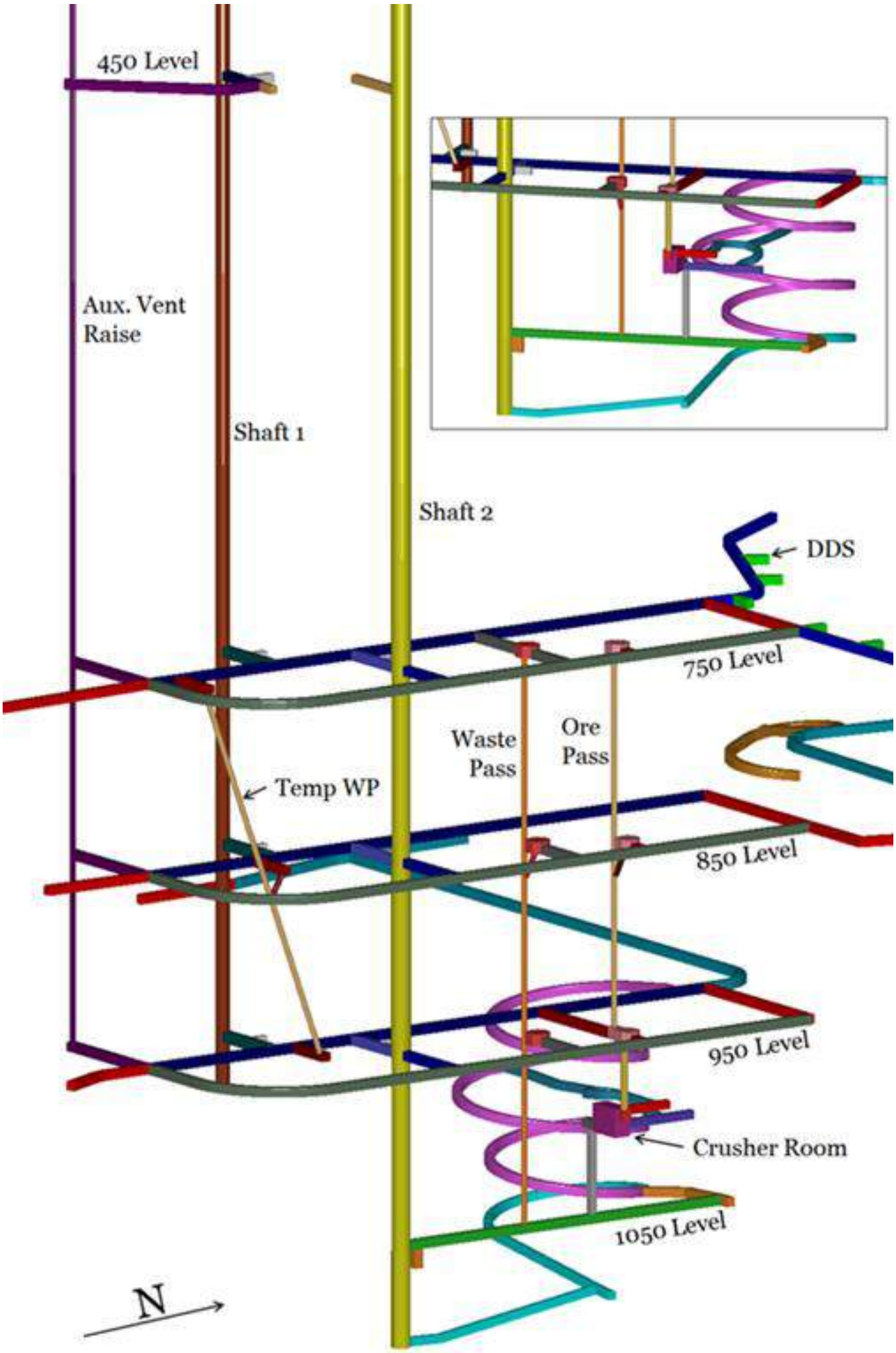


The mine schedule is based on an April 2014 start of construction on Shaft No. 1. Shaft sinking contracts are in place, early engineering activities are in progress, and it has been assumed that construction of Shaft No. 2 will begin in January 2015, Shaft No. 1 commissioning will be completed in May 2016, and commissioning of the production shaft (Shaft No. 2) will be completed in June 2018 (two years after Shaft No. 1).

In order to maximize the opportunity to commence early level development from Shaft No. 1, a temporary loading station will be installed that will allow for hoisting of up to 1,500 t of development waste per day using the shaft sinking buckets. During this time, shaft stations will also be developed. After commissioning of the shaft is completed, a temporary waste pass will be bored. The waste pass will enable development to be done concurrently from Shaft No. 1 prior to completion of the commissioning of Shaft No. 2.

Much of the infrastructure required for the material handling system, mine infrastructure, and orebody access will be developed from Shaft No. 1. Development resources will initially focus on development of orebody delineation / diamond drilling stations to the north and south and on excavations for the material handling facilities for Shaft No. 2. Critical development items for the materials handling facilities include the shaft stations, main ore and waste passes, truck dumps, crusher room and associated coarse and fine ore bins, and loading conveyor drift. Refer to Figure 16.26 for a detailed view of excavations required for the material handling system.

Figure 16.26 Material Handling System



Secondary development objectives will be driving of the main haulage levels to connect with Shafts No. 3 and 4 and, in the process, establishing the ventilation circuit to support mine production. Another secondary objective is development for production-related infrastructure such as maintenance shops, fueling stations, explosives magazines, backfill facilities, etc.

Shaft No. 4 will be sunk upon completion of sinking Shaft No. 2. Concurrent lateral development will reach Shaft No. 4 at approximately the same time Shaft No. 4 sinking is complete. This southern connection will complete another primary ventilation circuit.

Commissioning of the crusher and materials handling systems for the production shaft will occur in June 2018 and initial ore production from stope development will begin in October 2018. Production from longhole stoping will begin in December 2018.

Table 16.35 presents major milestones in the preproduction development and construction schedule.

Table 16.35 Preproduction Development Milestones

Milestone	Date
Begin Shaft No. 1 Construction	Apr 2014
Begin Shaft No. 2 Construction	Jan 2015
Complete Shaft No. 1	May 2016
Complete Sinking Shaft No. 2	Sep 2017
Commission Shaft No. 2	Jun 2018
Complete Shaft No. 3	Jun 2018
Complete Shaft No. 4	Jul 2019
Commission Materials Handling System	Apr 2018
Begin Stope Development	Oct 2018
Begin Stope Production	Dec 2018

16.2.12.1 Development Productivity Rates

Development rates were calculated from first principles. SRK supplied ground support requirements. The support criteria were used in the establishment of the advancement rates. Table 16.36 lists the heading types and the advance rates for each. These rates were used in the EPS schedule.

Table 16.36 Lateral Development Rates

Heading Size		Performance (m/day)	
Width (m)	Height (m)	Single	Double
5.0	5.0	5.0	6.75
15.0	5.0	N/A	1.4

Development crews drive multiple headings whenever possible, and by doing so, increase utilization of crews and equipment. Double heading estimates are prepared based on a 35% increase over the single heading rate for the same drift size.

All internal ventilation raises and ore passes are designed to be raisebored. All raiseboring assumes that the drill rigs, drill pipe, bits, reaming heads, and crews are on site and available as necessary. Vertical advance rates exclude mobilization and demobilization of the raiseboring rig and crews. Advance rates are applied in accordance with raise diameter and length. Table 16.37 lists a summary of the vertical advance rates. Finger / drop raises are also incorporated into the design and schedule. These will be drilled with a production drill and blasted via methods similar to those used in creating longhole stope slot cuts.

Table 16.37 Vertical Advance Rates

Heading Description	Finished Size (m)	Excavation Performance (m/day)
Shaft Sinking		
Shaft No. 1	7.25	2.84
Shaft No. 2	10.0	2.29
Shaft No. 3	7.25	2.84
Shaft No. 4	6.0	2.84
Other Vertical Development		
Vent Raise – Internal and Auxiliary Surface	3.0 m diameter	6.0
Orepasses – Internal (excluding lining)	3.0 m diameter	6.0
Crushed Ore Bin	4.0 m diameter	3.38
Finger / Drop Raise	3.0 m diameter	3.0

16.2.12.2 Development Quantities

During mine development and production, a total of approximately 143,400 m of waste development or 11,721,000 t of waste material is mined. LOM waste development is summarised in Table 16.38.

Table 16.38 Life of Mine Waste Development Summary

Description	Units	Total
Shafts*		
Advance	m	5,041
Waste	t	725,956
Lateral Development		
Advance	m	96,020
Waste	t	7,855,125
Stope Accesses		
Advance	m	35,199
Waste	t	2,970,546
Vertical Development		
Advance	m	7,141
Waste	t	169,435
Total Waste Development		
Total Advance	m	143,401
Total Waste	t	11,721,062

16.2.13 Ore and Waste Handling Systems

16.2.13.1 Ore and Waste Passes

Secondary ore passes were integrated into the production development plan whenever applicable. The exception was in areas where the depth of the development was significantly deeper than the production shaft with no waste development at depths below and the dip of the orebody was too shallow for economical implementation. In the other areas, orepasses were included at regular intervals with most drawpoints being located as close to the primary haulage levels as was practicable to reduce equipment traffic in and around the production headings. Keeping truck traffic out of the production workings also reduced local ventilation requirements.

Chutes at the bottom of the satellite orepasses will be used to load trucks for the transfer of material to the three primary ore passes (or the three primary waste passes if the truck is hauling waste). All six central ore and waste passes will be equipped with permanent rock breakers and grizzlies.

16.2.13.2 Crushing Facilities

The ore and waste handling system is designed to handle the production requirement of 4 Mtpa of ore and 0.54 Mtpa of waste with design allowances for future expansion. A jaw crusher will be used to crush coarse ore to minus 150 mm material.

The waste handling system will be comprised of a 3 m diameter borehole with three feed locations equipped with 300 x 300 mm grizzlies and rock breakers. The borehole will report to the load-out conveyor on the 1050 Level.

The 1050 Level load-out system will be comprised of apron feeders at the bottom of the crushed ore bin and waste pass and a 2,100 mm wide high-speed load-out conveyor Variable speed drives are incorporated in the conveyor design to facilitate low speed operation during conveyor loading and high speed operation during conveyor discharge into the shaft loading pocket.

16.2.14 Mine Ventilation and Cooling Design

16.2.14.1 Introduction

The purpose of the mine ventilation system will be to provide sufficient air in quantity and quality to support the climate conditions of underground openings and, most importantly, to maintain the working condition at acceptable level and in accordance to regulations.

The ventilation design is based on proven and internationally recognised principles. The prime objective is to provide a safe and healthy environment 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 were used.

The annual ventilation estimate was developed based on requirements for mobile equipment and in conjunction with heat load calculations and refrigeration requirements. As a pull system, fresh air will be downcast via the main shafts and exhausted through a number of internal bored raises, and the main exhaust shafts. A total airflow of 1,157 m3/second was determined to be required at full production.

RME was tasked with the mine air cooling and refrigeration evaluation and design studies. RME's study identified surface BAC system as the best available option for the Platreef Project.

An overall heat load of 56,806 kW was calculated for the mine. Considering the air cooling capacity from the ventilation system, 20,000 kW of mechanical cooling will be required for maintaining a reject wet bulb temperature of 28.5°C throughout the mine. The proposed BAC refrigeration plant in this study is designed for 26,500 kW of refrigeration capacity.

16.2.14.2 Method and Design Criteria

The ventilation system is an exhaust pull system. The two main intake shafts (Shaft Nos. 1 and 2) will provide fresh air, while Shaft No. 3 (in the north) and Shaft No. 4 (in the south) will be exhausts. Fresh air will be further distributed through a series of main stations, ventilation raises, major ramps, and haulages. Regulators will be used to direct air to the active mining areas.

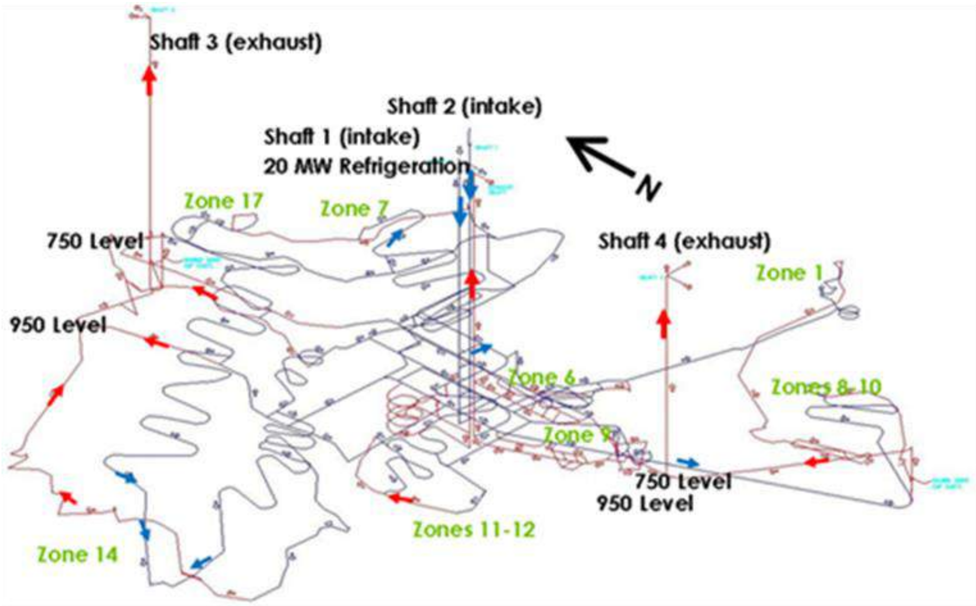
Key points of the ventilation strategy and design criteria for the Platreef project at the full production can be summarised as follows.

- The ventilation system, and therefore the cooling supplied from surface, is designed on a "once-through" basis, with series ventilation and recirculation being avoided. Shaft Nos. 1 and 2 will be the main intakes, with Shaft No. 1 designated as the dedicated refrigeration and cooling provider.
- Production haulages will be supplied with no less than 42 m3/second of air (based on one truck and one LHD per level).

- Main access ramps will be 5 m x 5 m in cross section and will serve as the main distribution hub for fresh air throughout the mine, along with internal raises and main haulage levels.
- The main fan motors will be variable speed controlled, thus the change in quantity and pressure will be attained to meet production build-up.
- At the main levels, a quantity of 45 m^3/second (20% leakage included) will be allowed for during the primary development on the initial three levels.
- All vent raises will be bored at 3 m Ø.
- There will not be a maximum velocity cap for the Shaft No. 1 facility, although the velocity will be dictated by pressure drop over the system and main fan capabilities.

Figure 16.27 presents the simplified version of the ventilation network at its full production.

Figure 16.27 Mine Ventilation Layouts in the Full Production Stage



Blue lines are intake fresh air, and red lines are return air.

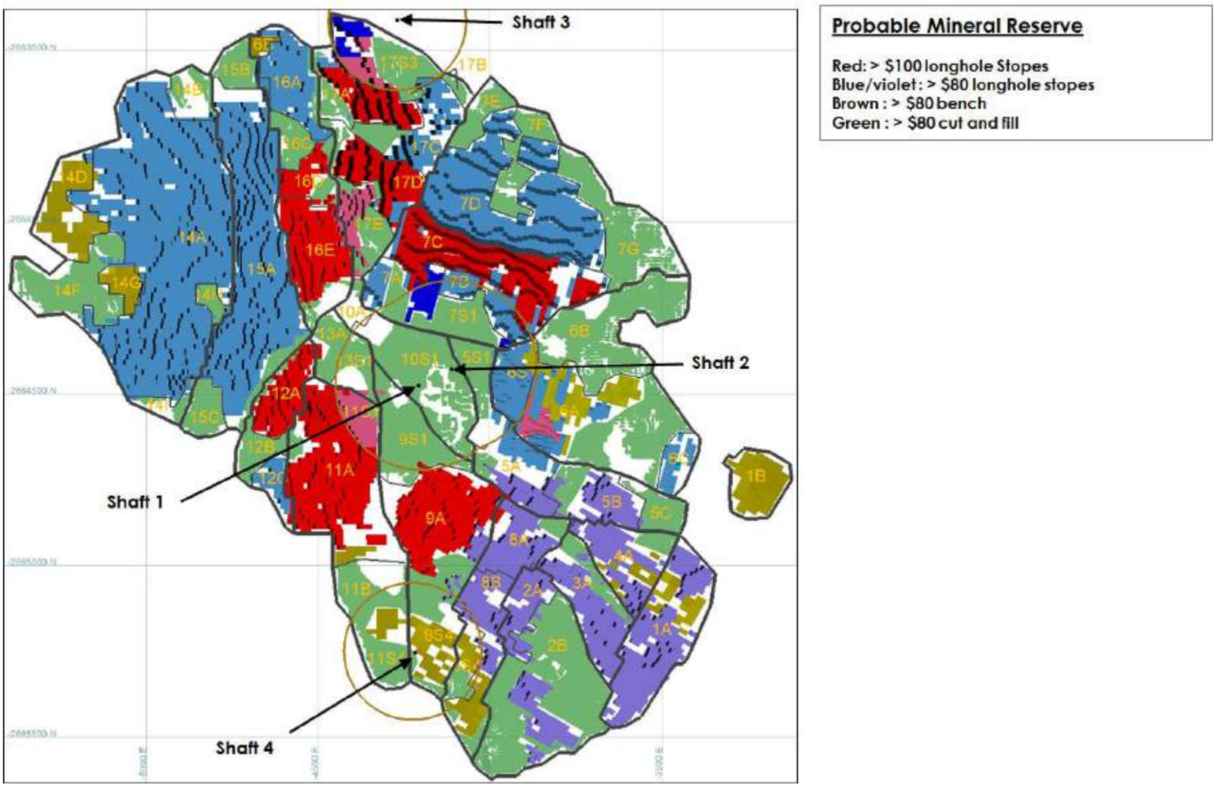
16.2.15 Production Plan

16.2.15.1 Production Planning Criteria

Stantec’s Mineral Reserve estimates included detail by mining methods for each of the mining zones and subzones shown in Figure 16.24. These detailed estimates were used to determine production sequencing. The following general planning criteria were applied to determine priorities for initial production.

- Shaft Pillars are not Extracted Until the End of the Mine Life
- Proximity to Shafts and Early Development
- Highest Grade
- Highest Productivity
- Lowest Mining Cost

Figure 16.28 Location of Mining Areas



Longhole stoping offers the lowest operating cost and the highest productivities; therefore, development resources were focused on the development and extraction of higher grade longhole stoping areas. Drift-and-bench and drift-and-fill mining were delayed until later in the mine life.

Productivities used in preparing production schedules are shown in Table 16.39.

Table 16.39 Stoping Productivities

Stoping Method	Rate	Units
Develop Drilling Drift – 15 m wide x 5 m high (including ground support)	1.44	m/day
Longhole Stoping – 15 m high (slot, bench, fill, fill cure)	385	t/day
Longhole Stoping – 25 m high (slot, bench, fill, fill cure)	455	t/day
Drift-and-Bench – 14 m high x 60 m long (slot, bench, fill, fill cure)	140	t/day
Drift-and-Fill (advance and fill)	145	t/day/heading

16.2.15.2 Ramp-Up

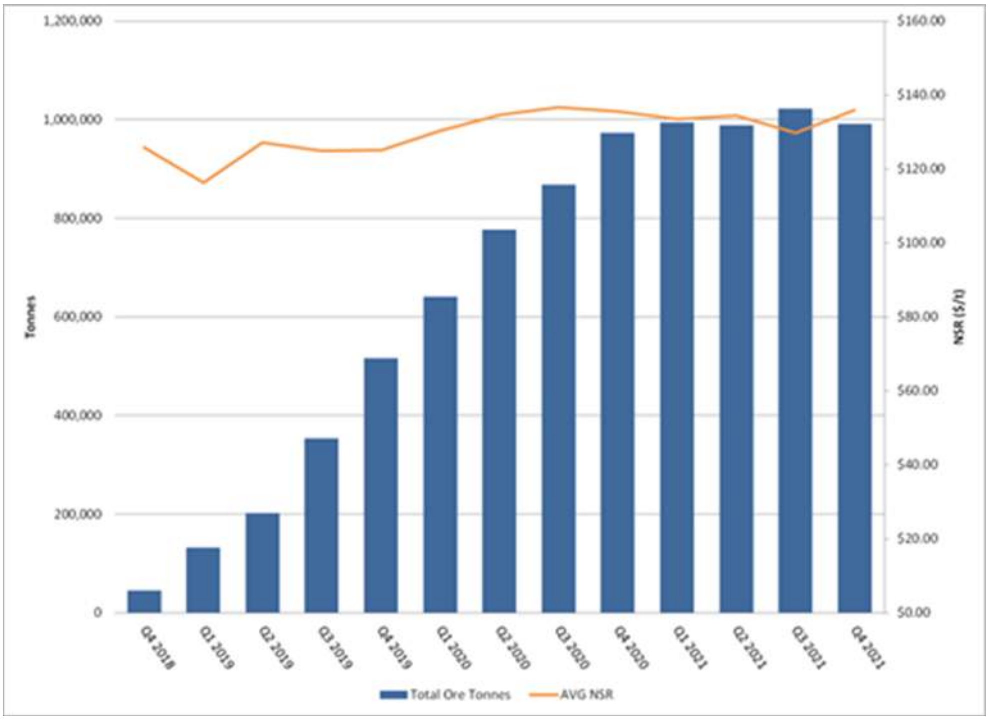
Table 16.35 shows mining areas targeted for early production. Average diluted NSR values and production start month are also shown.

Table 16.40 Initial Production Areas

Mining Area	Avg. NSR ($/t – diluted)	Start Month
6A	$174	Oct 2018
16E	$138	Nov 2018
9A	$146	Dec 2018
17D	$159	Apr 2019
15A	$158	Dec 2019
7C	$136	Feb 2020
12A	$158	Apr 2020

The ramp-up period was defined as the period of time required to achieve the full ore production target of 1 Mt per quarter after production begins. Approximately 2.5 years were required to achieve this objective. Full production was achieved during the Q4'20. Figure 16.29 illustrates the production ramp-up by quarter.

Figure 16.29 Quarterly Production Ramp-Up and Average NSR



16.2.16 Life of Mine Production Schedule

The LOM schedule was generated with a series of Software suites. Vulcan modelling software from Maptek was used for the generation of the initial stope shapes. The stope shapes were then used to design workings used to tie ore access development to previously optimised shaft locations. Additionally, Vulcan was used to assign many known attributes to the design strings and solids that would be carried forward into the scheduling suite (Studio 5DP and EPS) and later cost estimating software (Hard Dollar).

16.2.16.1 Production Summary

A yearly production of 4 Mtpa was achieved with full production starting at the year 2021. A total of 120,203,196 t of ore with an average NSR value of $134.87 was scheduled to be mined during the 31-year mine life.

During the mine life, a total of 11.7 Mt of waste will be produced. Years 2018, 2019, and 2020 will have the highest waste production rate due to the required early developments before the full production period.

In the ore produced from the reserve base (designed stopes), a significant amount of economic grade material (+$50 NSR) will be produced during stope and access development. This material is included as ore in the production schedule.

Table 16.41 summarizes LOM ore production.

Table 16.41 Life of Mine Production Summary

Production Summary	Units	Total
Drift-and-Fill	Mt	9.87
Drift-and-Bench	Mt	3.49
Longhole Stopes	Mt	58.92
Ore Development	Mt	47.92
Total Ore	Mt	120.20
Diluted Grades		
NSR	$/t	$134.87
Cu	%	0.154
Ni	%	0.322
Pt	g/t	1.759
Pd	g/t	1.875
Au	g/t	0.256
Rh	g/t	0.126
3PE+Au	g/t	4.015
Max Daily Production Rate at 360 days per year	t/d	11,132
Waste	Mt	11.72

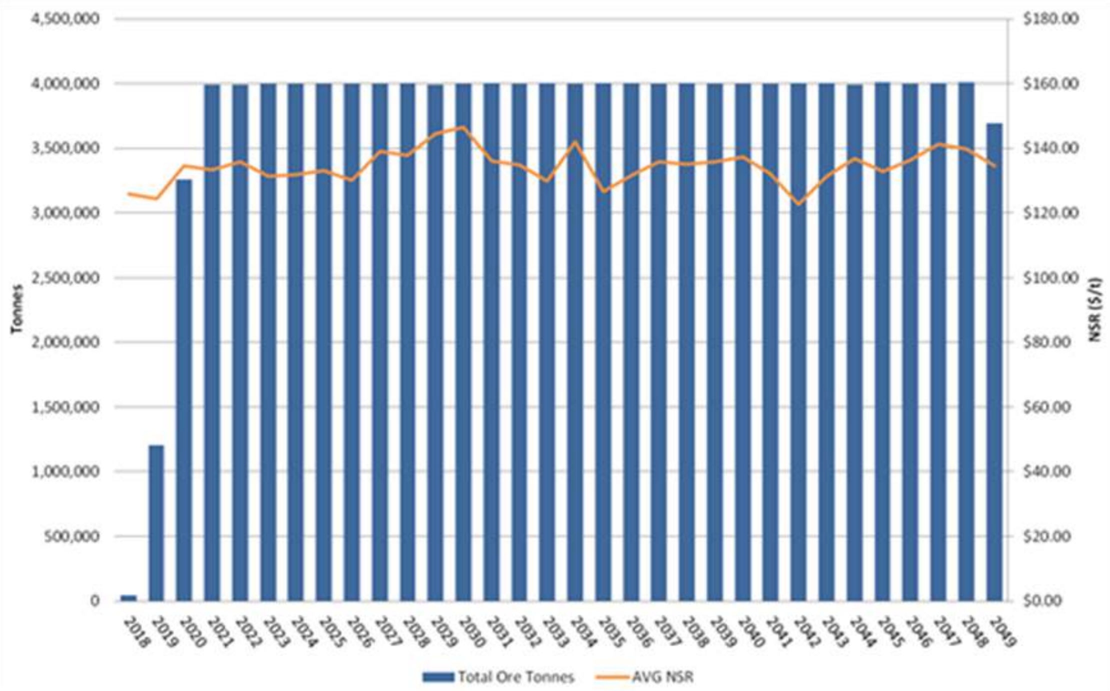
An additional 5.15 Mt of low-grade stockpile material (NSR ranging from $17/t to $50/t) is produced from development headings outside the defined mineral reserve boundary. This material is not included in the reserve estimate nor is it part of the production plan or the project economics. It is presented only as an opportunity since the grade of this material is sufficient to pay for processing costs after it has been hoisted to the surface. The average NSR value of this low-grade stockpile material is $34.52/t. A summary of low-grade stockpile material is provided in Table 16.42.

Table 16.42 Summary of Low-Grade Stockpile

Description	Value
Ore Mt	5.15
Diluted Grades	
NSR ($/t)	$34.52
Cu (%)	0.082
Ni (%)	0.179
Pt (g/t)	0.525
Pd (g/t)	0.637
Au (g/t)	0.093
Rh (g/t)	0.042
3PE+Au (g/t)	1.322

Figure 16.30 shows the yearly production and average NSR value from the EPS scheduler program.

Figure 16.30 Yearly Total Production and Average NSR

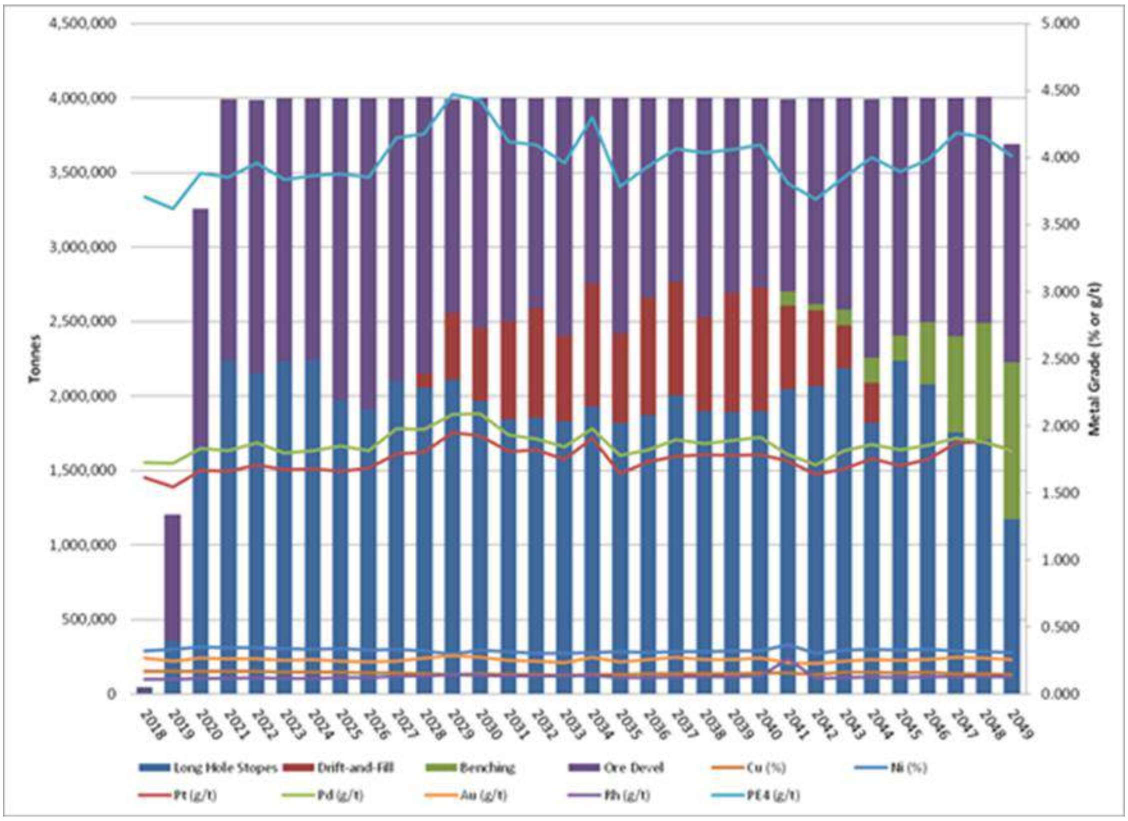


An overall summary showing production by mining method is shown in Table 16.43 . Average metal grades and yearly production by mining method are shown in Figure 16.31.

Table 16.43 Summary of Production by Mining Method

Description	Stope Development	Longhole Stoping	Drift-and-Bench	Drift-and-Fill
Ore Mt	47.9	58.9	3.5	9.9
% Tonnes	40%	49%	3%	8%
Diluted Grades				
NSR ($/t)	$119.41	$144.69	$154.61	$144.30
Cu (%)	0.14	0.16	0.16	0.14
Ni (%)	0.30	0.34	0.33	0.30
Pt (g/t)	1.57	1.85	2.06	1.99
Pd (g/t)	1.67	2.02	2.11	1.95
Au (g/t)	0.23	0.27	0.29	0.29
Ph (g/t)	0.11	0.13	0.15	0.13
3PE+Au (g/t)	3.59	4.27	4.62	4.36

Figure 16.31 Annual Production by Mining Method and Average Metal Grades



The ore produced at Platreef will be comprised as follows:

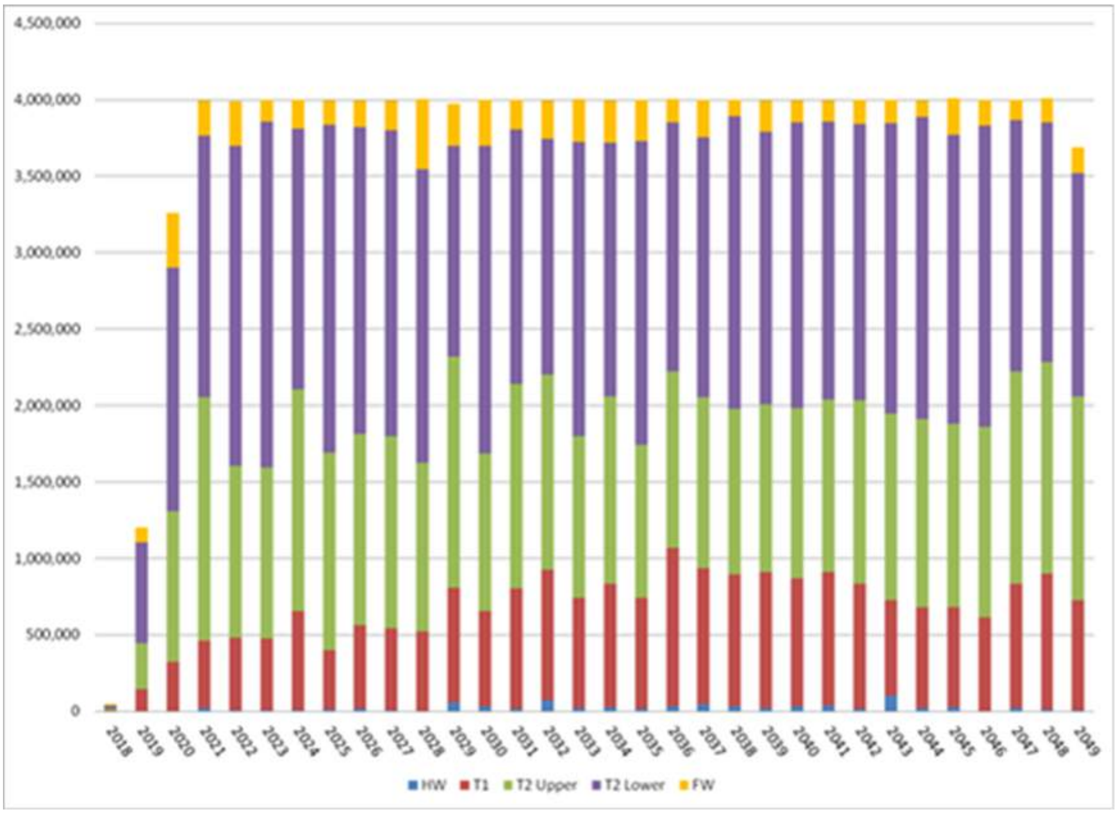
- Hanging wall – planned dilution from material that will occur immediately above the T1 mineralised unit.
- T1 Mineralised Unit – in most cases T1 will occur as planned dilution from the lower portion of the T1 unit and is included in the stopes designed to extract the T2 resources. In a few cases, however, the higher grade portion of T1 will be mined with the T2 ore.
- T2U Mineralised Unit – T2 pyroxenite unit.
- T2L Mineralised Unit – T2 harzburgite unit.
- Footwall – planned dilution from material occurring immediately below the T2 mineralised unit.

Table 16.44 summarizes the overall production from each of these units. Yearly production from each unit is shown in Figure 16.32.

Table 16.44 Overall Production by Ore Component

Description	Hanging wall	T1	T2U	T2L	Footwall
Ore Mt	0.7	20.9	36.8	55.2	6.4
% Tonnes	1%	17%	31%	46%	5%
Diluted Grades					
NSR ($/t)	$71.32	$109.81	$165.75	$132.33	$69.90
Cu (%)	0.08	0.10	0.18	0.16	0.12
Ni (%)	0.18	0.21	0.35	0.36	0.23
Pt (g/t)	1.01	1.69	2.20	1.59	0.95
Pd (g/t)	0.88	1.44	2.24	1.90	1.14
Au (g/t)	0.23	0.26	0.30	0.24	0.16
Ph (g/t)	0.06	0.10	0.15	0.13	0.07
3PE+Au (g/t)	2.18	3.48	4.89	3.86	2.32

Figure 16.32 Annual Production by Ore Component



16.2.17 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 prefeasibility-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.

16.2.18 Risks

The following is a list of potential risks for Platreef:

- Schedule contingencies are excluded from the current mine development and construction schedules.
- There could be long lead times for delivery of fixed and mobile mining equipment.
- There could be a lack of available work force with sufficient skills to meet the specified performance rates at the stated labour rates.
- Expanding the bulk sampling program during Shaft No. 1 sinking will result in an extended preproduction schedule and associated cost.
- There could be ground conditions below 1,100 m that limit shaft depths.
- There is limited underground surge capacity.
- Additional orepasses may be required for transfer of production to truck haulage levels.
- There is a potential of sterilizing lower grade resources by initial mining of higher grade resources.
- The ore and waste passes may not last until the end of the mine's life.

16.2.19 Opportunities and Recommendations

The following is a list of potential opportunities Platreef may experience:

- Optimisation of stope access designs to reduce the amount of waste development.
- Continued optimisation of the stoping sequence to improve the grade profile during the early years of production.
- Automation of production LHDs and trucks.
- Remote operation of fixed rock breakers at ore and waste truck dumps.
- Potential for a second underground crusher and dedicated conveyor to transport ore from deeper resources to Shaft No. 2 for hoisting to surface.
- Potential to incorporate electric-powered mobile equipment in an effort to reduce ventilation and associated refrigeration requirements.
- Further analysis of equipment utilization to reduce fleet size.

- Handling of waste, including the following.
  - Opportunity to use production orepasses as waste passes during the development phases.
  - Optimisation of the mine air cooling system.
  - Reduction of the development width of stope drill drifts and extraction drifts by fan drilling and extending the depth of production holes.

17 RECOVERY METHODS

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

17.1 Introduction

The process design for the flotation concentrator has been developed using the testwork and assessments discussed in Section 13 of this report, and various desktop level trade-off studies.

Phase 1 includes the construction of a 4 Mtpa concentrator and other associated infrastructure in 2020.

A two staged production approach was used for the Phase 1 flow sheet development and design. The plant will be constructed in two increments of 2 Mtpa, with the selected flow sheet comprising a common 3-stage crushing circuit (2-stage crushing during stage 1A), two by 2 Mtpa milling-flotation modules, with a common thickening and filtration circuit. In Phase 1, the plant will have common circuits for water treatment, reagent preparation and compressed air generation.

17.2 Process Design Criteria

The basic process design criteria are summarised in Table 17.1 below.

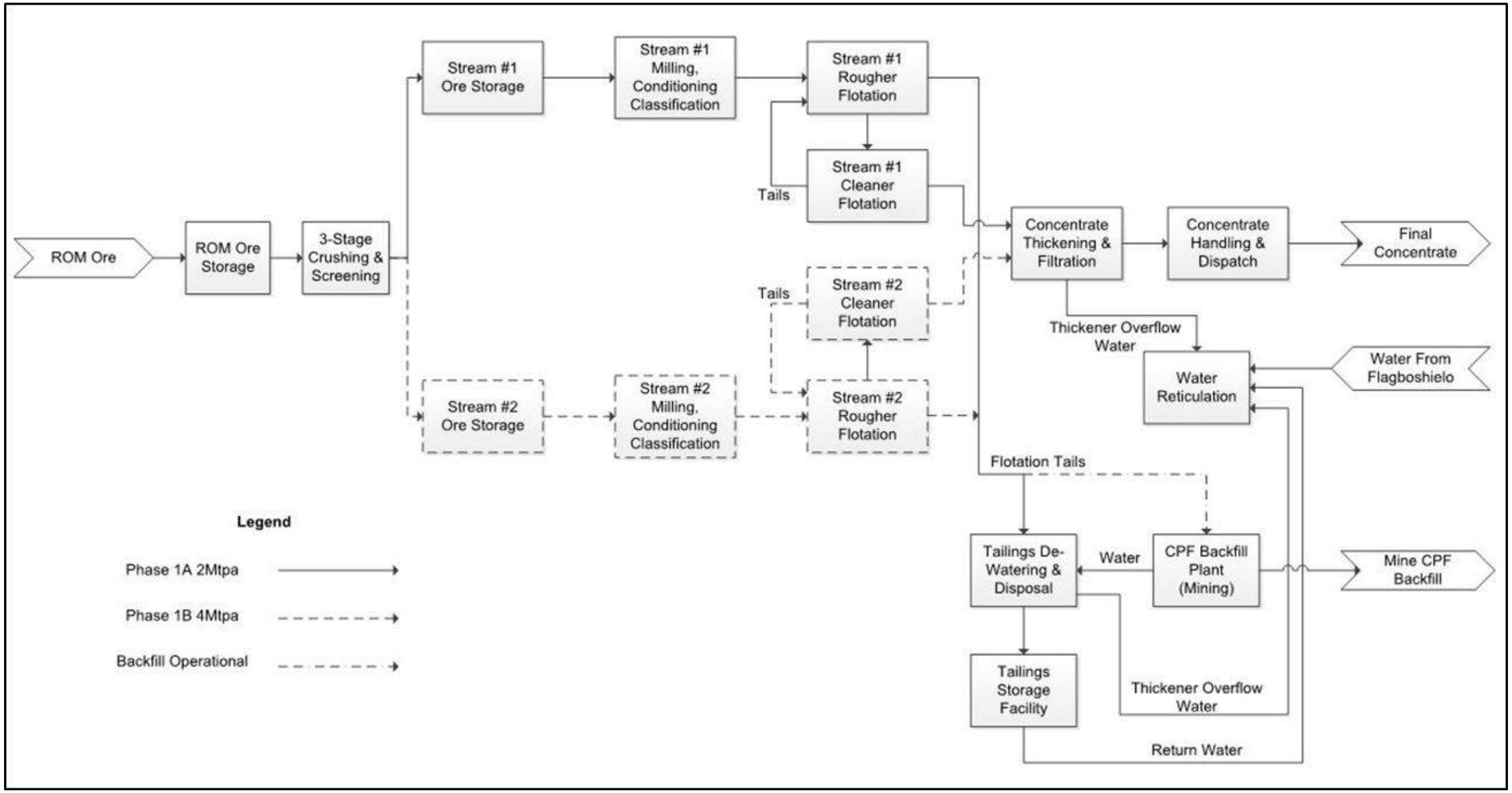
Table 17.1 Basic Process Design Criteria

Criteria	Units	Nominal	Design
Production Summary			
Mining			
Annual Mining Rate year –1	tpa	–	44 876
Annual Mining Rate year 1	Mtpa	–	1.20
Annual Mining Rate year 2	Mtpa	–	3.26
Annual Mining Rate year 3	Mtpa	–	3.99
Annual Mining Rate year 4+	Mtpa	–	3.99
Life of Mine	Years	–	32
Plant Throughput			
Design Throughput Phase 1A	Mtpa	–	2.00
Design Throughput Phase 1B	Mtpa	–	4.00
Mass Pull	%	3.50	4.20
Head Grades			
Platinum	g/t	–	1.52 – 1.95
Palladium	g/t	–	1.71 – 2.09
Rhodium	g/t	–	0.11 – 0.15
Gold	g/t	–	0.23 – 0.29
3PE+Au	g/t	–	3.58 – 4.47
Copper	%	–	0.14 – 0.17
Nickel	%	–	0.30 – 0.36
Overall Recovery			
3PE+Au recovery	%	–	85 - 88
Nickel recovery	%	–	68 - 70
Copper recovery	%	–	88
Concentrate Grades			
3PE+Au grade target	g/t	–	85
Nickel grade target	%	–	8
Copper grade target	%	–	4
Crushing Operating Schedule			
Operating days per annum	days	–	365
Operating shifts per day	#	–	2
Hours per shift	h	–	8
Availability	%	–	70
Utilization	%	–	90
Crushing circuit running time	%	–	63
Overall running time	h/annum	–	5519
Circuit maximum feed rate phase 1A	t/h	362	725
Circuit maximum feed rate phase 1B	t/h	725	725
Milling and Flotation Operating Schedule			
Operating days per annum	days	–	365
Operating shifts per day	#	–	3
Hours per shift	h	–	8
Availability	%	–	94
Utilization	%	–	98
Milling circuit running time	%	–	92
Overall running time	h/annum	–	8061
Circuit feed rate	t/h/module	248	273
Milling & flotation modules phase 1A	#		1 x 2 Mtpa
Milling & flotation modules phase 1B	#		2 x 2 Mtpa

17.3 Process Description

Refer to Figure 17.1 for a high level block flow diagram of the Platreef Project concentrator plant.

Figure 17.1 Block Flow Diagram



17.3.1 Run-of-Mine Storage & Reclamation

Run-of-Mine (ROM) is crushed underground to a top size (F_{100}) of 150 mm. The pre-production ROM material is conveyed to the 500,000 t RoM stockpile. Once the production plant is operating, this material is reclaimed by means of a front end loader into the 70 t ROM feed bin. ROM material is fed onto a 250 mm vibrating grizzly via vibrating feeders. Grizzly oversize is fed to a jaw crusher for ROM top size control. The combined grizzly undersize and jaw crusher product is conveyed to the 6,000 t ROM silo. During normal production the material from the mine will be conveyed directly into the 70 t reclaim bin and be fed to the ROM silo via the vibrating grizzly and jaw crusher circuit.

The ROM silo is equipped with apron feeders for extraction onto the crushing circuit feed conveyor. Tramp metal is removed prior to crushing by means of a tramp metal magnet situated at the conveyor head end.

17.3.2 Crushing & Screening

The crusher circuit consists of an open circuit primary cone crusher, a single cone crusher for secondary crushing duty operating in closed circuit with a set of double deck vibrating classification screens, and two cone crushers for tertiary crushing operating in closed circuit with a set of vibrating screens.

The material from the ROM silo is conveyed to the 125 t primary crusher feed bin from where it is extracted by means of a vibrating feeder and choke fed to a cone crusher for primary crushing. The primary crusher product (P100 93 mm) is conveyed, together with the secondary crusher product, to the coarse ore screening section.

The primary and secondary crusher product is collected in two 110 t coarse ore screening feed bins, from where it is extracted and fed to two double deck vibrating coarse ore screens by means of vibrating feeders. The top decks of the coarse ore screens have apertures of 38 mm, and the oversized material from the top decks are transferred to the 125 t secondary crusher feed bin. The secondary cone crusher operated under choke conditions. A vibrating secondary crusher feeder discharging onto the secondary crusher feed conveyor, feeds the secondary crusher. The secondary crusher product has a P80 of 50 mm (P100 of 89 mm), and is transferred back to the coarse ore screening section. The bottom decks of the coarse ore screens have 12 mm apertures. The oversized material from the bottom decks, together with the oversized material from the fine ore screens, are transferred to two 45 t tertiary crusher feed bins. Each tertiary crusher feed bin is equipped with a tertiary crusher belt feeder to choke feed the two tertiary cone crushers. The tertiary crusher product has a P80 of 11 mm (P100 of 23 mm) and is transferred to the fine ore screening section for final classification prior to milling. The tertiary crusher product is collected in two 45 t fines screening feed bins, from where it is fed to two single deck vibrating screens (10 mm apertures) via the fines screening belt feeders. The oversized material from the fine ore screens is recycled back to the tertiary crusher feed bins, together with the 12 mm deck oversize from the coarse ore screening section. The tertiary screening product (–13 mm) is conveyed to the mill feed silos.

During Phase 1A (2 Mtpa processing rate) the primary crusher installation is not required, and the ROM material is conveyed directly to the coarse screening section, together with the secondary crusher product.

The crushing circuit will have two, separate dust collection systems. One of these will have dust collection hoods at each of the material transfer points from the feeders and crushers. The other dust collection system will service the screen bins and screens.

17.3.3 Mill Feed Storage

The crusher circuit product (–13 mm) is stored in two 6,000 tonne mill feed silos (one dedicated silo per milling stream, 24 hours' storage capacity per silo). The second mill feed silo is only required during Phase 1B. The mill feed storage systems for Phase 1A and Phase 1B are identical.

The crushed material is extracted from each silo via a set of belt feeders, and discharged onto the mill feed conveyor. Grinding media is added to the mill feed conveyor via a steel ball addition hopper and vibrating feeder arrangement. The high chrome grinding media is loaded into the steel ball addition hopper by means of a ball loading magnet and hoist system.

17.3.4 Milling

The milling circuit for Phase 1B (4 Mtpa processing rate) consists of 2 identical milling installations, each capable of treating 2 Mtpa. Only one of the two milling streams will be installed during Phase 1A. A single milling stream is described below.

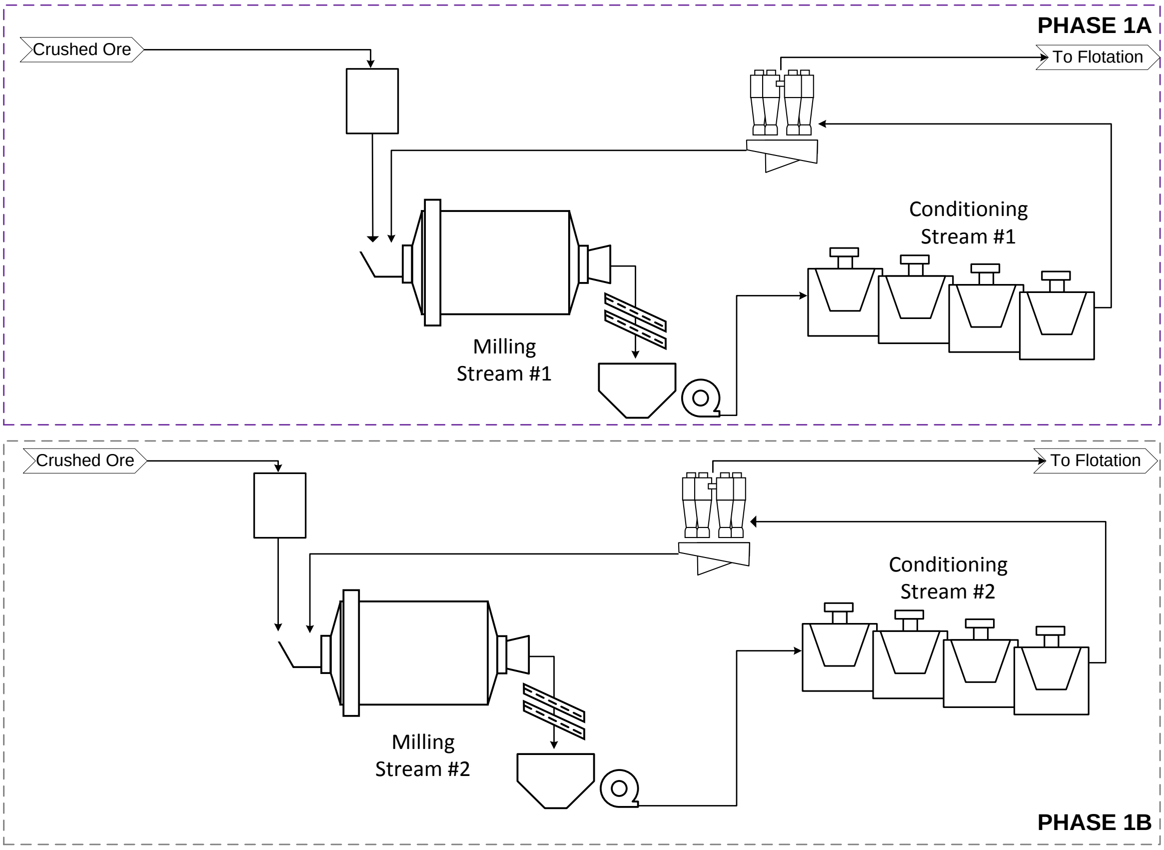
The milling circuit consists of a 10.4 MW (two of 5.2 MW), 22 ft diameter x 33 ft effective grinding length (EGL), grate discharge ball mill, operating in closed circuit with a classification cyclone cluster. The mill feed material (F100 13 mm, F80 9.5 mm) is fed to the mill feed hopper where process water is added for in-mill density control. Thiourea and oxalic acid are also added to the mill feed hopper.

The milled material (P80 250 µm) discharges onto the vibrating, mill discharge screen (5 mm apertures) for scats removal. Scats are collected in a bin from where they are manually removed. The screened material is collected in the mill discharge sump and pumped to the pre-conditioning circuit.

The pre-conditioning product is 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 regrinding. The cyclone overflow gravitates to the flotation bank via a linear trash screen and a two stage sampling system.

The milling and pre-conditioning circuit is schematically presented in Figure 17.2.

Figure 17.2 Milling and Pre-Conditioning Circuit



17.3.5 Pre-Conditioning

The pre-conditioning circuit for Phase 1B (4 Mtpa processing rate) consists of 2 identical installations, each capable of treating 2 Mtpa. Only one of the pre-conditioning circuits will be installed during Phase 1A. A single pre-conditioning stream is described below.

The milled product is pumped to the first of five 100 m^3 tank cells. No aeration is provided on these cells. A conditioning time of 30 minutes is provided. The conditioned slurry is pumped to the milling classification cyclone cluster.

17.3.6 Rougher Flotation

The rougher flotation circuit for Phase 1B (4 Mtpa processing rate) consist of 2 identical modules, each capable of treating 2 Mtpa. Only one of the two modules will be installed during Phase 1A. A single module is described below.

The milling classification cyclone overflow, at approximately 30% solids (w/w), reports to the rougher flotation bank. The rougher flotation bank consists of 9 × 100 m3, forced air, tank cells with total a residence time of 75 minutes.

Three rougher concentrates will be produced. The high grade rougher concentrate product reports to the high grade cleaner flotation circuit. The medium grade rougher concentrate product reports to the medium grade cleaner flotation circuit. The low grade rougher concentrate product reports to the scavenger cleaner flotation circuit.

The rougher flotation tails, together with the scavenger cleaner tailings gravitate to a tails sump via a two stage sampling system. The combined rougher flotation and scavenger cleaner tailings are pumped to the final tailings handling area.

SIPX, frother, and Aero 3477 is added to each of the high, medium, and low grade rougher feed boxes; while Oxalic acid and Thiourea is added to each of the concentrate sumps.

17.3.7 Cleaner Flotation

The cleaner flotation circuit for Phase 1B (4 Mtpa processing rate) consist of 2 identical modules, each capable of treating 2 Mtpa. Only one of the two modules will be installed during Phase 1A. A single module is described below.

The cleaner flotation circuit consists of a two stage high grade cleaner circuit in conjunction with a three stage medium grade cleaner flotation circuit.

The high grade rougher concentrate reports to the high grade cleaner circuit (2 × 20 m3 forced air tank cells, 12 minutes' total residence time). The high grade cleaner concentrate reports to the high grade recleaner flotation bank (2 × 5 m3 forced air box cells, 7 minutes' total residence time). The high grade recleaner concentrate product reports to the concentrate thickening area. The high grade recleaner tails feeds to the high grade cleaner bank. The high grade cleaner tails are pumped to the medium grade cleaner cells.

The medium grade rougher concentrate, together with the scavenger cleaner concentrate and high grade cleaner tailings, report to the medium grade cleaner circuit (6 × 30 m3 forced air tank cells, 25 minutes' total residence time). The medium grade cleaner concentrate reports to the medium grade recleaner flotation bank (2 × 20 m3 forced air tank cells, 15 minutes' total residence time). The medium grade recleaner concentrate reports to the medium grade re-recleaner flotation bank (2 × 20 m3 forced air box cells, 10 minutes' total residence time). The medium grade re-recleaner concentrate product combines with the high grade recleaner concentrate before being pumped to the concentrate thickening area. The medium grade re-recleaner tails feeds to the medium grade recleaner bank. The medium grade recleaner tails feed the medium grade cleaner bank, while the medium grade cleaner tails report to the scavenger cleaner flotation bank, together with the low grade rougher concentrate.

SIPX, frother, depressant and Aero 3477 are added to each of the high and medium grade cleaner, recleaner, and re-recleaner feed boxes; while Oxalic acid and Thiourea will be added to the medium grade cleaner concentrate sump.

17.3.8 Scavenger Cleaner Flotation

The scavenger cleaner flotation circuit for Phase 1B (4 Mtpa processing rate) consist of 2 identical modules, each capable of treating 2 Mtpa. Only one of the two modules will be installed during Phase 1A. A single module is described below.

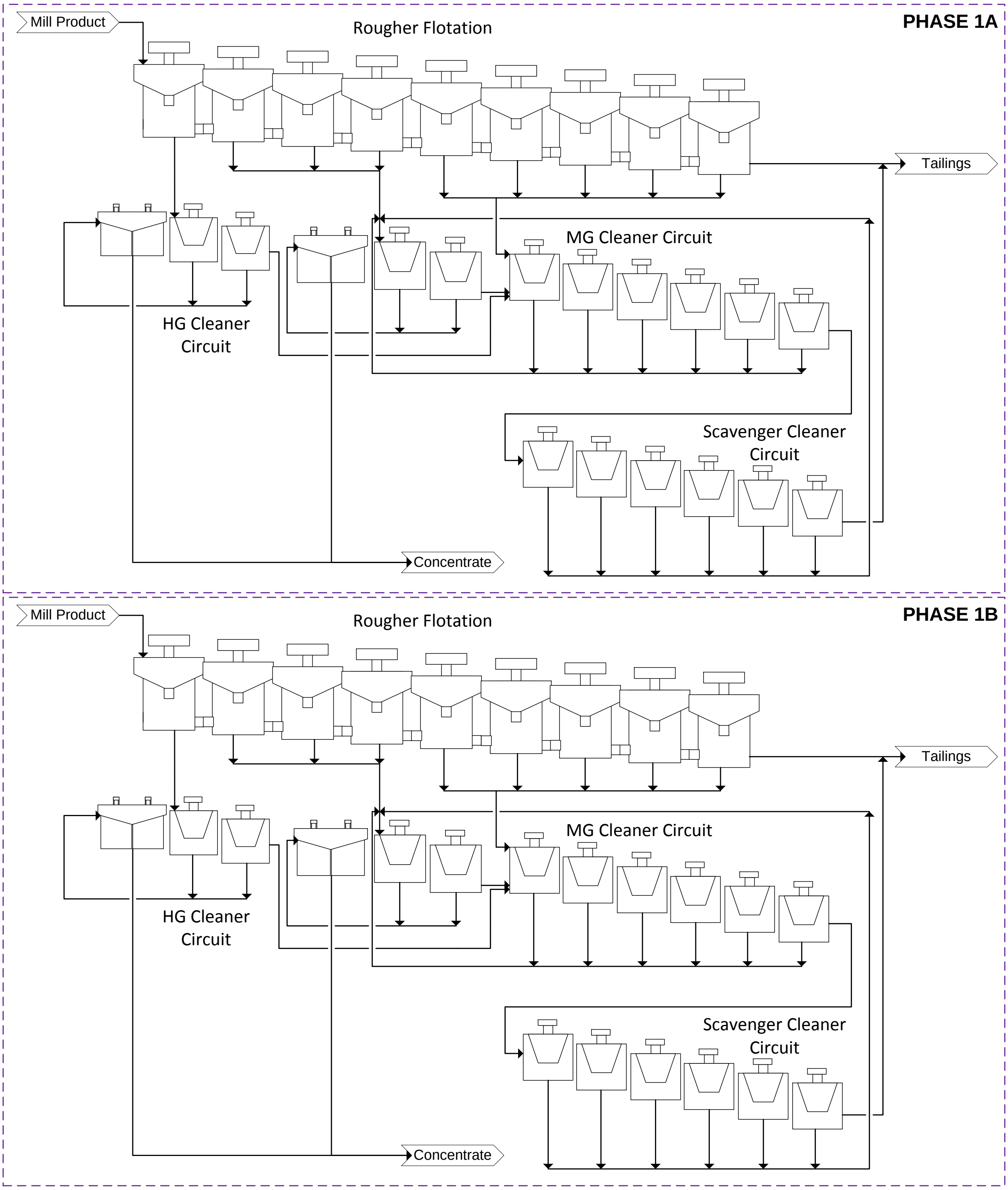
The low grade rougher concentrate and the medium cleaner tails are fed to the scavenger cleaner flotation bank. The scavenger cleaner flotation bank consists of 6 × 50 m3, forced air, tank cells with a total residence time of 37 minutes.

The scavenger cleaner concentrate product reports to the medium grade cleaner circuit. The scavenger cleaner tails gravitates to a tails sump via a two stage sampling system, from where it is pumped to the rougher tailings sump.

SIPX, frother, and Aero 3477 are added to the scavenger cleaner feed box.

The flotation circuit is schematically presented in Figure 17.3.

Figure 17.3 Flotation Circuit



17.3.9 Tailings Dewatering & Disposal

During phase 1A, when the backfill plant is not in operation, the flotation tailings product is pumped to the tailings storage facility (TSF) as dilute slurry (28%-32% solids w/w) and by-passes the tailings thickening and dewatering circuit. The tailings thickener and dewatering cyclone cluster are installed as part of phase 1B.

During both phases when the CPF backfill plant is operational, the dilute flotation tailings is pumped to the backfill plant which is operated by the mining division. The backfill plant produces paste backfill material at approximately 77% solids and returns all water recovered to the process plant tailings thickener or alternatively the process water dam. The backfill plant is expected to consume 60% of the process plant tailings and is based on treating the full flotation tailings stream from the plant for approximately 16 hours a day.

The process plant and TSF PFS design has been based on treating 4Mtpa of tailings material. When operating at a rate of 4Mtpa and the cemented paste backfill (CPF) plant is not in operation, the flotation tailings product, at approximately 28%-32% solids (w/w) is pumped to the tails guard cyclone cluster. The thickened cyclone underflow product bypasses the tailings thickener, while the overflow product gravitates to the 35 m Ø tailings thickener. Flocculant is added to the tailings thickener feed at a rate of 35 g/t to produce a thickener underflow product of 58%-60% solids (w/w) which is pumped to the tailings disposal tank, where it combines with the cyclone underflow product. The overflow product from the tailings thickener is utilised as process water in the circuit.

The final tailings product is pumped to the TSF at a solids content of 55%-58% (w/w) by means of a duty/standby pump train installation. The tailings line includes allowance for a booster station with duty/ standby pump train installation at a distance of 6km from the process plant. A spillage containment bund is located at two points along the tailings line in order to contain spillage that is generated when the tailings line is flushed and drained as part of a controlled stoppage. Each of these spillage bunds is located at a suitable low point along the tailings corridor.

17.3.10 Concentrate Thickening and Filtration

The final concentrate from the cleaner flotation circuit (at a density 1.21 t/m3) is pumped to a linear trash screen via a two stage sampling system. The screened concentrate gravitates to the 25 m Ø high rate, concentrate thickener. Flocculant is added to the thickener feed at a rate of 35 g/t. The thickened concentrate at 55% solids (w/w) is pumped to 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 (60 m2 expandable to 96 m2). The filter cake, with a moisture content of 12 - 15%, discharges onto a reversible conveyor which feeds two concrete storage bunkers (24 hours' storage capacity). The filter filtrate reports to the concentrate thickener. The concentrate product is loaded into trucks and sampled by an Auger sampler before dispatch.

17.3.11 Sampling and Ancillaries

17.3.11.1 Process Plant Sampling and Laboratory

The design includes an allowance for a sample preparation laboratory to prepare the daily samples that are to be sent for analysis at a centralized laboratory catering for mining, process and environmental samples. In addition to this, particle size distribution analysis of the flotation feed will be conducted daily.

The following sample points have been identified and appropriate sampling facilities have been allowed for as follows:

- Mill feed: Manual belt cut sample.
- Rougher flotation feed: Automatic sampling system comprising of a cross cut and rotary vezin arrangement.
- Rougher flotation tails: Automatic sampling system comprising of a cross cut and rotary vezin arrangement.
- Scavenger cleaner tails: Automatic sampling system comprising of a cross cut and rotary vezin arrangement.
- Concentrate thickener feed: Automatic sampling system comprising of a cross cut and rotary vezin arrangement.
- Combined tailings: Automatic sampling system comprising of a cross cut and rotary vezin arrangement.
- Final Concentrate product dispatch: Automatic sampling system comprising of an Auger sampler and tower to sample each concentrate batch that is dispatched.

The rougher flotation feed, combined tailings and final concentrate assays will be used to compile the plant metallurgical balance.

The labour plan used to estimate the process plant operating costs includes operational staff on each shift to cater for sample collection, preparation and particle size distribution analysis. The operating process plant cost has been based on the assumption that an external company would be used for assaying.

17.3.11.2 Process Plant Control

The design includes an allowance for a fully integrated control system to allow for control of the plant from a centralized control room. The plant will be fitted with an appropriate level of automation to allow for remote control of major processing equipment by a PLC and SCADA system. An integrated SCADA/HMI control system will be used for interfacing with the operational staff. The labour plan used to estimate the process plant operating costs includes operational staff on each shift to operate the control room as well as dedicated control and instrumentation technicians.

17.3.11.3 Process Plant Weighbridge

The design includes allowance for a weighbridge located at the proces splant main access gate. The weighbridge will be used to control delivery and dispatch of the following:

- Concentrate dispatch
- Reagent deliveries
- Grinding media deliveries

17.3.12 Reagents

17.3.12.1 Reagents – Collector A

Collector (SIPX) is delivered in liquid form (minimum 35% w/v strength) via bulk road tankers and offloaded into two storage tanks. The SIPX is pumped to a make-up tank where it is diluted to 10% w/v for dosing. Dosing to the required points is done via peristaltic pumps at a nominal rate of 92.5 g/t per module. Each flotation module is serviced by a dedicated make-up and dosing system.

17.3.12.2 Reagents – Collector B

Collector (Aero 3477) is delivered as a liquid in 1 t intermediate bulk containers, from where it is dosed directly without any dilution. Dosing to the required points are done via diaphragm metering pumps at a nominal rate of 92.5 g/t per module. Each flotation module is serviced by a dedicated dosing system.

17.3.12.3 Reagents – Depressant

Depressant (Sendep 30E) is delivered in solid form via bulk road tankers and offloaded pneumatically into a silo. The depressant is made up to 1.0% w/v strength prior to dosing. Dosing to the required points is done via variable speed helical rotor pumps at a nominal rate of 97.5 g/t per module. Each flotation module is serviced by a dedicated make-up/wetting and dosing system.

17.3.12.4 Reagents – Frother

Frother (Senfroth 522) is delivered as a liquid in 1 t intermediate bulk containers, from where it is dosed directly without any dilution. Dosing to the required points is done via diaphragm metering pumps at a nominal rate of 82.5 g/t per module. Each flotation module is serviced by a dedicated dosing system.

17.3.12.5 Reagents – Thiourea

Thiourea granules are delivered in 25 kg bags. The granules are dissolved and diluted to 10% w/v in a make-up tank before being transferred to the dosing tanks. Each milling-flotation module is serviced by a dedicated dosing system. Dosing to the required points are done via peristaltic pumps at a nominal rate of 76 g/t per module.

17.3.12.6 Reagents – Oxalic acid

Oxalic acid granules are delivered in 25 kg bags. The granules are dissolved and diluted to 8% w/v in a make-up tank before being transferred to the dosing tanks. Each milling-flotation module is serviced by a dedicated dosing system. Dosing to the required points is done via peristaltic pumps at a nominal rate of 310 g/t per module.

17.3.12.7 Reagents – Flocculant

Flocculant granules are delivered in 25 kg bags and the bags are manually loaded into a bulk bag bin receiver. The flocculant granules are transferred to a wetting system via a screw feeder. The wetted flocculant is diluted to 0.5% w/v strength for dosing helical rotor pumps. The nominal flocculant consumption is 70 g/t.

17.3.13 Water Services

The process plant raw water supply is filtered in a sand filter plant before use for process and fire water circuit top-up, potable water, reagent make-up water, gland service water, and dust suppression water. Potable water is produced in a potable water treatment plant.

Process water is stored in a 13 000 m^3 process water dam that is interlinked with a 300 m^3 process water tank, allowing for 12 hours' storage capacity. The process water dam is fed by the TSF return water, Backfill return water, the tailings thickener overflow product, as well as excess concentrate thickener overflow product. Filtered raw water is used as top-up to the process water circuit. Each milling-flotation module is equipped with a dedicated process water pumping installation.

17.3.14 Air Services

Low pressure blower air to the flotation circuit is supplied by positive displacement blowers. Each flotation module is serviced by a dedicated blower circuit, consisting of 5 blowers per circuit.

Plant and instrument air are supplied by rotary screw compressors, delivering compressed air at 850 kPa (g). The majority of the compressed air passes through an air filtration and drying system, before being used for instrument air. The remainder of the air is used as plant air.

The drying air to the Larox filter is drawn from the plant air circuit. Filter pressing air is supplied by two high pressure rotary screw compressors delivering compressed air at 1,600 kPa (g).

17.4 Comments on Section 17

The qualified person responsible for this section, Mr Val Coetzee, is of the opinion that the proposed plant design has captured the findings of the testwork programme.

Minor concerns have been raised as to the selected residence times and the number of flotation cells that have been used per flotation stage. This will however not have a material impact on the overall capital cost and the design will have to be critiqued in the next stage of the Platreef Project. It is most likely that the size of the flotation circuit may reduce in size once further variability testwork is conducted.

The proposed design is by all means not the final design and the design will have to be adapted based on the findings of the next stage of metallurgical testwork. The impact of the footwall and added dilution may have a material impact on the final circuit configuration.

18 PROJECT INFRASTRUCTURE

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

18.1 Introduction

The Platreef Project site is located approximately 280 km north-east of Johannesburg in the Limpopo Province – falling under the Mogalakwena Local Municipality. The Mine Lease Area is on the Turfspruit 241 KR, Macalacaskop 243 KR and Rietfontein 2 KS 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 closest international airport is the OR Tambo International Airport, about a three-hour drive from Mokopane, and the regional hub is at Polokwane (formerly Pietersburg) 60 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.

Figure 18.1 Project Location within South Africa

Figure supplied by Ivanhoe, 2014

The 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 Site was taken into consideration during development of the infrastructure estimates.

Bulk Power will be supplied by Eskom to the mine from the Borutho substation approximately 26 km to the east of the Mine via 132 kV overhead lines and a dedicated Eskom power servitude. Power is reticulated to the Eskom incoming substation yard inside the Platreef Project site; 33 kV construction power is transmitted to site from the Mokopane township.

Bulk water supply to the mine via the Olifants River Water Resource Development Project, undertaken by the Department of Water and Sanitation, is currently still under investigation. As part of this strategy, the ORWRDP has been identified to supply water to the Middle Olifants as well as the Polokwane and Mogalakwena areas in the Limpopo Province. Alternative bulk water supply sources are being looked at by the Platreef Project.

Figure 18.2 Locality Map Showing Major Townships and Roads

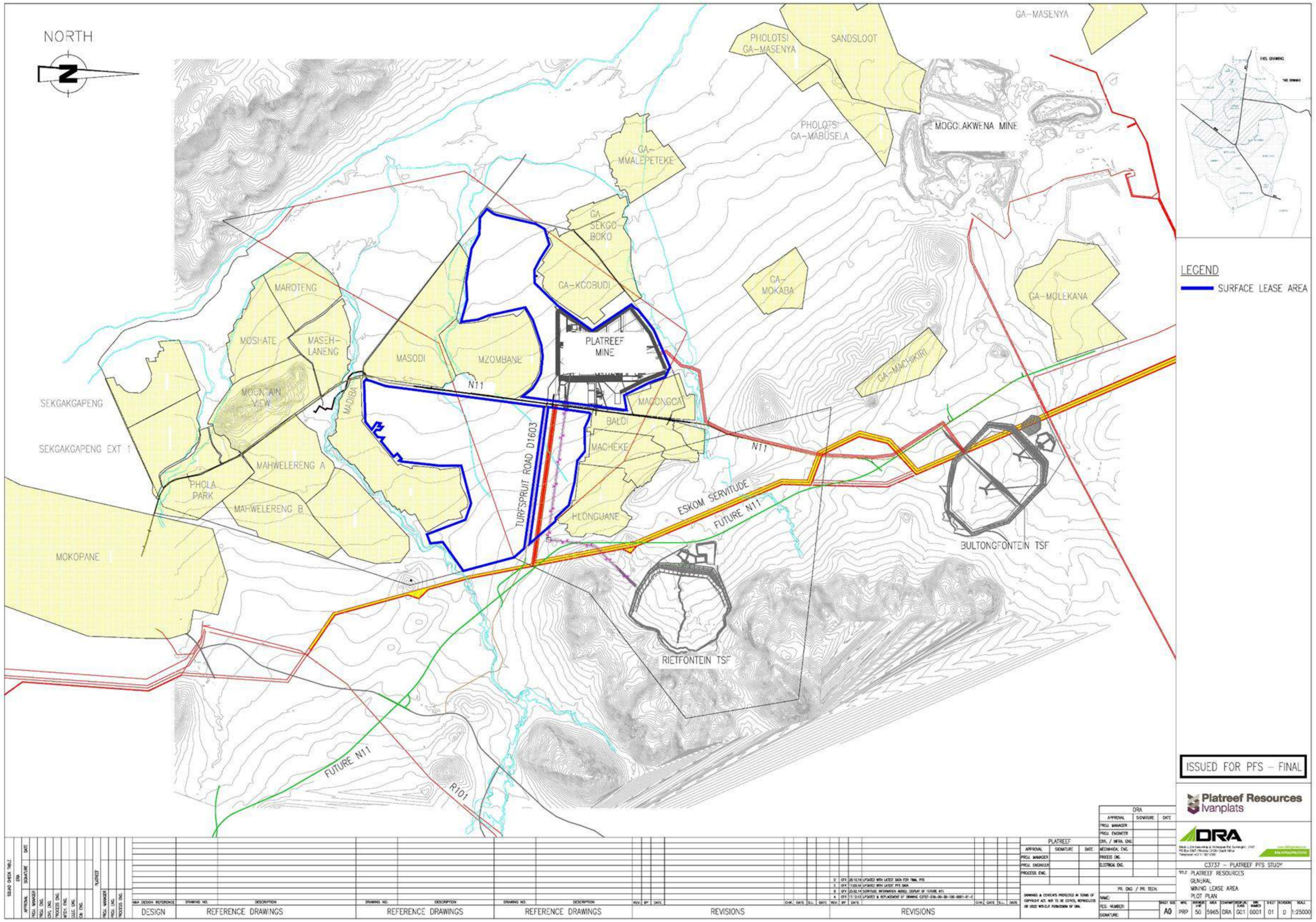
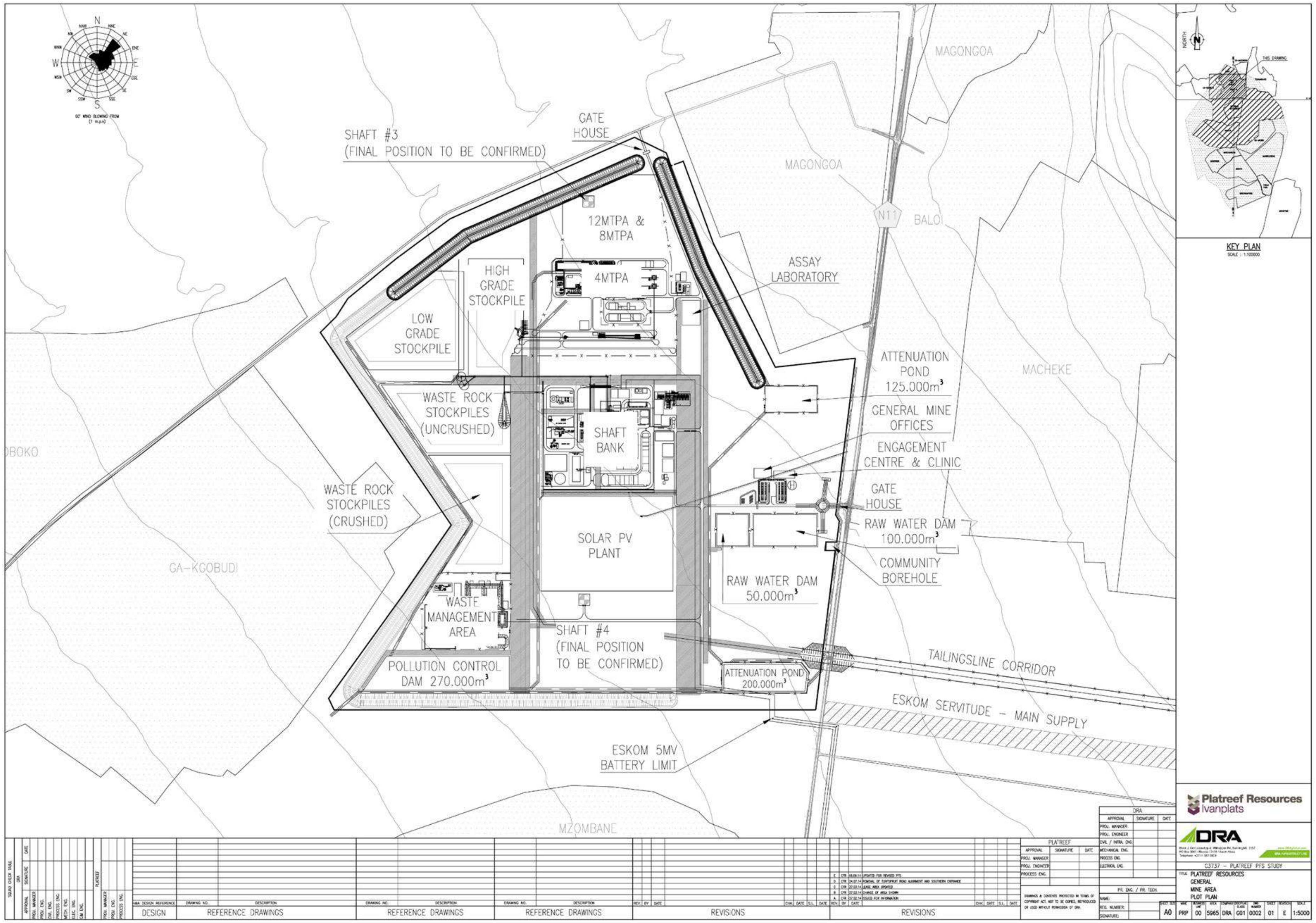


Figure 18.3 Platreef Mining Area and Rietfontein TSF Plot Plan

Figure 18.4 Platreef Mining Area Block Plan



18.2 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 Site. The main line of the national railroad system passes approximately 6 km east of the Site.

A business survey conducted, showed that a larger number of businesses are located near the Mine area. Most 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 5 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.

18.3 Local Labour

Mining activity is moderate within a 100 km radius. A large, potential labour force lives 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.

Findings from an accommodation survey conducted are that adequate town-site facilities and infrastructure exist to support an influx of personnel. The survey showed that Mokopane has the capacity to provide housing to just over 1,600 people, which will be sufficient for the mine’s maximum requirements

The majority of labour will be employed from the surrounding villages and communities in Mogalakwena Local Municipality, minimizing the need for the development of new housing facilities. Instead of providing additional housing facilities, Ivanplats will support junior level staff with a Housing Rental Subsidy / House Ownership Allowance relative to their employment level to improve current dwellings or subsidise rental. Senior level staff will receive a housing allowance factored into their Total Cost to Company. The financial housing support measures shall allow employees to provide their housing needs independently.

Ivanplats is planning a residential housing development scheme consisting of various types of dwellings in different parts of Mokopane. These units will serve as an interim housing solution to accommodate new employees relocating from other areas or special housing needs. The housing scheme will be developed and managed separately from the mining activities and will be available in future to private tenants at normal market rental prices when the demand from the mine reduces.

18.4 Water – General Infrastructure

18.4.1 Water Balance and Bulk Water Availability

Water is a valuable resource in South Africa and a prudent approach has to be taken in the management thereof. In order to ensure effective use of water over the whole project, a water balance was developed. The objective of the water balance was to calculate the volumes of make-up water that will be required for the mine and process plant facilities under equilibrium conditions, size the pollution control dam to meet the spill criteria and to ensure effective water use by the proposed project in all sections of the process. Simultaneously peak demands were determined using the model.

The main sources of water are anticipated as follows:

- Bulk water supply – options currently being investigated;
- Dewatering of underground mine workings;
- Surface run-off to contained areas;
- Rainfall directly onto water dams and tailings storage facilities; and
- Potable water supplies used in the processes as well as sewage effluent treated on site.

The main water losses from the mine water circuits include:

- Evaporation from dams and tailings storage facilities;
- Entrainment and retention in tailings dam;
- Seepage; and
- Spill to environment during extreme events (> 50 year recurrence interval).

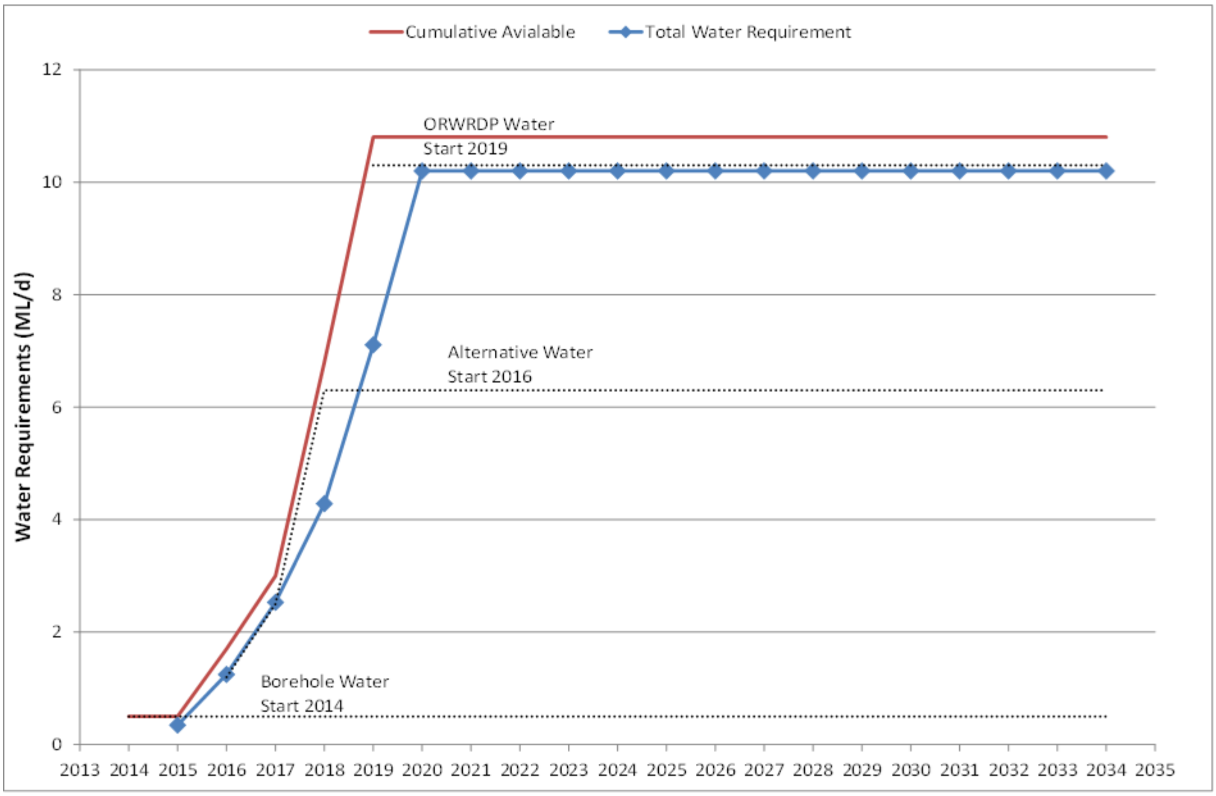
During the prefeasibility study, the bulk water supply volumes were determined. The water balance model simulations showed that the average bulk water supply over the life of the mine is 5.5 ML/d. The bulk water supply volume however varies over the life of the mine as the production ramps up and the groundwater inflows to the underground workings vary over the life of the mine. The water balance modelling highlighted the variation in the bulk water supply volume due to seasonality, wet and drought periods. The modelling showed that the maximum daily bulk water supply volume that is required during dry years, assuming limited groundwater ingress to the workings, is 10.3 ML/d.

The graph below plots the total water demand against the bulk water supply sources over the life of mine. Initially bulk water is required for the shaft sinking and mine construction. The bulk water supply requirements ramp up from the initial 0.3 ML/d to the maximum of 10.3 ML/d when the first ore processing starts in Q2'19.

During the initial sinking and construction phases of the Platreef Project, water will be supplied from a number of onsite boreholes extracting groundwater and the Phase 2B pipeline developed as part of the Olifants River Water Resource Development Project (ORWRDP). The groundwater volumes from the various boreholes have been confirmed and their use licenced by the Department of Water Affairs (DWA), but the construction of the Phase 2B pipeline has not yet been confirmed and will possibly be delayed from to 2022. As shown in the water demand-supply graph, the delay of the Phase 2B pipeline will result in a shortfall in the bulk water supply of 6 ML/d between 2016 and 2019.

Ivanplats has identified 3 possible alternative water sources to mitigate the potential shortfall and a possible delay to the Platreef Project. These are treated sewage effluent, and further development of local groundwater extraction. The sources of bulk water are discussed in more detail in the sections to follow.

Figure 18.5 Bulk Water Demand-Supply Graph

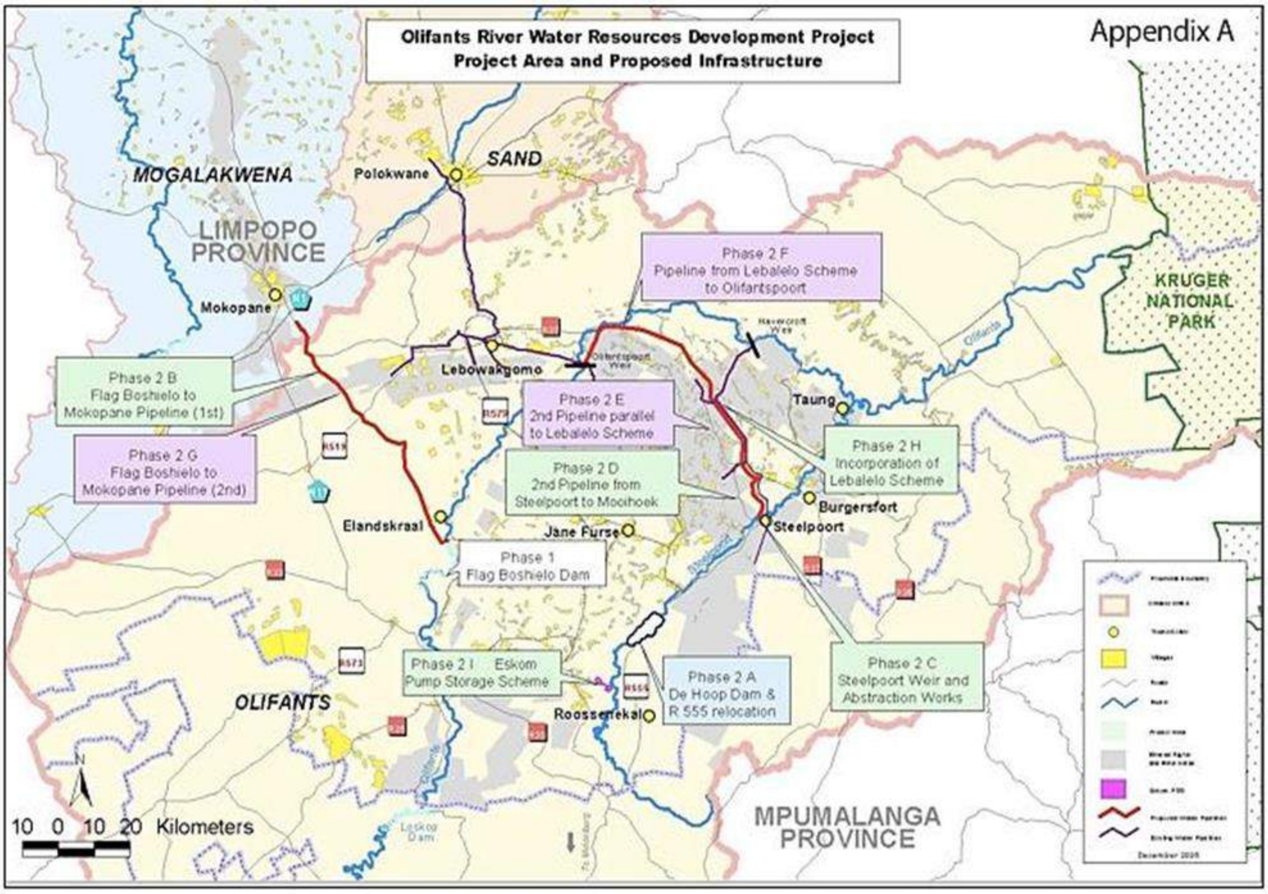


18.4.2 Bulk Water Source

South Africa is a country of relatively low rainfall and, in particular, the Limpopo Province (typical rainfall ~600 mm per annum) will require additional water sources to meet the growing water requirements from the mining, domestic and agricultural 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 which need to be addressed efficiently in order to ensure that sufficient water supply is achieved. To this end the Department of Water and Sanitation (DWS) has developed a reconciliation strategy for the Olifants River Water Management Area. As part of this strategy, the ORWRDP was identified to supply water to the Middle Olifants as well as the Polokwane and Mogalakwena areas in the Limpopo Province.

The ORWRDP is designed to deliver water to the Eastern and Northern limbs of the Bushveld igneous complex of South Africa. The project consists of the new De Hoop Dam, the raising of the wall of the Flag Boshielo Dam and related pipeline infrastructure which will ultimately deliver water to Pruissen, located to the south-east of Mokopane and the Platreef mine. From this point, the Pruissen pipeline project will be developed to deliver water to the communities and mining projects on the Northern Limb. An outline of the ORWRDP is shown in the figure below.

Figure 18.6 ORWRDP Outline



Ivanplats is a member of the Joint Water Forum (JWF) (part of the ORWRDP) and the Mogalakwena Development Forum. These forums have been established to facilitate and coordinate discussions with the various participants in the scheme within the Eastern and Northern limbs of the Bushveld igneous complex. Other major participants in these forums are Anglo Platinum and Lonmin.

Participants in the water scheme are required to indicate their water requirements from the scheme in order for total water requirements to be calculated relative to the capacity of the scheme. These requirements are translated into a non-binding Memorandum of Agreement and then a binding Off-take Agreement. Ivanplats has indicated that their water requirement for the Platreef mine is 16 ML/d. Ivanplats is committed to work with the JWF to develop the ORWRDP as the primary source of bulk water to service the needs of the Platreef Project.

18.4.3 Potential Alternative and Interim Sources of Water

Ivanplats is pursuing alternative bulk water sources to fill the shortfall for the interim period until the ORWRDP is operational. To date, treated sewage effluent and local groundwater has been identified as sources of water to meet the 6 ML/d shortfall for the period. All the options are pursued until agreement is reached for one or more of the sources.

The progress with the three options to secure an interim bulk water supply is:-

- Two sources of treated sewage effluent have been identified. 6 ML/d is available from the Mogalakwena Local Municipality (MLM) and up to 12 ML/d is potentially available from the Polokwane Local Municipality (PLM). Ivanplats has been negotiating with MLM for the available 6 ML/d and a basis to set the price for the effluent is planned to be agreed on by end of 2014. Initial meetings have also been held with PLM and further meetings have been setup to take the negotiations to secure the effluent forward. A high level costing of the pipeline from Polokwane to the mine has been undertaken. The treated sewage effluent, depending on the volume, timeframe and cost could reduce or replace the volume required from the Phase 2B pipeline.
- The PFS groundwater study identified that there is potential to abstract more water than the 0.5 ML/d from the local groundwater resources on the Ivanplats properties. These are being further investigated.

Investigations into other water sources is ongoing as a mitigation method to the possible bulk water delay risk.

18.4.4 Potable Water

It is expected that potable water for ablution facilities and amenities will be obtained from the bulk water supply system or from treated groundwater resources currently being investigated. A potable water treatment plant has been included to treat raw water to potable quality.

Potable water is distributed from the potable water treatment plant to the Shaft Area, Process plant area and the waste area via surface run racks and sleepers. Local to each area, header tanks are used to gravity feed buried potable water systems.

18.4.5 Wastewater

Sewage is collected from change houses, offices, stores, workshops and external ablution facilities throughout the plant and mining areas. The mine area is equipped with a system to empty underground sewage cassettes into the main sewerage system. In each area sewage is gravity fed via buried piping to dedicated sewage pits. From these pits, sewage is pumped to a sewage treatment plant. Grey water from this plant is reintroduced into the process water circuit and sludge is pumped to a dedicated composting facility.

Wash down water and contaminated waste water from workshops etc., is fed through oil/water separators, with the skimmed water introduced to the pollution control system and the oils drummed and stored in the hazardous waste area.

18.4.6 Storm and Contaminated Water

Storm water is defined as the clean water that enters the mine area during a rainfall event, either by direct rain on non-polluted areas in the mining area or as collected stormwater from outside the mining area. Storm water is the water that has to be managed around the property and cannot be used by the mine but has to be discharged downstream from the mining area.

Polluted water is all water that is contaminated from mining operations and has to remain within the closed loop water balance internal to the mining area. Typical sources are rainwater falling on contaminated or dirty areas, spillage water and dust suppression water.

The Stormwater management measures will include the following features:

- Cut-off drains to intercept storm runoff upstream of the mining area.
- Cut-off drains to intercept storm runoff on the eastern boundary off the N11.
- Attenuation ponds for attenuating potentially large water inflow volumes to protect downstream infrastructure and villages when water is discharged downhill from the mining area.

The Pollution water management measures include the following features:

- Run-off drains local to the process pant and shaft area to collect all polluted water.
- Site wide run-off drains to collect polluted water from other areas in the mining area and deposit it to the pollution control dam.
- Dedicated contaminated water drainage systems around the stockpile areas.
- A single large pollution control dam to capture this water and return to the mining and process water circuits.
- Silt traps to collect water from the plant run-off drains and remove grit before discharge into the pollution control dam.

In accordance with the overall water balance, water will be pumped out from the pollution control dam back into the water circuit.

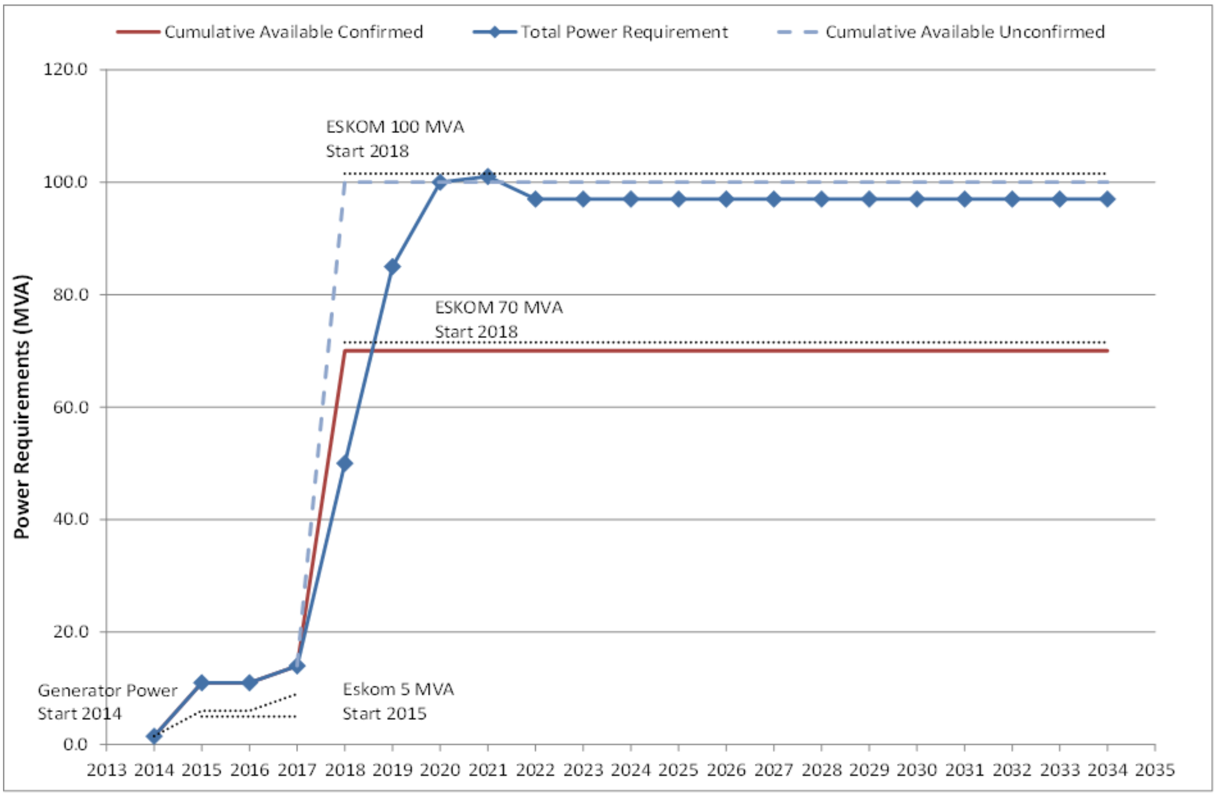
All contaminated and storm water systems have been estimated in accordance with the requirements of the EMP and Integrated Water Use Licence.

18.5 Electrical – General Infrastructure

18.5.1 Predicted Electrical Consumption and Notified Maximum Demand

The graph below plots the electrical power consumption ramp up against the power supply sources.

Figure 18.7 Electrical Power Consumption Graph



The power demand during the initial years of mine development is required to support shaft sinking and construction activities. During this construction period the demand will peak at 14 MVA in 2017. From 2018 to 2020 the demand will increase sharply with the commencement of underground mining activities and the process plant start-up in 2019. The steady state operation power consumption is predicted at an average Notified Maximum Demand (NMD) of 97.3 MVA. A bottom up estimating methodology was used to arrive at a predicted NMD and electrical consumption for the proposed installations at the Platreef Project. The following numbers were developed from the loads as designed for in the prefeasibility study:

Mining Area as calculated by Stantec:

Connected MW	61.0
Diversified MW	30.0
Running MVA	59.0

Process Plant and Surface Infrastructure Areas as calculated by DRA:

Connected MW	45.1
Diversified MW	31.5
Running MVA	38.3

Thus a resultant 97.3 MVA required as a running load.

18.5.2 Bulk Power Supply

Eskom has advised that sufficient power is not available at present in the Mokopane area due to transmission line limitations and generating shortfalls. The generating shortfall should be alleviated when the first unit of the new Medupi power station is brought on line.

A new Main Transmission Substation (MTS), called the Borutho MTS (400 kV/132 kV/22 kV) is sized at 500 MVA (extendable to 1000 MVA) and will be commissioned during the H1'15. The Borutho substation is approximately 26 km from the site and will provide the main feed to the new Platreef mine.

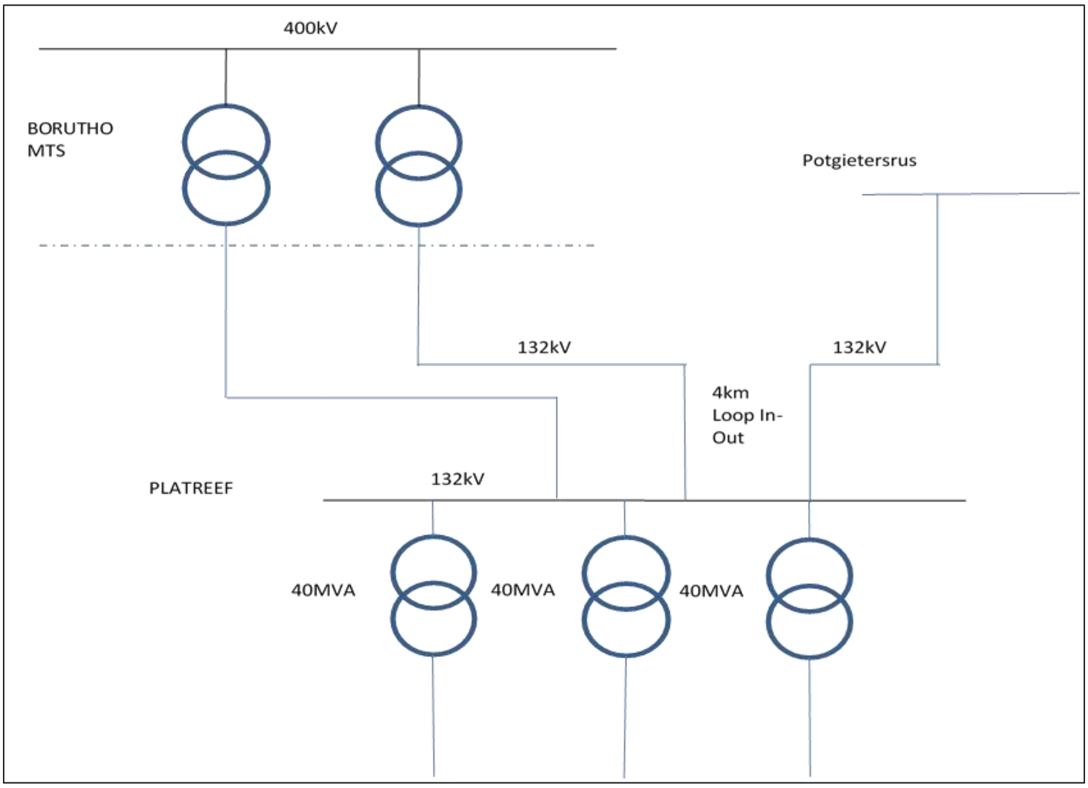
There are two transmission alternatives to supply power to the Platreef mine; a standard and a premium supply scheme. Ivanplats has selected the premium supply scheme. The standard supply scheme essentially consists of a radial line to the Platreef mine whilst the premium supply scheme would include a loop in and loop out from the Platreef site totalling 8 km. This will allow redundancy in power transmission lines should one of the lines go down.

During 2011 Ivanplats submitted an application to Eskom for the supply of bulk power to the Platreef mine. 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 then requested that Eskom complete the budget quote study for 70 MVA that considers the premium supply option. Furthermore, Ivanplats has informed Eskom regarding a potential additional power demand for future expansion phases. Ivanplats paid a commitment fee of US$1.9 million to initiate the budget quotation phase for the following (i) EIA; (ii) engineering and design; and (iii) procurement of long lead items if required.

Eskom has completed the relevant land and rights as well as EIA processes. The Eskom self-build option Budget Quote (BQ) has been accepted and paid by Ivanplats and the detail design package has been completed by Eskom. The latest forecast energisation date of the Platreef Eskom incoming substation is H2'17.

Figure 18.8 70 MVA Premium Supply



With the increased production volume from 3 Mtpa to 4 Mtpa for the initial phase of the Platreef mine and based on the pre-feasibility study design work prepared by Stantec and DRA, the Platreef Project power requirement has been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats is currently preparing the notification and request to ESKOM for the 30 MVA additional power demand required by the Platreef Project. The pre-production capital estimate has been updated to include for a fourth 40 MVA transformer at the incoming substation.

18.5.3 5 MVA Construction Power Supply

Ivanplats submitted an application to Eskom for a temporary 5 MVA 33 kV construction supply to the Platreef mine. The supply will be taken from the Mahwelereng 33/11 kV substation which is 7 km from the Platreef mine. A feeder bay at Mahwelereng is being constructed to feed the 7 km 33 kV OHL to the Platreef Mine. The Platreef end of the OHL will be equipped with an isolator and metering equipment.

The Environmental Impact Assessment (EIA) process is complete and the design fees for the 70 MWA has been paid. The latest forecast energisation date of the Platreef Eskom incoming substation is H2'18.

Based on the prefeasibility study design work prepared by Stantec and DRA, the Platreef Project power requirement for a 4 Mtpa underground mine have been updated to predict an average Notified Maximum Demand (NMD) of approximately 100 MVA. Ivanplats has notified of and requested from Eskom the 30 MVA additional power demand required 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. The latest forecast energisation date for the 5 MVA construction power supply is Q3'16.

18.5.4 Alternative Power Supply

18.5.4.1 Diesel Generator Power

There will be a shortfall of between 6 to 9 MVA construction power during the sinking and construction period. This shortfall will be supplemented with diesel generated power and installed by the shaft sinking contractor(s). Adequate allowance has been made in the pre-production capital estimate for the cost of generator sets and fuel consumption.

18.5.4.2 Solar Power

Ivanplats is investigating the feasibility of a 3 MW solar photovoltaic (PV) power system for the Platreef Project. A solar plant cannot provide power at a constant level, but provides a varied output depending on the sun. The Eskom and/or diesel generated power supply must still be able to provide the full demand for periods when solar power is not providing at peak. The intent for this plant will be to reduce generator diesel consumption and electrical power consumption during operation.

18.5.4.3 Emergency Power Supply

Emergency power has been allowed for as a battery of 2 MVA 11 kV containerised generators feeding into the main MV distribution instead of localized generator sets. Winder brakes will have their own generator set.

18.6 Access Roads

Access from Mokopane to Johannesburg, Polokwane and Rustenburg (for concentrate delivery) is via the newly upgraded N1 highway. The Platreef mine 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. Current plans by SANRAL are being implemented to upgrade the N11 national road.

The N11 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 mine.

Current transportation means to and from major mining operations in the area is by road which has resulted in concerns regarding the capacity of the existing N11. Further concerns exist with regards to community safety as the N11 traverses many formal and informal settlements with large volumes of pedestrian traffic.

The South African National Roads Agency (SANRAL). Is considering two options with regards to the N11 road.

- Option 1 is to upgrade the existing road through Mokopane, to cater for the increased traffic volumes.
- Option 2 is to build a reroute of the N11, exiting the N1 north of Mokopane and entering the existing N11 approx. 5 km north of the Platreef mining area.

Neither option has been decided on, however both options will improve the road access to the Platreef mine. Ivanplats has started negotiations with SANRAL to acquire permissions for an intersection on the existing N11 for access to the Platreef mine.

Provincial and regional roads of importance to the Platreef project are the R518 and the Turfspruit Road. The R518 connects Mokopane to Marken, running to the south of the Mine area. The Turfspruit Road is a badly maintained gravel road joining the R101 north of Mokopane with the mine site, as it enters the N11 just south of the Platreef mine area.

18.7 Fuel and Lubrication Offloading and Storage Facility

A fuel and lubrication offloading and storage facility is provided for. The facility comprises 2 off 82 m3 diesel tanks and five 23 m3 lubricant tanks that are adequately sized to cater for 4 days of operation during steady state. Further day tank capacity has been allowed for in the shaft area, at which point the diesel is transported underground via boreholes and lubricants via dedicated cassettes. Diesel distribution on surface will be piped. There is no facility for petrol storage or distribution on site.

18.8 Fire Protection and Detection

The specific fire protection requirements will be developed in conjunction with Platreef's Insurer and a fire consultant during the feasibility phase of the Platreef Project. The guidelines used in the prefeasibility study are described below.

The water supply for fire-fighting will be capable of providing the required firewater flows for any combination of hydrant-monitors, sprinkler systems, standpipe systems, deluge systems and responding fire apparatus based on a single fire event. The firewater distribution system will be designed to supply the specified rate to each fire hazardous area. The firewater system including connections to the system will be designed to provide a high degree of reliability and redundancy. Connections to the firewater system for any other service will not be permitted.

Fire hydrants shall be to an industrial standard. Hydrants shall be located on the roadside of all pipelines and drainage ditches and shall not be located within dyked areas for tanks. Hydrants shall be located within 6 m of the roadway, and in areas granting sufficient water to all offices, workshops and other building structures not equipped with other dedicated fire protection systems

Hose reels will be placed on each equipment structure platform level and distributed inside process buildings wherever fire hazardous equipment is located. At least one hose reel is required per 1,000 m2 of plant area.

- Deluge systems shall be considered for:
- Pumps handling flammable liquids.
- Transformer Bays.
- Conveyor head and tail pulleys and loading points.
- Flammable Liquid Storage areas (Foam System).
- All other hazardous materials areas.

Gas suppression systems shall be considered for all electrical substations.

Hand extinguishers shall be positioned at regular intervals on all levels within buildings.

A fire detection system independent of any Process Control System (PCS)will be required as part of the overall fire protection system, which shall include the fire water system, the inert gas suppression systems and remote tripping units.

Three independent fire water systems, have been allowed for the Platreef Project as follows:

- 600m3 steel sectional panel tank and associated civil works.
- Three fire water pumps: one diesel powered pump, an electrical powered pump, and an electrical jockey pump
- Buried and surface mounted piping to all above mentioned suppression points.

18.9 Waste Facilities

The following waste facilities were allowed for and designed in line with the requirements stipulated in the Integrated Water & Waste Management Plan submitted as part of the legislative requirements:

- Domestic Waste Landfill
- Garden Waste & Composting Facilities
- Sewage Treatment Plant
- Used Tyre & Conveyor Storage
- Hazardous Waste Storage
- Centralized Salvage yard
- Used PPE Storage

18.10 Stockpiling

The following stockpiles and stockpiling facilities have been included:

- Pre-Production High Grade Ore Stockpile
- Low Grade Ore Stockpile
- Waste Rock Stockpiles complete with conveyor systems from both shafts one and two
- Overburden and Waste Rock Perimeter Berm
- Waste Rock Crusher plant to crush waste rock for use as cemented rock fill, backfill material for earthworks and possible future sales of crushed waste rock.

18.11 General Mine Offices, Clinic and Engagement Centre

The following buildings have been allowed for in a general mine office area, close to the mine entrance and is common to both the shaft area and process plant.

- General Mine Office, for Executive Management, Human Resources, Accounting and Finance, Health Safety and Environmental, and General Administration
- Clinic with medical facilities to stabilize a patient prior to transport to a hospital and also with adequate facilities to perform the legislated employee medical examinations.
- Offices for an onsite doctor has been included.
- Engagement centre, where employee engagement and inductions to commence work are to be done.

18.12 Workshops, Stores and Change House Facilities

The shaft area and process plant will be operated as different business units, each with its own workshops, stores, offices and change houses.

18.13 Logistics

The assumption for the pre-feasibility study is that the Platreef concentrate will be sold to local smelters.

It is envisaged that Platreef concentrate will be transported to a smelter in the Rustenburg area via road using covered side tipper trucks with a payload of 32 t to 34 t. This is standard mode of transport for moving concentrate between concentrator and smelter in the area. Platreef will not use owner transport and it is envisaged that this service will be outsourced on a contract basis.

Considering the 4 Mtpa base case scenario and assuming a ~ 4% mass pull through the concentrator, it is expected that 160 ktpa of concentrate will be produced. Standard 32 t to 34 t side tipper trucks are ideally suited for this size of operation. This equates to 392 to 416 truckloads per month or ~14 truckloads per day.

There are various side tipper configurations and the type used will be dependent on the physical properties of the concentrate.

Internal road networks will be such that offloading of materials and loading and removal of concentrate can be done without any congestion. Taking consumables such as diesel, explosives and reagents into account, it is assumed that 20 to 25 trucks will enter and exit the mine property per day, and sufficient allowance has been made to cater for the logistics around this matter.

18.14 Comments

Ongoing liaison with external stakeholders must be maintained with regard to external projects directly affecting the Platreef project. Projects such as the ESKOM upgrades, the ORWRD Project, N11 upgrades need close monitoring and frequent updates to ensure that potential negative impacts to the Platreef project are mitigated as early as possible.

Further to the above, the Platreef Project risk register indicates water and power availability as two of the highest risks to the Platreef Project that require dedicated and frequent interaction with the relevant stakeholders.

18.15 Tailings Storage Complex

18.15.1 Project Requirements

The tailings storage facility requirements of the pre-feasibility study have been as follow:

- Design a tailings storage complex that can accommodate platinum tailings.
- The tailings production rate will be approximately 1.5 Mtpa (Phase 1) increasing to 4.4 Mtpa in Phase 3. A linear backfill model has been assumed for capacity analysis purposes.
- The design life for the tailings storage complex is approximately 30 years.

18.15.2 Design Objectives

- Create a safe and stable tailings storage complex and minimize risk to human lives, health and property.
- The design will be such that it will remain fit for the intended purpose and resist all external environmental influences that are reasonably likely to occur (sustainability).
- The design should conserve all resources as far as possible i.e. land area, water, airspace, topsoil, mineralisation and energy.
- Comply with South African legal requirements and benchmarking against best practice international standards.
- Minimize environmental impacts, where potentially possible.
- Separation of clean and dirty water.
- Minimum storage of supernatant on the tailings storage complex.
- Cost effective construction, operation and closure.
- The tailings storage complex should be situated so that it will not sterilise or be in conflict with any mining activity.

18.15.3 Complex Description

The Rietfontein 2 KS site is the preferred site for the 1.5 Mtpa Phase 1 scenario and 2.9 Mtpa Phase 2 scenario. Together, the Rietfontein 2 KS and Bultongfontein 239 KR sites can accommodate the 4.4 Mtpa Phase 3 expansion scenarios.

The Rietfontein 2 KS site will be developed first as a single compartment side-hill type tailings storage facility with a footprint of approximately 250 ha.

The construction phases for Rietfontein 2 KS can be summarised as follows:

- Phase A: Tailings deposition will initially take place behind a compacted earth starter embankment. The maximum height of the embankment will be approximately 30 m. The embankment construction material will be sourced from the water storage dam location and tailings dam basins and from the Amplats and/or Platreef waste rock dumps.
- Phase B: Low perimeter embankments will be constructed with tailings following the upstream construction method.

The Bultongfontein 239 KR site is proposed to be developed in time for the Phase 3 production scenario. A preliminary footprint for a tailings storage facility of sufficient capacity for Phase 3 has been identified. However, EKSOM has advised that there is an easement for high voltage power lines within the footprint. EKSOM has been approached to negotiate a realignment of the easement.

Representative tailings samples were not made available for laboratory testing during the scoping and prefeasibility study phases. It is recommended that the geotechnical properties of a representative tailings sample should be concluded during the feasibility study phases.

The following investigations were concluded as part of the prefeasibility study:

- Geotechnical investigation.
- Geohydrological investigation and modelling, including groundwater geochemistry. In particular, the final liner system specification should be concluded during this investigation.

Although a liner design has been included in the prefeasibility designs, it is recommended that the final liner system specification for the TSF should be concluded during the feasibility study phase. The air space models should also be optimised during the feasibility phase in order to reflect the final tailings production profiles.

19 MARKET STUDIES AND CONTRACTS

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

19.1 Introduction

Ivanhoe conducted a marketing study for the Platreef 2014 PFS the conclusions of the marketing study confirmed the assumptions of previous studies and have been used as the basis for the realisation and other marketing assumptions in the Platreef 2014 PFS. The marketing assumptions in the Platreef 2014 PFS are based on studies prepared for the PFS, not contracts.

The Platreef 2014 PFS examines the Phase 1 run of mine (ROM) production scenario of 4 Mtpa (approximately 130 ktpa concentrate and ~10 ktpa Ni). This could potentially be followed by multiple expansions to 8 Mtpa, 12 Mtpa and beyond. The optimum combination of sales, tolling and investment in order to provide the metallurgical capacity for the various phases in mining growth is being investigated.

With the advent of a number of smaller PGE mining firms, toll smelting and refining contracts and purchase agreements have become more prevalent in South Africa than in the past. The main PGE mining companies have some internal purchase contracts with their own mining/concentrating operations and external and length purchasing or toll contracts with independent or JV companies. Within the industry and along the value chain there are various possibilities for metal sales contracts: concentrates, furnace and converter mattes, PGE residues or concentrates have all been sold or toll treated in the past. The conclusions of the marketing study have been used as the basis for the realisation and other marketing assumptions in the Platreef 2014 PFS.

Only four PGE companies in RSA have any metallurgical facilities downstream of concentrator operations. PGE concentrate is sold within South Africa and into Europe under long-term contracts. The three major PGE producers have a full suite of process facilities to produce final PGE metal and hence tend to be purchasers rather than sellers of any PGE containing materials. Other PGE 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.

19.2 Trade in Flotation Concentrates

The sale of PGE flotation concentrates from the various Merensky and UG2 mining operations in Southern Africa has increased over the past decades. These agreements are quite variable in net payment for metal contained but will have some of the following elements; smelter charges, refining charges, metal recoveries, pipelines, penalties, delivery terms, assay charges, metal accounting provisions, dispute resolution mechanisms, length of contract periods, renewal conditions, etc. Most significant of these in economic terms are the metal recoveries, any treatment charges, penalties and the pipelines or delays between concentrate delivery and payment. This pipeline for some metals is as long as 9 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 8 to 10 weeks. Flotation concentrates because of their bulk and the South African government's focus on beneficiation do not find a market offshore and are all sold locally.

19.3 Trade in Intermediate Products (Mattes)

The sale of PGE rich intermediate products from further down the value chain such as furnace mattes or converter mattes is less developed as a market and no regular sales of these mainstream products from PGE mining activities are being made.

Some companies have long-term contracts for the sale of their nickel sulphates in place, on occasions there have been sales of nickel-copper matte after magnetic removal of the PGE-containing alloys and there is a long standing contract for very low PGE nickel concentrates in Botswana. Nickel mattes from Botswana are sent to Europe. Small amounts of PGE-rich alloy are traded with various purchasers both locally and offshore. Outside of local buyers, some nickel and PGE products are sold internationally to Asian and European customers. PGE terms from offshore buyers are normally worse but base metal terms can be better.

19.4 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 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 at deleterious and penalties are a possibility for unusual amounts of these contaminants. Purchasers of these concentrates are both local and international, although international buyers do not routinely purchase such concentrates at this time. Local refiners could be quite competitive as their Rand-based operating costs are low compared to the offshore operators but without any regular contracts in place comparison is somewhat speculative.

19.5 Capacity Available Locally

It is sensible to examine capacity at several key points in the process sequence; namely, furnace capacity, converter/acid plant capacity, base metal refining capacity and precious metal refining capacity. Currently furnace capacity is available within South Africa.

The Platreef concentrates will place a high demand on the smelting facilities because of their relatively high sulphur and iron content and the converting and acid plant capacity may become a constraint.

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 this will not be a concern and Platreef concentrate can potentially be used to dilute high chromite containing concentrates.

Further afield in Southern Africa, there is some capacity in Zimbabwe that has been unused since 2009 and in Botswana.

In conclusion there is currently adequate furnace capacity at the South African operations for the first phase of Platreef production as defined in the Platreef 2014 PFS. An investment in sulphur capture and converting operations may be needed. The converter mattes produced from the smelting operations can be sold to offshore nickel producers or refined in the local base metal and precious metal refineries.

Total nickel refining capacity in South Africa has been estimated to be 52 ktpa Ni and in 2013 it was estimated that only 42 kt Ni was used.

The availability of nickel refining capacity could increase if the PGE producers in Zimbabwe build a smelter, BMR and PMR in Zimbabwe. This could add an estimated 7kt Ni of refining capacity in South Africa. The Zimbabwe government is proposing an export ban on raw material containing PGE. PGE concentrates and matte produced in Zimbabwe are currently processed in South Africa. Zimbabwean producers are currently investigating the upgrading of existing smelting and refining infrastructure in Zimbabwe.

Any expansion plans at either refinery would in all probability take several years to implement should more capacity be required locally. Within Southern Africa there is potentially some smelting and converting capacity available for which the PGE recovery would require further definition. PGE mattes could come back to RSA for refining at one of the majors.

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 PGE's are refined locally but Ni-Cu mattes are sent offshore. This would give Platreef competitive base metal terms for the Ni-Cu mattes while allowing the 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.

19.6 Potential for Offshore Processing

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 and matte, and in exploring ways to cooperate metallurgically in South Africa. This interest highlights the importance of the Platreef Project to the PGE industry. Ivanhoe is committed to working within the South African Government's guidelines on domestic beneficiation where local capacities and treatment options are available so there is a high likelihood that the Phase 1 concentrate production profile will be processed in South Africa.

19.7 Smelting and Refining Contracts and Cost Structures

There are many ways of structuring concentrate-tolling contracts and, as noted in section 19.2 above, 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 ton of concentrate to cover drying, smelting, converting and acid production costs. Longer-term deals will see these costs escalated.

Metal losses in the smelting and refining complex as well as base metal and PGE refining costs are often but not always absorbed in the value of the metal retained by the toller; that is, in the lower recovery or percentage payable offered to the tollee. There is reluctance on the part of the local refiners to share too much detailed information in regard to tolling costs/recoveries to protect any competitive advantages they may have however it appears cash costs of smelting are approximately US$115/t to US$170/t.

Base metal refining costs are significant when separated from the metal recovery offered and cost drivers in this operation are essentially the tons of nickel/copper to be refined, although the cost of removing sulphur is significant. Typical refining charges are around 80USc/lb copper and 20USc/lb nickel. 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. The choice of hydrogen reduction versus electro winning coupled with recent increases in electricity costs may explain some of the difference. 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.

The recovery of PGEs from high grade refinery concentrates (40% PGE) is very high and most offshore purchasers of these materials will give payable amounts of around 99% for platinum and palladium and a few percentage points less for rhodium, gold, iridium and ruthenium. PGE refining charges of US$25/oz platinum and palladium, US$10/oz gold and US$75/oz rhodium are expected to be typical.

19.8 Metal Recoveries

Actual metal recoveries by the major PGE company process divisions (smelting, base metal refining and PGE refining) are quite high as is to be expected in light of the value of the metals concerned. Only cobalt recoveries are unusually low as a consequence of the matte smelting and converting technology used.

Table 19.1 Typical Metal Recoveries

Metal Concerned	Recovery Range %
Base minerals	
Nickel	95-96%
Copper	96-98%
Cobalt	30-40%
Precious Metals	
Platinum	96-98%
Palladium	96-98%
Gold	95-98%
Rhodium	95-97%
Iridium	90%+
Ruthenium	90%+

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 payable metals only) 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. These metals have limited markets that have been historically in oversupply creating large stockpiles. 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 so payment for this metal is not often offered.

19.9 Payment Pipelines

The process required to produce pure metals from both refining operations takes significant time and thus a large inventory of metal is held within the process. Metals that enter the smelting operations only appear as refined metal for sale some months later. Each metal flows through the process circuits with a different time distribution and most refiners simply apply a single fixed period to each of the metal values to cover the cost of holding each metal in process. Various residue streams that are recycled to the smelting operations or within the refining operations add significantly to the 'pipeline' effect and impact on the operations cash flow and hence the business returns from a tolling contract.

19.10 Penalties

Penalties can be levied against the seller of concentrates for high moistures, low PGE grades and high chromite levels. Within the smelting and refining circuits, elements such as Fe, As, Bi, Sb, Se, Te, Pb, Zn, and SiO2 can be problematic such that buyers may levy penalties for some of these elements.

19.11 Terminal Pure Metal Sale Agreements

The final refiners of the base metals and PGE's 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 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 basis 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 is always subject to negotiation.

19.12 Conclusion

South Africa has a number of smaller PGE mining companies. Toll smelting and refining contracts and purchase agreements have therefore become more prevalent in South Africa than in the past. The major PGE mining companies have some internal purchase contracts with their own mining/concentrating operations and external and arm's-length purchasing or toll contracts with independent or JV companies. Within the industry and along the value chain there are various possibilities for metal sales contracts: concentrates, furnace and converter mattes, and PGE residues or concentrates have all been sold or toll-treated in the past. The conclusions of the marketing studies have been used as the basis for the realisation and other marketing assumptions in the Platreef 2014

Potential purchasers must consider competitive, cost and capacity pressures while weighing up impurity tolerances with changing feed mixes and process efficiency improvements. Final terms will be significantly influenced by contract term and escalation for both treatment/refining charges and exchange rate movements the estimates made for the Platreef 2014 PFS have been made based on knowledge of concentrate sales contracts that have been agreed in South Africa with local purchasers for historical and current concentrates. Actual terms may vary and will be dependent on the negotiations at the time the contracts are agreed.

Metallurgical work carried out to date on Platreef concentrate has so far indicated that the historically documented range of 80-84% return would equally apply, taking into account the variables and conditions above. The midpoint of 82% has been applied at this stage to the Platreef 2014 PFS analysis. The contractual terms for Platreef and comparable products are highly variable and are subject to change as conditions for any one local or offshore purchaser alter.

The local purchase of furnace mattes or converter mattes has only occurred infrequently and for small volumes, an estimate has been based on the major companies operating costs and an approximation as to their required margin, consequently these figures are 'best estimates' only. The only means of determining the true value of a particular material at a point in time is to approach the various potential buyers and obtain indicative quotations; this is particularly true in regard to the possibility of selling mattes where buying/selling is very infrequent.

It appears that current smelter, converting and acid plant capacity as well as nickel refining capacity are the constraining factors that may even limit the mining rate for Platreef. Sufficient furnace capacity is probably currently available but converting and sulphur removal capacity are probably constrained by equipment and environmental issues.

It does appear that there is some upside capacity for increased trading in concentrates. For instance, Zimbabwe may be successful in the government's goal of providing refining capacity in Zimbabwe for all Zimbabwean PGE's and base minerals. This will impact upon the available smelting, converting, acid capture and nickel refining in South Africa. Beneficiation options in Zimbabwe are currently being explored by local producers.

The PGE mining industry in South Africa is currently in a state of flux. Labour unrest, closures of unprofitable shafts and the threat of an export ban in Zimbabwe are all factors which could free up smelting and refining capacity in South Africa. Expansion plans, in particular at Anglo Platinum's Mogalakwena mine and the reopening of closed shafts will have the opposite effect. At this stage this is difficult to predict and it has been assumed that there will be sufficient smelting and refining capacity in South Africa to accommodate the first phase of the Platreef project by 2020. It has been concluded that the Platreef project is a clear demonstration of the evolution of the South African PGE mining industry. As a highly mechanised, low cost and high grade operation Platreef is expected to be well placed to supply into the PGE market.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

20.1 Introduction

In 1998, Ivanhoe acquired the mineral prospecting licence on the two farms, Turfspruit 241 KR and Macalacaskop 243 KR. In addition, Atlatsa Resources Corporation (Atlatsa) holds the prospecting rights for the Rietfontein 2 KS farm. Ivanhoe applied for a mining right over the Turfspruit 241 KR and Macalacaskop 243 KR farms and the Rietfontein 2 KS farm forms part of the Environmental Authorisation. The Bultongfontein 239 KR farm also formed part of the investigations.

The Platreef Project site lies in a north westerly direction, approximately 8 km from the town of Mokopane (previously known as Potgietersrus) in the Limpopo Province of South Africa. The Platreef Project is situated in the magisterial district of the Mogalakwena Local Municipality and within the Waterberg District Municipality.

There are several communities which fall within the proposed project area that may be affected by this project.

20.2 Legislative Requirements

20.2.1 Local Legislation

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

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

20.2.2 International Standards

All environmental and social studies completed for the proposed project is in conformance with the framework provided in the World Bank Group (WBG) and International Finance Corporation (IFC) policies and guidelines for Environmental Assessments (EA).

With this in mind, the main guidelines that were followed during the development of the Environmental and Social Impact Assessment (ESIA) report are those provided in the Environmental Assessment Sourcebook, Volumes I, II, and III, (World Bank Technical Paper No. 139, 1991), and all relevant updates. There are numerous other WBG and IFC documents that may be considered (if relevant).

20.2.3 Equator Principles

The Equator Principles (EP) is a set of environmental and social benchmarks. Once adopted by banks and other financial institutions, the EPs commit the adoptees to refrain from financing projects that fail to follow the processes defined by the Principles. The EP has become the de facto standard for banks and investors on how to assess major development projects around the world. The Principles apply to projects over 10 million US dollars.

20.3 Platreef Licences and Permits

20.3.1 Mineral Rights and Mining Right Application

Ivanhoe was the holder of a converted old-order prospecting right over the Macalacaskop 243 KR and Turfspruit 241 KR farms (Prospecting Area) granted in terms of Schedule II of the MPRD Act, which entitled it to exclusively prospect for base minerals and precious metals in, on and over the Prospecting Area. The Prospecting Right was registered in the Mineral and Petroleum Titles Registration Office under MPT 55/2006 PR and was renewed for further period ending 31 May 2014, which renewal was also registered in the same office under MPT 38/2011 PR.

An application in terms of Section 20 (2) of the MPRD Act for ministerial consent to do bulk sampling on the Prospecting Area was lodged on 21 September 2012 and was approved on 29 August 2013.

The Prospecting Right granted Ivanhoe the exclusive right to, before the expiry date of the Prospecting Right, apply for a mining right in terms of Section 22 of the MPRD Act over the Prospecting Area for the same minerals.

The MRA was submitted to the DMR electronically through the SAMRAD portal on 6 June 2013, and was accepted by the Regional Manager on 17 July 2013. The mining right was approved on 30 May 2014 but the notarial execution is still pending. The application number in respect of the mining right is LP30/5/1/2/2/10067MR.

20.3.2 IWULA, WULA and NEMWA

20.3.2.1 General Authorisation, Integrated Water Use Licence Application and Integrated Waste Water Management Plan

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 Ivanhoe 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 Ivanhoe 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 Ivanhoe.

A meeting with four members of the DWS Regional office in Polokwane was held at the Ivanhoe Mokopane offices on 17 February 2014 to discuss the BSS IWULA. This comprised a detailed presentation of the application and a question and answer session. This meeting was followed by a site inspection of the BSS and additional information was requested in support of the application.

20.3.2.2 IWULA Main Mine

A meeting was held at the DWS Polokwane offices on 23 April 2014 to present and submit the Platreef IWULA for the main mine. The additional information requested during this meeting relating to the water balance was subsequently submitted to the Department.

An engineering design review meeting took place with Mr Kelvin Legge and his team from the National DWS Department. All the engineering designs excluding those for the Tailings Storage Facility (TSF) were reviewed and amendments to some aspects requested.

20.3.3 National Environmental Management Act (NEM Act)

An application in terms of the NEM Act, reference number 12/1/9/2-W32; LIM/EIA/0000538/2013, was made to LEDET. Environmental Authorisation was received on 27 June 2014 which stipulate certain conditions that the Platreef Project must adhered to over and above the commitments from the EMP. Amendments of some conditions were applied for and the amended authorisation was received on 19 September 2014 from LEDET. Possible further amendments are considered.

20.3.4 Other Applications

Further applications in terms of other applicable legislation, including, inter alia, the Precious Metals Act No 37 of 2005, will be lodged to ensure compliance once Platreef moves into the mining phase.

20.4 Terms of Reference

Baseline studies were undertaken within the Platreef Project area, in support of an ESIA in support of the MRA as well as all the other environmental licences applied for under South African legislation. These studies were conducted to comply with national legislation and international requirements.

The ESIA summarises relevant results of the environmental and social baseline of the Platreef Project area. Future studies will be a continuation of the baseline studies and will be in line with conditions stipulated in the Environmental Authorisations, commitments made in the ESIA and legislative requirements.

20.4.1 Environmental and Social Impact Assessment Process

The following activities were conducted during the compilation of the ESIA to comply with national and international requirements.

20.4.1.1 Scoping Phase

- Compilation and distribution of the Stakeholder Engagement Plan (SEP) announcement documents.
- Pre-consultation authority and community meetings.
- Project categorisation in terms of the IFC Principles.
- Compilation of legislative application forms.
- Specialist baseline and impact identification report compilation.
- Compilation of a draft scoping report (DSR).
- Public Review of the DSR.
- Community feedback meetings.
- Compilation of a SEP interim report.
- Compilation of a final scoping report (FSR).
- Submission to the relevant authorities for authorisation.

20.4.1.2 EIA Phase

- Conduct specialist investigations and compile specialist reports.
- Compile Environmental Legal Register for construction, operations and closure.
- Compile the following draft reports.
- MPRD Act compliant ESIA report.
- NEM Act and IFC compliance ESIA report.
- Draft Environmental Management Plant (EMP).
- Public Review of the above-mentioned draft documents.
- Community feedback meetings.

- Compilation of a SEP report.
- Compilation of final ESIA and EMP Reports.
- Submission of final reports to the relevant authorities.

20.4.1.3 Permitting Requirements

The following application forms were submitted during the course of the proposed project:

- IWULA in terms of the NWA.
- Integrated waste management license application in terms of the NEMWA.

20.4.1.4 ESIA Specialist Studies

The following specialist studies were conducted during the ESIA process to ensure compliance with local and international requirements:

- Visual and Topography Assessment.
- Heritage and Archaeology.
- Aquatic Ecology and Wetlands.
- Surface and Groundwater Investigations.
- Fauna and Flora.
- Air Quality.
- Noise Assessment.
- Soils, Land Use and Land Capability.
- Traffic Assessment.
- Socio-economic Assessment.
- Community Health Assessment.
- Rehabilitation and Closure.

20.5 Summary of Baseline Results

20.5.1 Topography Assessment

The topographical model indicates that the elevation of the Platreef Project area increases from 1 030.5 metres above mean sea level (mamsl) in the Mogalakwena River floodplain in the south-western corner of the Platreef Project area to 1 759 mamsl on the ridges in the north-eastern corner of the Platreef Project area.

The majority of the Platreef Project 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 Platreef Project area is generally in a south-westerly direction towards the Mogalakwena River. Slopes of various other directions occur in isolated areas along the river valleys/channels and on the ridges.

The relatively flat topography of the Platreef Project area will only provide minimal screening of the Platreef Project. The mountainous areas to the east and west of the Platreef Project area will provide screening of the Platreef Project to those areas on the opposite sides of the mountains.

20.5.2 Visual Assessment

Two viewshed models were completed for the Platreef Project. The first was a theoretical viewshed model, while the second is known as a mitigated viewshed model.

The theoretical viewshed model depicts the area from which the Platreef Project will potentially be visible. The theoretical viewshed covers an area of approximately 663 km^2. The second viewshed model for the Platreef Project illustrates the potential mitigation effect of vegetation screening. The mitigated viewshed model depicts the area from which the Platreef Project would potentially be visible if the existing noise berm was used as a vegetation screen. This viewshed covers an area of approximately 631 km^2.

The receptors identified within the theoretical viewshed area include residents of the town of Mokopane as well the following villages:

- Ga-Kgobudi
- Ga-Madiba
- Ga-Magongoa
- Ga-Mapela
- Ga- Masenya
- Ga- Molekana
- Mahwelereng
- Masodi
- Moshate
- Mzombane
- Phola Park
- Sekgakgapeng
- Tshamahansi

The villages of Ga-Kgobudi, Ga-Madiba, Ga-Magongoa, Mzombane and Tshamahansi are closest to the proposed development and are therefore expected to experience the highest visual impact. The theoretical viewshed model indicates that the Platreef Project will potentially be visible from the N1 and N11 national routes and the R101 and R518 regional routes as well as other smaller roads within the Platreef Project area. The southern part of the Witvinger Nature Reserve will potentially be visually affected by the Platreef Project.

The mitigated viewshed model indicates that the screening effect of the vegetated noise berm will result in the village of Ga-Masenya no longer being visually impacted on by the Platreef Project.

20.5.3 Heritage and Archaeology

20.5.3.1 Geological Background and Paleontological Potential

Most of the development area is underlain by Precambrian igneous rocks of the Rustenburg Layered Suite (RLS) of the Bushveld Complex. The south-west section part of the property is underlain by the Molendraai Magnetite Gabbro of the RLS. The south-eastern portions of the property are underlain by the Duitschland Formation and the Malmani Subgroup of the Chuniespoort Group. To the extreme south-east, a small section of the property is underlain by the Uitloop Granites of the Mashashane Suite. The Bushveld Complex is a layered igneous intrusion containing a large reserve of PGEs (Lee, 1996; Eales & Cawthorn, 1996). Associated with this complex is the RLS known to be the oldest mafic layered complex on earth (Wilson, 2012). As these rocks are Precambrian in age and of igneous origin it is unlikely that fossils will be affected. The Malmani Subgroup generally comprises dolomite, interbedded chert and shales, quartzite, and a variety of stromatolite structures. The dolomitic rocks this subgroup will contain stromatolites and will also have the potential to have sinkholes and caves which may have Quaternary deposits.

20.5.3.2 Archaeological and Historical Background

Makapansgat Valley World Heritage Site

The Makapansgat World Heritage Site (WHS) which is about 20 km east from the Platreef Project area is part of a group of sites that were nominated as a collection of sites that display the same or similar characteristics. This group includes sites such as fossil hominid sites of Sterkfontein, Swartkrans, Kromdraai and environs as well as the Taung Skull Fossil Site. The sites of Sterkfontein, Swartkrans and Kromdraai were inscribed on the World Heritage Site list in 1999 and received an extension in 2005 to include the Taung Skull Fossil Site and Makapansgat (UNESCO, 2013).

According to the United Nations Educational, Scientific and Cultural Organization (UNESCO) website, "Fossils found in the many archaeological caves of the Makapan Valley have enabled the identification of several specimens of early hominids, more particularly of Paranthropus, dating back between 4.5 million and 2.5 million years, as well as evidence of the domestication of fire 1.8 million to 1 million years ago" (UNESCO, 2013).
The sites as a whole were nominated to become UNESCO World Heritage Site according to the following criteria:

- Criterion (iii): The nominated serial site bears exceptional testimony to some of the most important Australopithecine specimens dating back more than 3.5 million years. This therefore throws light on to the origins and then the evolution of humankind, through the hominisation process.
- Criterion (vi): The serially nominated sites are situated in unique natural settings that have created a suitable environment for the capture and preservation of human and animal remains that have allowed scientists a window into the past. Thus, this site constitutes a vast reserve of scientific data of universal scope and considerable potential, linked to the history of the most ancient periods of humankind.

- Integrity (2005): The Fossil Hominid Sites of Sterkfontein, Swartkrans, Kromdraai and environs together with Makapan Valley and Taung Skull Fossil Site comprise five separate components situated in different provinces and each has a buffer zone. Collectively these components contain the necessary evidence of sites where abundant scientific information on the evolution of modern humans over the past 3.5 million years was uncovered. Furthermore, the nominated serial site covers an area big enough to constitute a vast reserve of scientific information, with enormous potential.
- Authenticity (2005): As regards to authenticity, the sites contain within their deposits all of the key interrelated and interdependent elements in their natural palaeontological relationships. Thus, the breccia representing the cave fillings contains the fossilised remains of hominids, their lithicultural remains (from about 2.0 million years onwards), fossils of other animals, plants and pollen, as well as geochemical and sedimentological evidence of the conditions under which each member of the deposits was laid down. They represent a succession of palaeo-ecosystems. The caves, breccias and strata from which quantities of fossils or tools have been extracted, together with the landscape are generally intact, but are vulnerable to development pressures, villagers' use of the environment and tourism.

All the sites are protected as National Heritage sites in terms of the National Heritage Resources Act, 1999 (Act No. 25 of 1999) (NHRA). In terms of this legislation, no person may destroy damage, deface, excavate, alter, remove from its original position, subdivide or change the planning status of any heritage site without a permit issued by the heritage resources authority responsible for the protection of such site. The property size of the Makapansgat WHS is 2 220 ha, while the buffer zone extends 48 065 ha around the site according to the Government Gazette GR. 1197 of 2007.

Stone Age

Evidence suggests that the region surrounding the Platreef Project area has been inhabited during all periods of the Stone Age, which are the Early Stone Age (ESA), Middle Stone Age (MSA) and Later Stone Age (LSA). This is most evident and extensively documented at the Cave of Hearths in the Makapans Valley some 20 km to the east (McNabb & Binyon, 2004; Phillipson, 2005).

Previous impact assessments (Huffman, 1997; Fourie, 2002; Pistorius, 2002; Roodt, 2007; Roodt, 2008a; Roodt, 2008b) conducted within and surrounding the Platreef Project area have all reported stone tool scatters associated with the MSA and LSA. Fourie (2002) also reported on a possible ESA core found on the surface. These finds are commonly associated with water sources, such as rivers and pans. LSA stone tools are commonly associated with hunter-gathers, but are also known to occur with Iron Age communities.

Resounding rocks or "rock gongs" are features that are often associated with the San/Bushmen culture. These are natural occurring ironstone boulders which either rest on top of ironstone rocks or other rocks that have natural resonating qualities. While these features are natural and occur all over the country, not all show signs of human interaction and use. The area which was constantly beaten to produce sound shows a distinct difference in surface patina to the surrounding cortex of the rock. The rocks were either beaten by hand, other rocks or pieces of wood. The "rock gongs" were often used in rain-making rituals and medicine dances in which the concussive and resonating sound helps the shaman enter a trace like state in which he/she enters the "Spirit World" to conduct ritual activities (Ouzman, 2001).

Iron Age

Based on ceramic distributions as defined in Huffman (2007), the Platreef Project area may possibly produce sites that span from the Early Iron Age through to the Late Iron Age (LIA). Several Eiland facies ceramics have been identified in the region surrounding the Platreef Project area (WITS, 2010). Huffman (1997) identified two 'Moloko' settlements in the region dating to approximately 1500 CE – 1600 CE and several have been recorded by the University of the Witwatersrand. Based on these dates and ceramic distributions, these sites are likely associated with the Madikwe facies of the western Sotho-Tswana. It is also possible that these ceramics belonged to the Ndebele that also occupied the area but whose ceramics belonged to the Letaba or Moloko Traditions (Loubser, 1994). Sites recorded on the University of the Witwatersrand Archaeological Database (WAD) indicate that several Ndebele sites occur around this Project area. Ethnographically, the Ndebele of the region are divided into two groups with claims to similar origin in the north-west of Kwa-Zulu Natal. It is from here that they moved into the Gauteng and Limpopo region during the 16th – 17th century where they settled and subdivided into separate groups.

Metal smelting sites are prevalent within the North-West Province near Zeerust Rooiberg and the Waterberg region in Limpopo approximately 150 km south-west from the Platreef Project area (Boeyens, Küsel, & Miller, 1995). Evidence of iron, tin and copper smelting is present in these areas with smelting furnaces, tuyere pipe fragments and slag excavated from sites near Rooiberg, North West province (Miller & Hall, 2008).

Historical Period

By the 19th century, several local Ndebele communities occupied the region around the Platreef Project area, one of the most prominent being the Kekana. In 1837, the Boers arrived at Louis Trichardt marking the first contact between the Boers and Ndebele (Naidoo, 1987). During the latter part of the 19th century the Boers assumed control over the slave and ivory trade after the establishment of the town Piet-Potgietersrus (later Potgietersrus and today Mokopane) in the 1850's causing tension between the two groups (Tobias, 1945; Bonner, 1983; Delius & Trapido, 1983; Hofmeyr, 1988; Esterhuysen, Sanders, & Smith, 2009; Esterhuysen, 2010). Three incidents resulting from tensions between the Ndebele and the Boers culminated in the infamous Mugombane siege of 1854 at Historic Cave in the Makapans Valley (Tobias, 1945). After this siege in 1858 a second group of Ndebele, the Langa of Hlubi (Nguni) origin under the Chief Mankopane, were attacked by a Boer expedition. Approximately 800 Langa Ndebele were killed. After their defeat, Chief Mankopane settled on Thutlwane Hill which is today located on the Kromkloof 744 LR farm, approximately 40 km north-west of the Platreef Project area (Jackson, 1969; Jackson, 1982). After these incidents, the Ndebele wanted nothing to do with Boers or Europeans. With regards to literacy, writing was seen as 'Boer business' and in 1864 the Ndebele refused to adopt it (Hofmeyr, 1991). Despite this, in 1865 the Berlin Mission Station was given permission to establish a mission under W. Moschutz at the foot of Sefakaola Hill (Macalacaskop) on whose summit resided the capital of Mokopane's chiefdom. Tensions between the Boers and Ndebele resulted in the mission station's abandonment and use by the Boers as a garrison where they could fire upon Mokopane's chiefdom, ultimately resulting in the destruction of the mission station. The mission was reoccupied in 1868 but in 1877, Mokopane exercised his authority and ousted the missionaries as he decided that it was a good vantage point for his enemies to spy on him. The chief erected an iron structure from the remains of the station as a symbol of his resistance to European interference.

In 1890, Mokopane died and his successor was Lekgobo Valtyn. Valtyn's view of literacy was different to that of Mokopane as he embraced the idea of literacy and saw it as a resource that could be exploited (Hofmeyr, 1991) and therefore allowed the mission station to be rebuilt.

Also in 1890, a 'location' was unofficially established named after Chief Valtyn. By the early 20th century the Berlin Mission Society began to fence off portions of land which again caused tension between local inhabitants and Europeans resulting in what was termed 'The Fence War' (Hofmeyr, 1990). It was believed that Europeans were stealing land from local inhabitants. Plans for the official establishment and expansion of the location are evident in a letter dated 6 January 1937 between the Controller of Native Settlements and the Deputy Director of Native Agriculture, where it was discussed that the establishment of the Valtyn Location on the edge of Potgietersrus was intended to provide the growing town with a large cheap labour supply (National Archives and Record Service, 1996). Some measures at mitigating this tightening of control over the land in the area were attempted by Chief Kutter Seleka in the early 1930s. This included the proposed purchase of farms bordering the location, in order to try and extend the pasture for cattle. The Rietfontein 2 KS farm was eventually bought with the aid of a bond taken out at the Transvaal Consolidated Land and Exploration Company (Ltd) (TCLEC) by Chief Kutter Seleka, and his followers. The bond was granted with interest set at 6%. Rietfontein 2 KS was bought by the Kekana under Chief Seleka for a sum total of £1983 in November 1929 (National Archives and Record Service, 1996).

The present day settlements of Tshamahansi, Mahwelereng, GaMadiba, Maroteng and Masodi are situated on the three farms, Rietfontein 2 KS, Turfspruit 241 KR, and Macalacaskop 243 KR, which were originally expropriated from the local farmers.

20.5.3.3 Current Status of the Heritage Environment on the Platreef Project Area

The Platreef Project footprint area consists mainly of agricultural land, grazing land and road servitudes. These areas have been previously impacted upon due to agricultural and pastoral activities, as well as the construction of roads and exploration drill rigs. Medicinal plants that were identified during the fauna and flora assessment, as well as through consultation were found to occur across the Platreef Project area; however, they are not endemic to the Platreef Project area. According to Dr Mohatla (the District Chairperson of Traditional Health Practitioners for the Waterberg Municipality), the plants are highly significant to the Traditional Healers within the community; however, they can be sourced elsewhere.

The two archaeological sites identified at the proposed Operational Area and the Alternative Plant area share similar characteristics, such as a mixture of circular and rectangular stone foundations, and both sites are located in areas dominated with Aloe sp. While one site has monolithic stone walling and a "gong rock", it can be assumed that they were settled at the same time and both settlements may have been inhabited at the same time as they share similar characteristics. During times of peace, these sites with good access to grazing and agricultural areas would have flourished within the floodplains. Features such as "gong rock" are usually associated with hunter-gatherers; however, this example may have had a role within the Iron Age/Historical community that resided in the settlement nearby. These sites may also have a link to the historical Ndebele that resided on Sefakaola Hill (Macalacaskop 243 KR), the capital of Mokopane's chiefdom in 1854 approximately 5 km south from the proposed Operational Area and the Iron Age/Historical community.

The smelting site identified at TSF location option 2 (Rietfontein 2 KS farm) may be a representation of a different group and time period, as the stone walling is different to that of the archaeological sites at the operational area and the Iron Age/Historical community. Smelting sites are not common within the Mokopane region and this site has the potential to broaden the archaeological model of the Iron Age of the Limpopo Province. There are still community members within Tshamahansi who remember that a group was living in the hills behind the present day village and recall that they smelted iron.

Although no signs of settlements were identified within the TSF site 3 (Bultongfontein 239 KR farm) area, community members from Machikiri have strong cultural, historical and spiritual ties to the area due to their on-going rain rituals that are performed on the mountain to the east of the Platreef Project area and the collection of medicinal plants.

According to background research, the werf identified along the TSF location option 3 (Bultongfontein 239 KR farm) pipelines is a common representation of a typical structure that can be identified in the surrounding areas. Though it could not be fully accessed during the time of the survey and only a cursory assessment was conducted, it is already impacted upon by the Olifants River Water Resources Development Project and the N11 with its associated infrastructure. The werf may have historical ties to the Witvinger Nature Reserve as it is situated at the entrance to the reserve.

Burials identified within Project areas are mostly recent and are still connected to those inhabitants residing in nearby villages. They are part of the living heritage of the communities and are significant as shown by the comments made during public meetings. Community members still visit their ancestors as shown by various burial grounds showing signs of on-going maintenance

During the Heritage Impact Assessment survey, a total of three archaeological sites, one historical werf and 55 burial grounds were identified within the Platreef Project areas. Areas associated with intangible heritage were identified within the TSF Option 2 and 3, the Alternative Plant area and the Operational Area. All of these sites are located in proposed infrastructure footprint areas and will be impacted on by the proposed development.

20.5.4 Aquatic Ecology

A total of two types of systems were identified for the Platreef Project area, namely the Mogalakwena River floodplain and the Rooisloot, Ngwaditse and Dithokeng Rivers. The Rooisloot, Ngwaditse and Dithokeng rivers are ephemeral systems and were predominantly dry during the field survey periods. Water was noted in the lower reaches of the Rooisloot and Ngwaditse rivers and it was concluded that this is largely attributed to household effluent. No water was noted in the upper catchment areas of these systems, supporting this conclusion. As a result of this, an ecological state assessment of these systems could not be conducted.

In order to establish the ecological integrity of the associated aquatic ecosystems, several sites were selected on the Mogalakwena, Groot-Sandsloot, Ngwaditse, Dithokeng, Nyl, Rooisloot and Dorp rivers associated with the Platreef Project area.

A total of seven sampling points were selected for the study. The Global Positioning System (GPS) coordinates for each of the sampled sites are given in Table 20.1.

Table 20.1 GPS Coordinates and Short Descriptions of the Various Study Sites

Site name	Coordinates	Description
PLA 1	23°59'35.92"S 28°57'34.24"E	The site is the intended downstream sampling point for the Groot-Sandsloot River. It was characterised by soft mud with steep eroded flood banks. The channel itself looked to have been recently mechanically dug.
PLA 2	24°3'44.22"S 28°58'48.49"E	This site is a flood plain of water trapped behind a sand berm. It appears to be fed by the drainage channels of the Dithokeng River located to the north east of the water body. Approximately 250 metres above the system sand mining is taking place.
PLA 3	24°6'25.21"S 29°1'40.67"E	This is the upstream site of the Rooisloot River, it had a moderately wide channel and 100 m above the road crossing was large dam wall that had fallen into disrepair. The site was dry during the site visit.
PLA 4	24°8'11.46"S 28°57'49.96"E	This was the mid-stream site for the Rooisloot River, it runs through a high density settlement. The stream was flowing and bedrock sand and gravel were present.
PLA 5	24°5'39.94"S 28°54'5.62"E	This was the downstream site of the Mogalakwena River. The site was characterised by alien riparian vegetation (Eucalyptus sp.). Dry grassy sandy channel.
PLA 6	24°10'21.01"S 28°59'11.67"E	The site was at the N11 road crossing of the Dorps River, The water was coloured white, large amounts of litter was scattered on the eroded banks.
PLA 7	24°16'32.56"S 28°58'31.55"E	The upstream site of the Mogalakwena River was dry with wide banks. A large amount of grasses had grown within the channel
PLA 8	23°59'40.34"S 28°59'25.88"E	This is the site of a dam built along a minor drainage channel at the time of survey it was dry but a single frog was found
PLA 9	24° 0'9.74"S 28°58'59.63"E	This site falls within the proposed TSF site. The river bed was dry although water still persisted in the dam below.
PLA 10	23°56'57.17"S 29° 0'23.97"E	The stream flow here originated from a seepage point at the base of the dam wall. Fish and invertebrates were found in the pools that formed at the head the stream.

20.5.4.1 Water Quality

Organisms which are present within freshwater ecosystems are directly affected by water quality. It is therefore essential to collate the water quality data in order to understand the responses of biota within the freshwater systems. The assessment of water quality of local river systems is based on selected in situ variables.

The in situ water quality analysis for the low flow period of 2013 indicated that the water quality at site PLA 1 was within acceptable pH and temperature range, however, conductivity was elevated and dissolved oxygen was below guideline levels. The high conductivity of the river is most likely associated with the pollution in the stream. The levels recorded are elevated and this could be negatively effecting the in stream biota.

The in situ water quality analysis for the low flow period indicated that the water quality at sites PLA1, PLA2, PLA4 and PLA6 was poor with conductivity and dissolved oxygen concentrations being out of the recommended DWAF (1996) guidelines values.

The elevated levels of conductivity may be attributed to the associated urban pressures these rivers find themselves under, namely the discharge of chemicals and untreated effluent would increase the levels of conductivity and could negatively affect aquatic biota.

The levels of dissolved oxygen were a concern as oxygen is the most important measure of water quality for aquatic biota (Mason, 1991). Levels below 5.0 mg/l (Kempster et al., 1980) were seen to negatively affect aquatic biota and the levels of oxygen at PLA1, PLA2, PLA4 and PLA6 may be negatively affecting the aquatic biota during the low flow period.

The in situ water quality associated with the Platreef Project area is considered to be in a poor and degraded state. Several signs of sewage effluent and urban runoff were present within the associated river systems as seen in the below figure. Results from the surface water analysis confirm the above statement and refer to eutrophic conditions and high concentrations of chloride and nutrients.

20.5.4.2 Habitat

The Index of Habitat Integrity (IHI) assesses the number and severity of anthropogenic impacts and the damage they potentially inflict on the habitat integrity of aquatic ecosystems.

The current land-uses have impacted on the functioning of this system. Local agricultural practices, pertaining predominantly to livestock have impacted on the ability of this system to provide important services. Agricultural activities have altered the natural hydrology of the system. The decrease in surface roughness due to overgrazing has resulted in a potentially destructive hydrological regime for the system. In addition to this, livestock also impact directly on the quality of water as a result of nutrient input and trampling of the system. Owing to the fact that agricultural practices are on-going for the Platreef Project area, coupled by the absence of mitigation measures for the current land-uses, it is assumed that the ability of the units to provide important ecological services will continue to deteriorate. The severity of the current identified impacts was however determined to be minor at this stage.

20.5.4.3 Fish Assessment

The use of fish as a means to determine ecological disturbance has many advantages (Zhou et al., 2008). Fish are long living, respond to environmental modification, are continuously exposed to aquatic conditions, often migratory and fulfil higher niches in the aquatic food web. Therefore, fish can effectively give an indication into the degree of modification of the aquatic environment. The River Health Programme (RHP) uses the Fish Response Assessment Index (FRAI). The FRAI is based upon the preferences of various fish species as well the frequencies of occurrence in which the species occur.

Electroshocking was carried out in rivers that contained water during the field study. The expected species of the A61F and A61G quaternary catchments was adapted.

The FRAI assessment was adjusted to suit the site specific requirements with the Frequencies of Occurrence (FROC) of particular species adjusted from the expected species list (Kleynhans et al., 2007). The FRAI and FROC have been adjusted according to the following factors: sampling effort, habitat type, cover combination, stream lengths and altitude.

Based on the outcome of the fish assemblage assessment the fish community associated with the Platreef Project area can be considered to be largely modified. This largely modified state of the fish community is a result of poor water quality compounded with low water availability. Many species of the fish are believed to be present within the refuge areas in the local impoundments. Due to the reliance of local communities on the fish as a protein source it is important to maintain these aquatic systems.

Many of the absent fish species such as Chiloglanis pretoriae are sensitive to pollutants and modified flow regimes. The absence of these species confirms that the water quality as well as the in-stream habitat of the associated river systems is currently largely modified.

Oreochromis mossambicus was present during the field study. This organism is a Red Data species.

20.5.4.4 Macroinvertebrate Assessment

As a result of aquatic macroinvertebrates integrating the effects of physical and chemical changes in the aquatic ecosystems, they are good, short-term indicators of ecological integrity. Integration of biological indicators (like aquatic invertebrates) with chemical and physical indicators will ultimately provide information on the ecological status of a river (RHP, 2001).

Habitat for Aquatic Macroinvertebrates

The reaches which were assessed consisted of a variety of biotopes with each of the systems comprising of different habitat structures. The dominant feature of the invertebrate habitat is the sandy-clay substrate which dominates the river systems under study. Generally, no stones in or out of current biotope were found to be available at any of the sites except for PLA 4 where bed rock and small stones were present. During both surveys aquatic and marginal vegetation was limited due to low flow volumes. Flow velocities during the surveys were also found to be low or not discernible. Four of the seven sites visited were found to have water in them.

The Invertebrate Habitat Assessment System (IHAS) results were found to be poor to inadequate for PLA 1, PLA 2, PLA 4 and PLA 6. This was largely due to sandy benthic conditions coupled with poor riparian vegetation. Flow rates were below 1 m/s at all sites containing water. PLA 2 is a standing water body with no flow. PLA 3, PLA 5 and PLA 7 were all dry sites and could therefore not be assessed. The poor habitat conditions would not be able to support a large degree of species diversity within the invertebrate taxa.

South African Scoring System (version 5)

The findings of the macro-invertebrate assessment for the system recorded taxa with sensitivity scores ranging from highly pollution tolerant to moderately pollution tolerant. A large variety of taxa with low tolerances to pollution were found in the study site associated with the Platreef Project area.

According to Kleynhans (2000) the associated sites consist of aquatic biota that is moderately sensitive and of a moderate ecological importance. During the current surveys (2013) no sensitive organisms were found. The absence of these sensitive taxa confirms the classification of Kleynhans (2000).

The SASS5 scores for the low flow survey ranged from 4 at site PLA 6 to 37 at site PLA 4. The ASPT ranged from 1.3 at site PLA 6 to 5.28 at site PLA 4. The SASS5 scores were then placed into the biological bands based on Dallas (2007). According to Dallas (2007) the sites associated with the Platreef Project area are considered to be within the E category indicating that the macroinvertebrate community is present in a seriously modified state.

According to the SASS 5 interpretation guidelines there is a major deterioration in water quality at all of the sites investigated during the field study. The results of the in situ and FRAI corroborate this finding. Additionally, only pollution tolerant species were found to be present at the selected sites. The IHAS assessment revealed that the invertebrate habitat at the sites were inadequate to support a diverse community of invertebrate. Although the habitat was determined to be inadequate sensitive species should still be present. The complete absence of sensitive species is indicative of water quality impairment.

The seriously modified SASS 5 category confirms the observation of the negative effects and presence of sewage effluent and urban runoff.

Macro-Invertebrate Response Assessment Index

In order to compressively understand the structure and status of the invertebrate population, the Macro-invertebrate Response Assessment Index (MIRAI) was implemented. The MIRAI was implemented based on the collective score of the sites associated with the Platreef Project area and is considered as per the reach of the river assessed.

The results of the MIRAI indicate that the invertebrate community that is currently present is in a seriously modified state. The invertebrate communities present at all the sites of the current study are indicative of modified water quality. This is confirmed by the absence of pollution sensitive species from the selected sites.

The majority of the sample sites are located within non-perennial river systems and therefore, confidence in the invertebrate assessment is low. Based on the results of the SASS 5 and MIRAI, the invertebrate communities present at the sites are in a seriously modified state. Based on the findings of the in situ water quality analysis, as well as previous baseline information, eutrophication and the concentration of salts, as a result of evaporation and sewage effluent, have negatively influenced the water quality of the associated river systems resulting in a seriously modified state of invertebrates.

20.5.4.5 Integrated Ecological State

Although the RHP does not take the water and habitat quality into consideration when determining the eco-status of a system, it is noted for the purposes of transparency that the sites associated with the Platreef Project had impaired water quality and modified habitat states. The final eco-status for the associated sites was determined to be a Category E meaning the conditions at the biological communities present at the sites are in a seriously modified state.

The reason for the seriously modified biological community is a result of impaired water quality. Water quality modification is occurring in the form of treated and untreated sewage effluent resulting in eutrophication at sites as well as the influx of urban runoff. These factors are compounded by low rainfall and high evaporation leading to water that has a high level of dissolved salts with a low concentration of dissolved oxygen.

When the current study is compared to the ecological and management categories for the quaternary catchments set out in Kleynhans (2000), the following findings can be noted: The PESC of river reaches included in this study is not largely natural (Class B), but the current PESC is a Class E. The ecological importance and sensitivity as described in Kleynhans (2000) was moderate; the current study sampled aquatic species which were of importance (Oreochromis mossambicus) and therefore, the ecological importance is seen as high.

20.5.5 Wetlands

20.5.5.1 General Description of the Wetland Systems

The Floodplains

The floodplain surface usually slopes away from the channel margins as a result of preferential sediment deposition along the channel edges and areas closest to the channel which can then result in the formation of backwater swamps at the edges of the floodplain margins (DWAF, 2007). According to Kotze et al. (2007) floodplains usually receive most of their water during high flow events when waters overtop the stream banks. According to McCartney (2000) flood attenuation is likely to be high early in the season until the floodplain soils are saturated and the oxbows and other depressions are filled. Additionally, the flood attenuation capacity is drastically reduced in the late season. It is unlikely that floodplains contribute significantly to stream flow regulation (Kotze et al., 2007). The contribution of water from floodplains to stream flow and groundwater recharge is limited as a result of the clayey floodplain soils which retain water (Kotze et al., 2007).

20.5.5.2 Channelled Valley Bottom Systems

According to Kotze et al. (2007), channelled valley bottom systems are characterised by less active deposition of sediment and an absence of oxbows and other floodplain features such as levees and meander scrolls. These wetland types tend to be narrower and have somewhat steeper gradients and the contribution from lateral groundwater input relative to the main stream channel is generally greater. The primary cause of this channelling is the result of erosion (Kotze et al., 2007).

20.5.5.3 General Functional Description of the Wetland Systems

The Floodplain

According to Kotze et al. (2007) floodplains are considered to be important for flood attenuation because of the nature of the vegetation and the topographic setting that they occupy. The velocity of flow decreases laterally as the flood overtops the river banks, thus allowing for the deposition of particles within the floodplain landscape (Kotze et al, (2007). According to Hemond and Benoit (1998) phosphorous and other toxicants bound to trapped sediment are likely to be retained on the floodplains and this is a vital mechanism through which wetlands trap phosphates. According to Kotze et al. (2007) nitrogen removal via denitrification is likely but also limited due to the short flooding periods. Additionally, due to the dilution effects, the concentration of nutrients in flood waters entering the floodplain is often low (Kotze et al., 2007).

Channelled Valley Bottom Systems

A key benefit of the valley bottom wetlands with channels associated with both farms is the enhancement to the quality of water. According to Kotze at al. (2007) these wetlands contribute less towards flood attenuation and sediment trapping, but would supply these benefits to a certain extent. These wetlands would thus provide a service through limited flood attenuation by the spreading out and the slowing down of floodwater in the wetland, thereby reducing the severity of floods downstream and by trapping and the retention in the wetland itself of sediment carried by runoff waters. Additionally, these wetlands would offer some nitrate and phosphate removal potential, particularly from the water being delivered from the adjacent hillslopes (The Federal Interagency Stream Restoration Working Group, 1998).

20.5.5.4 Floodplain Functional Description

The general features of the wetland units were assessed in terms of functioning and the overall importance of the hydro-geomorphic units were then determined at a landscape level.

No ecological services considered to be of low or moderately low importance were identified for the system. The majority of the ecological services (46%) provided by the Mogalakwena River floodplain was determined to be of moderately high importance. These services may be attributed to the enhancement of water quality with the removal of phosphates as well as by removing nitrates and toxicants. Owing to the dependence of the local communities on the system, it is likely that there is an important cultural relationship with the community and the system. The floodplain is adjacent to the Waterberg Wilderness Reserve which indicates the importance of this system to provide both tourism and recreational activities such as fishing and birding.

In addition to the above mentioned services, there is an important opportunity to conduct further research into the system, especially considering the ecological significance of the Nylsvlei Ramsar site in the upper catchment areas and the relationship between the two systems.

The dependence of the local communities on the system is indicated by the high importance of selected services. These services pertain largely to water supply and food resources. The water of the floodplain is used for drinking, cooking, cleaning and watering of plantations. In addition to this, the system is also fished by locals for food. An additional service identified to be of a high importance and not directly beneficial to the local communities is the maintenance of biodiversity. This is further supported with the location of the Waterberg Wilderness Reserve on the periphery of the system.

The current land-uses have impacted on the functioning of this system. Local agricultural practices, pertaining predominantly to livestock, have impacted on the ability of this system to provide important services. Agricultural activities have altered the natural hydrology of the system. The decrease in surface roughness due to overgrazing has resulted in a potentially destructive hydrological regime for the system. In addition to this, livestock also impact directly on the quality of water as a result of nutrient input and trampling of the system. Owing to the fact that agricultural practices are on-going for the Platreef Project area, coupled by the absence of mitigatory measures for the current land-uses, it is assumed that the ability of the units to provide important ecological services will continue to deteriorate. The severity of the current identified impacts was, however, determined to be minor at this stage.

20.5.6 Surface Water Investigation

The Platreef Project area falls in the Limpopo Water Management Area (WMA). The two quaternary catchments in which the Platreef Project falls are A61F and A61G. A61F is drained by the Rooisloot River and A61G by the Mogalakwena River. The Nyl River is the headwaters of the Mogalakwena River. The Nyl River flows in a north easterly direction from Modimolle located in the headwaters of the Nyl River, towards Mokopane. At Mokopane, the Nyl River becomes the Mogalakwena River and turns to flow in a north westerly direction passed Mokopane and the Project area. The Mogalakwena River flows to the west of the Project area and ultimately flows into the Limpopo River. The Mogalakwena River is characterised by the presence of vleis and wetlands along its drainage course on both the Turfspruit 241 KR and Macalacaskop 243 KR farms. The Sterk River is a major tributary of the Mogalakwena River and joins the Mogalakwena River from the west some 30 km below the Project area. The Doorndraai Dam is located on the Sterk River. The Doorndraai Dam is the main water supply dam for Mokopane.

There are four main water courses that drain across or adjacent to the Project area. The Dithokeng, Ngwaditse, Rooisloot and the Dorps Rivers flow in a westerly direction across the Project area into the Mogalakwena River. The Dithokeng stream crosses the corner of the mine property in the north before joining the Mogalakwena River. A dam has been constructed on this stream upstream of the town to the north-east of Turfspruit 241 KR farm. The dam is used for domestic water supply.

20.5.6.1 Flow and Water Quality

There is limited flow information available for the Project. There is a DWS flow gauging station A6H033 located on the Nyl River upstream of Mokopane that has been measuring flow since December 1990 and there was a DWS flow gauging station A6H032 located in the Dorps River that measured flow between 1978 and 1980.

Routine monthly surface water flow and quality monitoring commenced in September 2011. Surface water monitoring will form part of the monthly surface and groundwater monitoring programme which will be operated by the mine.

The discharge was measured on the Dorps River downstream monitoring site using an OTT flow meter. As the site visit occurred during the dry season, the Dithokeng, Rooisloot and Mogalakwena streams were not flowing. The Dithokeng was dry while there were stagnant pools of water on the Rooisloot and Mogalakwena Rivers. As a result, the flow measurements at these sites were not performed. During the monitoring period it was found that most rivers were dry with the exception of the Dorps River and twice the Rooisloot. As a result, the flow information is not conclusive enough to make any flow predictions from the data.

20.5.6.2 DWS Water Quality Database

Water quality data was obtained from the DWA WMS database (DWA, 2011).

Results indicate that the upstream water quality sometimes exceeds the standards for pH, Sodium, Fluoride and Ammonium. Within the Project area, pH, Fluoride and Ammonium were sometimes measured above the limit. At the downstream monitoring sites values that exceeded the standard were recorded for Sodium and Fluoride.

20.5.6.3 Water Quality

The first round of water quality sampling took place on 26 September 2009 and the monthly water quality monitoring programme was setup and started from 9 December 2011 until 14 May 2013. The measured concentrations are compared to the SANS 241 (class 1) drinking water standards.

All rivers in the area show high concentrations of iron, manganese and aluminium. This suggests that there is some geological influence for the high concentrations of these metals in the area. The Total Dissolved Solids (TDS) are not of desirable standards for drinking water, while most sites remain under the limit of 1 000 mg/l. The ideal limit for drinking water is 450 mg/l which, with the exception of Dithokeng Upstream and Rooisloot Upstream, most sites fail to meet. The Rooisloot River shows low concentrations of nitrates upstream of the town of Madiba but high concentrations of nitrates downstream of the town. This could be due to leaking sewers in the town and animals defecating in the rivers. The Dorps River shows a eutrophic system enriched with nutrients due to sewage effluent coming from the sewage treatment plant and urban runoff. The Mogalakwena River shows high levels of chloride and high conductivity readings. The chloride in the river could be due to anthropogenic sources and the conductivity readings could be due to the high metal contents in the rivers in the area. The conductivity could also be coming from groundwater sources feeding the river.

20.5.7 Groundwater Investigation

The hydrogeology of the Project area was sourced from the published hydrogeological map series at a scale of 1:500,000, compiled by DWS during 1996 to 2003. The geology has been grouped together based on their general water bearing properties, using a simplified lithological description.

Two main aquifer types are present, i.e. primary and secondary. The Turfspruit 241 KR and Macalacaskop 243 KR farms are mainly underlain by intergranular and fractured aquifers, associated with the RSL. On the Rietfontein 2 KS farm, secondary aquifers are associated with formations of the Transvaal Sequence and basement granite.

20.5.7.1 Hydrogeological Units

Rooisloot Alluvial Aquifer

The primary aquifer is mostly restricted to the alluvium in the Mogalakwena River. Groundwater resources have been developed provided the clay component of the alluvium is negligible. Alluvial thicknesses of up to 20 metres occur and borehole yields in excess of 10 l/s have been established. Minor alluvium occurrences are associated with the Rooisloot River drainage.

In the south western portion of the Turfspruit 241 KR farm and the adjacent Blinkwater farm the alluvium is underlain by shallow (<45 m deep) high yielding secondary bedrock aquifers. The combined primary and secondary aquifers in this area are known as the Rooisloot Alluvial Aquifer.

Boreholes in the Rooisloot Alluvial Aquifer are drilled to depths between 35 and 45 metres. Water levels as shallow as 2 m are present. Calculated aquifer transmissivity values range between 315 and 400 m2/day. The aquifer storage coefficient (S) for both the alluvial and weathered bedrock aquifer is 2.7 x 10-3.

Rustenburg Layered Suite – Main Secondary Aquifer

The Turfspruit 241 KR and Macalacaskop 243 KR farms are mostly underlain by weathered and fractured aquifers, associated with the Rustenburg Layered Suite. The main secondary aquifer occurs at a shallow depth of less than 45 m. Several high yielding boreholes (5 to 10 l/s) have been drilled with the main water interceptions in the fractured bedrock below the weathered zone and at contact zones with intrusive dykes.

Water level depths vary from 3 to 25 mbgl. Water strike depths in the weathered bedrock range from 12 to 20 mbgl, with strike yields between 0.1 to 1.0 l/s. Water interceptions in the shallow fractured bedrock occur at 20 to 42 mbgl with strike yields between 1.0 to 10.0 l/s.

Calculated aquifer transmissivity values range between 17 and 113 m2/day. The aquifer storativity (S) is in the order of 5 x 10-3. The average saturated thickness of the main aquifer zone is 17.6 m. The base of the main aquifer zones is shallow and varies from 12 to 42 mbgl.

Seasonal water level fluctuations due to direct rainfall recharge are expected. Groundwater flow is mainly lateral following topography. Intrusive dykes may act as boundaries to lateral groundwater flow.

Rustenburg Layered Suite – Minor Fractured Aquifer

A minor fractured aquifer was intersected at depth (>45 mbgl) with strike depths varying from 45 to 156 mbgl and yields between 0.1 and 0.2 l/s. Slug testing of six deep core holes indicate very low hydraulic conductivities, between 1 x 10-4 m/d and 1 x 10-5 m/d, considered representative of the igneous rock matrix. Inspection of core samples indicate minor fracturing at the mineralised contact zone at a depth of some 800 m.

Water level depths of the deep fractured aquifer are currently similar to that of the main secondary aquifer. Artesian conditions are observed during nearby core drilling operations, which stop once drilling is discontinued, and are considered the result of the drilling process.

Banded Ironstone Formation (BIF)

The BIF of the Penge Formation outcrops as a prominent SE to NW striking topographic ridge, dipping to the south-west. One borehole was drilled in proximity of a fault zone to intercept the BIF at shallow depths. The borehole was drilled into weathered and fractured BIF to a depth of 37 m.

The static water level is 7 mbgl and water strikes were encountered from 14 to 21 mbgl, with a significant final airlift yield of 12 l/s. An aquifer transmissivity of 180m2/day was determined from borehole test pumping.

Steeply Dipping Dolomite Formation

Karst development in the steeply dipping and elevated dolomite formations is very limited to absent. This conclusion is based on gravity surveys conducted showing very limited gravity low anomalies indicating the general absence of leached dolomite formations.

Two boreholes were drilled to depths of 98 and 150 mbgl with no water strikes encountered below 29 mbgl. Very low yields of 0.05 to 0.1 l/s were encountered at shallow depths (<29 mbgl) associated with bedding plane contact zones within the dolomite formation. One borehole intersected schist, quartzite, dolomite and granite at depths between 45 and 120mbgl with no water strikes encountered.

Water level depths vary from 10 to 17 mbgl. An aquifer transmissivity of 1 m2/day was determined from test pumping.

Turfloop Granite

The granite on the Rietfontein 2 KS farm is presented by the Turfloop granite, comprising fine to medium grained grey and pink biotite granite. The topography of the areas drilled is elevated, indicating shallow bedrock conditions.

Eleven boreholes were drilled to depths to 120 m to investigate the hydraulic characteristics of the granite aquifer. A very low water strike (0.05 l/s) was intersected in one borehole at a depth of 33.5 m, with the other 10 boreholes “dry” (no blow test yield). One borehole (2012) intersected schist, quartzite, dolomite and granite at depths between 45 and 120 mbgl with no water strikes encountered.

Falling head permeability tests confirms an insignificant aquifer transmissivity range of «0.5 m2/day.

The water level varies between 20 to 23 mbgl. It has been observed that the water level rise in some of the older exploration boreholes has taken several months to reach the local water table elevation and thus indicates the extremely low hydraulic characteristics of the Turfloop Granite.

20.5.7.2 Hydrocensus

The hydrocensus survey identified 216 borehole sites within the Project area, which includes data from the desk study, recognisance hydrocensus survey and borehole drilling undertaken in early 2011. A total of 198 GRIP database (government owned) borehole sites were searched for of which 147 were located in the field. Boreholes surveyed with no existing allocated numbers were assumed to be privately owned and allocated an electronic database number (not marked in the field).

During the hydrocensus groundwater samples were collected from 81 boreholes, which included 43 private boreholes. Samples were taken from equipped boreholes within and in close proximity to the three farm boundaries. The samples were submitted to UIS Analytical Services in Centurion for analysis.

Privately owned boreholes are in use in all the local communities located in the study area. 34 boreholes sampled in the communities are equipped with submersible pumps and the remaining 9 with hand pumps. The potability of the water in 23 boreholes (~50%) was found not to be suitable for human consumption based on the nitrate NO_3 –N exceeding 11 mg/l (WRC Guideline, 1998). Thirteen boreholes have nitrate (N) values ranging from 40–129 mg/l, with a high health risk if consumed.

20.5.7.3 Ground Geophysical Surveys

Ground geophysical surveys were used to identify zones of deeper weathering, fracturing and possible leaching in the underlying bedrock, comprising mainly of norite, gabbro, BIF, dolomite and granite. Four different geophysical methods (magnetic, electromagnetic, resistivity imaging and gravity) were employed in view of the varying geological setting within the study area. Different combinations of geophysical techniques were used due to varying physical weathering properties of the rock types.

The geophysical surveys were conducted during October 2011 and extended in July-October 2013 on the Turfspruit 241 KR, Rietfontein 2 KS, and Bultongfontein 239 KR farms. The main objective was to investigate for zones of deep weathering and fracturing associated with geological structures which could act as preferential groundwater flow paths, and to assist in selecting positions for the drilling of the monitoring boreholes.

20.5.7.4 Groundwater Use

At the Macalacaskop 243 KR, Turfspruit 241 KR, and Rietfontein 2 KS farms, water is being abstracted from groundwater sources to supply the various rural communities. Dispersed boreholes are in use throughout the area, with the highest volume abstracted for domestic water supply from the Rooisloot Alluvial Aquifer in the south-western part of the Turfspruit 241 KR farm.

20.5.7.5 Groundwater Levels and Flow Direction

Groundwater levels measured were recorded in private, government and both exploration core holes and groundwater boreholes. Water levels from the local DWS monitoring network recorded during the course of this investigation from late 2010, as well as levels measured during the drilling and testing of boreholes undertaken in 2012, were collated.

Borehole elevations were determined for boreholes with verified coordinates (accuracy <10 m) using the DEM data from Platreef with altitude accuracies <0.25 mamsl. A total of 61 water levels depths and altitudes were compiled using the latest available water level data for the period November 2010 to August 2013. The Project area groundwater level monitoring program consists of 30 monitoring sites of which 15 are equipped with electronic data logging devices set to record/store water level measurements on mainly six hourly intervals and in two instances hourly.

Water levels were measured in the Project area since March 2012 and represent baseline reference water levels for future monitoring. Overall water level depths range from 3 to 36 mbgl. Since the groundwater monitoring programme has been initiated in the PSA, water level information on a time series principle is now available. The piezometric map portrays the March 2012 status which has not changed significantly in terms of its regional context. The water level difference between the March 2012 and April 2013 is small (<1.0 m for 29 monitoring sites).

Groundwater flow follows surface drainage with flow occurring from north-east to south-west (at right angles to the RLS succession) and eventually north-west following the Mogalakwena River. Groundwater elevations are highest (1,220 masl) on the Rietfontein 2 KS farm underlain by granite and lowest (1,030 masl) on the Turfspruit 241 KR farm associated with the Rooisloot Alluvial Aquifer. This represents a hydraulic head of 190 m across the Platreef Project area. Hydraulic gradients for the main hydrogeological units are:

- Turfloop Granite : 0.03
- Steeply Dipping Dolomite : 0.08
- Banded Ironstone Formation : 0.014
- Rustenburg Layered Suite : 0.016 to 0.02
- Rooisloot Alluvial Aquifer : 0.0025

Water Level Trends in the Platreef Project Area: 2012-2013

The water level trends report on the current water use scenarios in the surrounding community area which is mostly abstractions for domestic and stock watering. The differences between the water levels for the period March 2012 to August 2013 indicate variations within ~1m with the exception of two larger water table fluctuations. The one borehole is situated east of the community of Tshamahansi and could be impacted by local water abstractions. The second is approximately 1km south of the proposed Rietfontein TSF site. This significant water table rise is due to the extremely low hydraulic characteristics of the Turfloop granite and represents the actual water table recovery after drilling was completed to balance with the regional piezometric elevation.

The water level behaviour in the Platreef Project area does not indicate long-term positive (recharge) or negative (recession) type trends and remains a healthy balance between local abstractions and annual recharge events although not in perfect harmony with each other.

Regional Water Level Monitoring by Department of Water and Sanitation

Two current DWS regional water level monitoring stations are present within the Platreef Project area during the hydrocensus survey. The monitoring boreholes are equipped with automatic water level loggers which record water levels every hour since October 2005 to date.

Water levels in M03-3539 (Rooisloot Alluvial aquifer) show a steady recovery in water levels since 2006 of 5 m which have recovered from 18.3–13.3 m. The increase in water levels for this six year period represents a net recharge of the aquifer from indirect recharge from the Mogalakwena River system, direct recharge from rainfall and/or reduced groundwater abstraction. The water level curve indicates the presence of annual cyclic recharge, with no or limited annual discharge. The time series data indicates that water levels were lowered in excess of 13 m from 1985 to mid-1996 (11-year cycle), due to probable excessive abstraction during a drought period.

20.5.7.6 Groundwater Quality

The groundwater type for the Platreef Project area comprises predominantly of $MgHCO_3$ with mixing water types (no dominant ions). This is due to the geological mineralisation of the igneous rocks (RLS) which covers most of the Platreef Project area. Impacted water with elevated chloride and nitrate content are found either within the community settlements or downstream of the communities. Operational high yielding boreholes have elevated nitrate content due to high abstraction rates that draw pollutants from the surrounding or nearby community.

20.5.8 Fauna and Flora

The Platreef Project is situated within the Savanna biome, which is the largest biome in Southern Africa. It consists of a grassy ground layer and a woody plant upper layer. It is known as Shrubveld when the woody layer is close to the grass layer and as Bushveld in any intermediate phases. Factors that delimit this biome include sufficient rainfall, fires and grazing of animals (SANBI, 2011).

Field investigations were conducted during the dry season (June 2011) and during the wet season (September 2011). A second dry season survey was commissioned during August 2013, during which specific infrastructure placements were investigated. The findings and recommendations of these investigations are detailed below.

20.5.8.1 Flora

Four vegetation types were found to occur within the Platreef Project area, these include:

- Makhado Sweet Bushveld (Vulnerable) (Mucina & Rutherford, 2006).
- Mamabolo Mountain Bushveld (Least Threatened, because statutorily conserved in Witvinger Nature Reserve) (Mucina & Rutherford, 2006).

- Polokwane Plateau Bushveld (Least Threatened) (Mucina & Rutherford, 2006).
- Waterberg Mountain Bushveld (Least Threatened) (Mucina & Rutherford, 2006).

Vegetation Communities

During the field survey the vegetation was found to predominantly bushveld, however residential and farming regions allowed for secondary succession and the growth of pioneer species due to the disturbances exerted. One hundred and thirty five (135) species were identified throughout the Platreef Project area. Six vegetation communities were identified including:

- Ridge Bushveld.
- Impacted Ridge Bushveld.
- Degraded Mixed Bushveld.
- Secondary Grassland and Agricultural fields.
- Wetland vegetation.
- Residential areas.

Alien and Invasive Species

Alien invasion for the Platreef Project area was not regarded as severe and is not regarded as a major hindrance to biodiversity.

Alien species in South Africa are categorised according to the Conservation of Agriculture Resources Act, 1983 (Act No. 43 of 1983) (CARA) and the National Environmental Management: Biodiversity Act (NEM:BA).

Declared alien and invasive species have been divided according to CARA into three categories:

- Category 1: Declared weeds that are prohibited on any land or water surface in South Africa. These species must be controlled, or eradicated where possible;
- Category 2: Declared invader species that are only allowed in demarcated areas under controlled conditions and prohibited within 30 m of the 1:50 year flood line of any watercourse or wetland; and
- Category 3: Declared invader species that may remain, but must be prevented from spreading. No further planting of these species are allowed.

In addition, the NEM:BA Regulations (GN R. 506, GN R. 507, and GN R. 508 in Government Gazette 36683 of 17 July 2013) was issued on the 17th of July 2013. These regulations were used for categorising alien plant species found on site in this study. The NEM:BA categories for invasive species according to Section 21 are as follows:

- Category 1a: Species requiring compulsory control.
- Category 1b: Invasive species controlled by an invasive species management programme.
- Category 2: Invasive species controlled by area.
- Category 3: Invasive species controlled by activity.

Certain species have different alien invasive categories for different provinces in South Africa.

Grazing Intensity/Land-use

The Platreef Project is broadly managed either privately and fenced or by the community and not fenced. The majority of the site is currently not fenced and communally utilised for grazing. Large agricultural fields also exist. The communal grazing areas are severely overgrazed with the subsequent bush encroachment the result.

20.5.8.2 Fauna

Fauna expected to occur on site include assemblages within terrestrial and wetland ecosystems: mammals, birds, reptiles, amphibians and invertebrates. Each of these assemblages occurs within unique habitats; the ecological state of these habitats directly relates to the number of species found within them. The main habitats occurring in the Platreef Project area are bushveld plains and pans with little altitudinal variation.

Mammals

For a desktop review of mammals that could possibly occur within the Platreef Project area, South African National Biodiversity Institution's (SANBI) Biodiversity Information System (SIBIS) was used. The list shows all animal species that were previously recorded within the Limpopo Province and the Platreef Project area. The list also indicated the International Union for the Conservation of Nature (IUCN) status, as well as the NEM:BA status. By making a comparison between the previously recorded species list and the currently occurring species found during the field survey, the magnitude of impacts resulting in species reduction or loss can be estimated

The probability of occurrence was estimated based on habitat requirement and distribution. Protected species of Limpopo Province under Schedule 3 were also considered. Amongst these listed; the Leopard, Honey Badger, Hedgehog, Bat-eared fox and Civet were identified to have a high probability of occurrence within the Platreef Project area.

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Avifauna

Birds have been viewed as good ecological indicators, since their presence or absence tends to represent conditions pertaining to the proper functioning of an ecosystem. Bird communities and ecological condition are linked to land cover. As the land cover of an area changes, so do the types of birds in that area (The Bird Community Index, 2007). Land cover is directly linked to habitats within the Platreef Project area. The diversity of these habitats should give rise to many different species. According to the South African Bird Atlas Project (SABAP2), almost 300 species of birds have been identified in the area; the majority of these birds are comprised of bushveld species.

The Yellow-Billed Stork and African Spoonbill are protected by the Agreement on the Conservation of African-Eurasian Migratory Waterbirds (AEWA). The AEWA covers 255 species of birds ecologically dependent on wetlands for at least part of their annual cycle, including many species of divers, grebes, pelicans, cormorants, herons, storks, rails, ibises, spoonbills, flamingos, ducks, swans, geese, cranes, waders, gulls and terns. This conservation agreement includes issues such as species and habitat conservation, management of human activities, research and monitoring, education and information, and implementation.

Red Data bird species protected within the Limpopo Province was also considered during the field survey. The possibility of occurrence was based on the distribution and habitat requirements of these Red Data species. The Yellow-billed Stork is also included in this list and has a Near Threatened status (Barnes, 2000). The probability of occurrence is high for aquatic birds, due to the fact that the wetland to the south-west of the site forms part of the Nylsvlei Ramsar system and fulfils the habitat requirements of these species.

Herpetofauna

No Red Data status reptiles were found during the field surveys. The probability of occurrence was determined based on the distribution and habitat requirements.

20.5.8.3 Species of Special Concern

Flora

No Red data species considered identified by the Pretoria Computerised Information System (PRECIS) data for the grid squares were identified during the field survey, however the Limpopo Environmental Management Act (Act 7, 2003) and the National Forests Act (Act 84, 1998) was also taken into consideration. Under the National Forest Act three protected species were found on site, including:

- Combretum imberbe (Leadwood).
- Boscia albitrunca (Shepherds tree).
- Sclerocarya birrea (Marula).

The protected tree species Combretum imberbe was encountered on the lower lying flat areas within regular intervals as this tree species is not removed when agricultural fields are made.

Plant Species with Ethnobotanical Uses

Ethnobotany is a branch of botany that places focus on the use of plants for medicines and other practical purposes. The use of native plants for ethnobotanical uses can be detrimental to populations that are overexploited.

From the list of plant species identified during the field surveys there are 53 species that have cultural uses. Medicinal plants are important to many people and have been used traditionally for centuries to cure many ailments. Plants have also been used traditionally for other cultural uses, such as building material, and for spiritual uses such as charms.

20.5.8.4 Ecological Sensitivity Assessment

Protected Area: Witvinger National Reserve

The Witvinger National Reserve has an IUCN status listed as Category IV Protected Area. This means that management of the area is performed to ensure the maintenance of habitats and meet the requirements of certain species. The Nature reserve supports high levels of biodiversity which is endemic to the area and therefore extremely important to conserve. Ridges link the Platreef Project area to this reserve.

Protected Area: Nylsvlei Nature Reserve

The wetland area on site is part of the Nylsvlei floodplain which is one of South Africa's least impacted floodplain systems. Part of the system is conserved and is recognised as a Provincial Reserve; the Nylsvlei Nature Reserve. The reserve has statutory protection and is also recognised as a Ramsar Site. Ramsar recognition indicates the wetland to be of international importance for waterfowl.

The wetland area forms the western boundary of the site. A steep ridge area lies on the other side of the wetland; another area of high sensitivity which falls in the Waterberg Wilderness Reserve.

Protected Area: Waterberg Wilderness Reserve

The private reserve has national conservation protection status as a result of it supporting high levels of biodiversity. This reserve is important for populations of tree species such as Protea, Acacia, Combretum and Searsia that readily occur here. It also includes many protected mammal species such as Leopard (Panthera pardus), Serval (Leptailurus serval), African Wild Cat (Felis silvestris), Brown Hyaena (Parahyaena brunnea), Aardwolf (Proteles cristatus), Honey Badger (Mellivora capensis) and African Civet (Civettictis civetta)

Important Bird Areas

The Platreef Project area does not fall within any important bird areas as listed in the national or provincial biodiversity guidelines.

Nationally Threatened Ecosystems

The list of national Threatened Ecosystems has been gazetted (NEM:BA: national list of ecosystems that are threatened and in need of protection) and result in several implications in terms of development within these areas. The list was referenced in order to ascertain the level of ecosystem threat of the ecosystems present within the Platreef Project area.

The Platreef Project area does not fall within any demarcated National Threatened Ecosystems. The closest National Threatened Ecosystem is the Springbokvlakte thornveld which is located approximately 20 km north from the Platreef Project area.

National Protected Areas Expansion Strategy (NPAES)

The NPAES are areas designated for future incorporation into existing protected areas (both National and Informal protected areas). These areas are large, mostly intact areas required to meet biodiversity targets, and suitable for protection. They may not necessarily be proclaimed as protected areas in the future and are a broad scale planning tool allowing for better development and conservation planning.

20.5.9 Air Quality

Baseline monitoring and assessment of atmospheric pollutants was appraised using data from the South African Air Quality Information System (SAAQIS) database. The Air Quality Monitoring Station in Mokopane is one of the three air quality monitoring stations commissioned for the Waterberg-Bojanala Priority Area (WBPA) in 2012. The other two stations are in Lephalale and Thabazimbi. The National Priority Area covers the Bojanala District in the North West Province and the Waterberg District in the Limpopo Province. The database contains measurement for known priority pollutants, recording data based on the recommended averaging period. Archived measurements for the past five months are discussed below to emphasize the background conditions.

20.5.9.1 Measured Concentrations

PM10 Concentration

In literature, particulate matter (specifically PM2.5 and PM10) represents danger to the receiving population as it can penetrate into indoor environment increasing the exposure period to such pollutants. The PMs have the ability to penetrate the trachea-bronchial and alveolar regions of the human respiratory system leading to respiratory diseases. If the PM contains heavy metals, the risk to human is exacerbated based on the exposure period, age and wellbeing of the individual.

The average measured background PM10 concentration (recorded at the Mokopane Ambient Air Quality Station) for the last five months is generally within the current National Ambient Air Quality Standards (NAAQS) for PM10 of 120 µg/m³. The limit was exceeded once in December 2012. If the future NAAQS ambient standard of 75 µg/m³ is considered (which will come into effect on the 1st of January 2015), there are several days exceeding the limit value. However, the World Health Organisation (WHO) Guideline of 50 µg/m³ is actually exceeded on a number of occasions.

Measurements conducted in 2003/2004 by WSP Walmsley confirm historical levels of ambient particulate matter in the Platreef Project area. The PM2.5 and PM10 levels were measured using Single Striker Filter Units (SSFU) installed at Mahwelereng to continuously monitor ambient concentrations of pollutants. The PM10 concentrations from April 2003 to April 2004 show five of the twelve months of sampling exceeding the South African Standard of 120 µg/m³. If reference is made to the IFC (WHO Guideline), only one of the twelve months of sampling was within compliance (March 2004).

PM2.5 Concentrations

The NAAQS for particulate matter with aerodynamic diameter less than 2.5 microns (PM2.5) was promulgated on 29 June 2012 by the Minister of Environmental Affairs. Ever since, ambient PM2.5 standard of 65 µg/m³ is in force until 31 December 2015, and a new standard of 40 µg/m³ would take effect from 1 January 2016.

The PM2.5 concentration observed over the five months’ period is observed to be well below the IFC (WHO Guideline) value of 25 µg/m3 and the NAAQS of 65 µg/m3 (which takes immediate effect from the date of promulgation).

If the results obtained from the WSP Walmsley monitoring (2003/2004) are compared to the current standard, two months (August and September) of the twelve months of sampling were not compliant. The values measured were exceeding the 65 µg/m³ recommended by the current standard. On the other hand, nine months recorded values in excess of the IFC (WHO Guideline) of 25 µg/m3.

Carbon Monoxide

The Carbon Monoxide (CO) concentration measured is below the recommended NAAQS 8 hr and 1 hr limit values of 8.7 parts per million (ppm) and 26 ppm respectively. The peaks observed in CO concentration are not in exceedance of the standard. The pollutant is known to contribute to greenhouse effect and global warming.

NO2/NOx Concentration

The NO2 standard specified by the WHO and South African NAAQS are the same - 200 µg/m3 (106 ppm). It is assumed that the complete conversion of all emitted NO to NO2 has occurred, as per US EPA's Guideline on Air Quality Models, 40 CFR Part 51, for Tier 1 screening approach. The recorded values for NOx are generally below this limit, except an incident in October 2012 with a peak that exceeded the limit slightly.

Sulphur Dioxide

The Sulphur Dioxide (SO2) concentration observed over the five months' period is seen to be very low with values generally below 10 ppm, a factor of 4 below the prescribed SA 24 hour limit of 48 ppb. The values are also within WHO recommended guideline value of 20 ppb.

Measurements observed over a 10 minutes averaging period were within SA NAAQS value of 191 ppb, except on one occasion when the recommended limit value was violated, but below the WHO guideline value of 500 ppb.

Benzene

From the measurements, there are a number of times when the NAAQS limit of 3.2 ppb was exceeded slightly. Once in September 2012, ambient concentration went above 16 ppb.

Ozone

Ozone (O3) is formed in the atmosphere by the reaction of nitrogen oxides, hydrocarbons and sunlight. Ozone levels were within the recommended South African standard of 61 ppb (120 µg/m3), with some exceedances observed during the first few days of February 2012. If WHO guideline value is considered, several exceedances occurred.

20.5.9.2 Measured Dust Fallout Levels

A dust fallout monitoring network was commissioned in August 2013 to monitor the ambient dust deposition rates in the Platreef Project area. The network was commissioned at the selected sensitive receptor areas around the proposed mine area. Dust fall out monitoring are conducted over 30 day periods.

During the monitoring window, exposure is expected to comply with the standard operating procedure of ~2 days. If monthly dust fallout rates measured in 2003/2004 are compared to the current dust fallout limits spelt out in the National Dust Control Regulation (GN R. 827 published in Government Gazette 36974 of 1 November 2013), the area will likely be in violation of the residential and industrial limits 92% and 67% of the sampling period. Although the permitted frequency of exceedance is twice within a year (not sequential months), the area recorded seven consecutive months of exceedance, with dust deposition rates well over 1 500 mg/m2/day. The above background results from historical data are considered a serious violation of the current standard.

The seasonal average dust deposition rates per site confirm the variability from season to season in the area. In autumn, all the sites (seven in total) exceeded the residential and industrial limit values of 600 mg/m2/day and 1 200 mg/m2/day – with the highest value reaching 2 760 mg/m2/day (site – Ga-Madiba). In winter, only one site was within compliance – as the other sites exceeded residential and industrial limit values. The highest value was observed to be above 3 350 mg/m2/day in winter. In the spring, majority of the sites were in violation of the residential and industrial standards as in previous seasons. Lastly, the values recorded in summer were within the residential and industrial limits, except at the site Moholerwe with dust deposition rates of 740 mg/m2/day.

20.5.9.3 Dispersion Model

Dispersion models are used to predict the ambient concentration in the air of pollutants emitted to the atmosphere from a variety of processes (SANS 1929:2011). Dispersion models compute ambient concentrations as a function of source configurations, emission strengths and meteorological characteristics, thus providing a useful tool to ascertain the spatial and temporal patterns in the ground level concentrations arising from the emissions of various sources.

All emission scenarios have been simulated using the USA Environmental Protection Agency's Preferred/Recommended Models: AERMOD modelling system (as of December 9, 2006, AERMOD is fully promulgated as a replacement to ISC3 model).

For the Platreef Project, the RoM stockpile, waste rock and TSF stockpile sources have been modelled as area sources. Crusher, material handling processes (tipping to RoM stockpile, tipping to waste rock stockpile, conveyor to crusher) have been modelled as volume sources. The ventilation shaft was modelled as a point source. The paved road in the mine project areas and the link road to N11 were modelled as line volume sources.

Simulations were undertaken to determine concentrations of particulate matter with a particle size of less than 10 microns (µm) in size (PM10), particle size of less than 2.5 microns (µm) in size (PM2.5), and of deposition of Total Suspended Particulates (TSP) from operations at the proposed Platreef Platinum Mine. Scenarios with mitigation measures were simulated using control factors.

Isopleths of PM10 generated from the dispersion model for both unmitigated and mitigated scenarios have shown that concentrations above the recommended limit value can reach distances of ~2 km from the mine boundaries, especially for the western and southern section of the mine boundary. With mitigation measures applied, the ground level concentrations of PM10 observed at the selected sensitive receptors showed decreases ranging between 33% and 59%. Annual PM10 levels were observed to have decreased by between 25% and 47%.

Isopleths from the dispersion modelling plots have indicated that the area impacted by PM2.5 arising from the proposed Platreef mine operation is minimal and greater portion falls within the Platreef Project area. For PM2.5, with mitigation the diurnal concentrations observed at the defined sensitive receptors decreased by between 22% and 83%. The decreases observed for the annual levels ranged between 33% and 54% respectively.

The predicted dust deposition rates before mitigation at the sensitive receptors were all within the SANS limit for residential areas (i.e. 600 mg/m²/day) except for site PLA 06. When mitigation measures are applied, the anticipated deposition decreased at the selected sensitive receptors i.e. PLA 06 – from 629 mg/m2/day to 317 mg/m2/day. In general, the levels decreased by between 35% and 50% at the selected sensitive receptor sites.

20.5.10 Noise Assessment

20.5.10.1 Noise Measurement Locations

A baseline assessment was undertaken to determine the current ambient noise levels in areas surrounding the Platreef Project area. The criteria that were used for the siting of the measurement locations were:

- The locations were the nearest noise sensitive receptors surrounding the Platreef Project and subsequently the most likely to be impacted on by the proposed mining activities; and
- The locations serve as suitable reference points for the measurement of ambient sound levels surrounding the Platreef Project area. The noise measurement locations cover the surrounding communities that represent a comprehensive soundscape of the area.

The list of noise measurement locations can be seen Table 20.2. A Cirrus, Optimus Green, and precision integrating sound level meter was used for the measurements. The instrument was field calibrated with a Cirrus sound level calibrator.

Table 20.2 Noise Measurement Locations

ID	Receptor	Receptor Type	GPS Coordinates
Plat 1	Masodi	Suburban community with little road traffic	24° 7'41.31"S 28°57'19.21"E
Plat 2	Madiba	Suburban community with little road traffic	24° 7'48.53"S 28°58'49.29"E
Plat 3	Ga-Kgobudi	Suburban community with little road traffic	24° 5'20.37"S 28°56'47.33"E
Plat 4	Magongoa	Suburban community with little road traffic	24° 4'26.58"S 28°57'58.14"E
Plat 5	Tshamahansi	Suburban community with little road traffic	24° 5'3.44"S 28°58'22.78"E
Plat 6	Molekana	Suburban community with little road traffic	23°59'29.35"S 28°57'22.32"E

20.5.10.2 Baseline Results and Discussions

The results from the noise meter recordings for all the sampled points as well as the rating limits according to the SANS 10103:2008 guidelines are presented in Table 20.3 below.

Table 20.3 Baseline Noise Measurements Results

Sample ID	SANS Rating Limit			Measurement Details		
	Type of district	Period	Acceptable Rating Level dBA	$L_{Areq,T}$ dBA	Maximum/Minimum dBA	Date
Plat 1	Suburban	Daytime	50	48	75/36	07/11/2011
		Night time	40	44	65/41	07/11/2011
Plat 2	Suburban	Daytime	50	47	75/33	08/11/2011
		Night time	40	43	66/27	08/11/2011
Plat 3	Suburban	Daytime	50	46	73/32	09/11/2011
		Night time	40	45	64/36	09/11/2011
Plat 4	Suburban	Daytime	50	51	79/33	10/11/2011
		Night time	40	52	70/30	10/11/2011
Plat 5	Suburban	Daytime	50	47	71/37	11/11/2011
		Night time	40	45	68/33	11/11/2011
Plat 6	Suburban	Daytime	50	28	80/20	22/08/2013
		Night time	40	27	65/20	22/08/2013
	Indicates $L_{Aeq,T}$ levels above either the daytime rating limit or the night time rating limit					

The noise sources that were influencing the baseline measurements at the time of the noise survey and that were responsible for the day time and night time measurements are summarised in Table 20.4.

Table 20.4 Summary of Noise Sources Influencing Baseline Measurements Around the Proposed Site

ID	Day	Duration	Night	Duration
Plat 1	Vehicular traffic (did not cause noise levels to measure above SANS guideline)	Continuous	Gryllidae (crickets)	Continuous
Plat 2	Vehicular traffic (did not cause noise levels to measure above SANS guideline)	Intermittent	Gryllidae (crickets)	Continuous
Plat 3	Vehicular traffic (did not cause noise levels to measure above SANS guideline)	Intermittent	Gryllidae (crickets)	Continuous
Plat 4	Livestock (roosters crowing)	Intermittent	Gryllidae (crickets)	Continuous
Plat 5	Vehicular traffic (did not cause noise levels to measure above SANS guideline)	Continuous	Gryllidae (crickets)	Continuous
Plat 6	Vehicular traffic (did not cause noise levels to measure above SANS guideline)	Intermittent	Gryllidae (crickets)	Continuous

20.5.10.3 Predictive Modelling

Predictive modelling was performed for the proposed mining activities through the use of the modelling software SoundPlan. The software specializes in computer simulations of noise pollution dispersion. Estimates of the cumulative mining noise levels from the study were derived from the noise emissions from all the major noise-generating components and activities of the Platreef Project.

The models were run as a conservative scenario with worst case assumptions, so the following should be noted:

- The average yearly temperature was used.
- The average yearly humidity was used.
- Calm wind conditions were used.
- The mitigation effect of vegetation was not taken into account.

The following table indicates the noise power levels used in the model simulations. The sound power levels were derived from a number of previous studies.

Table 20.5 Sound Power Levels from Main Continuous Noise Cause Sources

Noise Source	**Sound Power Levels dB**						
Octave band frequencies, Hz	63	125	250	500	1000	2000	4000
Construction Phase							
Haul Truck	108	118	115	114	110	106	102
Excavators	113	117	107	108	106	101	95
Front end Loader	108	116	107	108	105	99	95
Drill	109	118	113	113	113	112	110
Dozer	110	122	113	114	110	108	104
Operational Phase							
Processing plant (cumulative including milling operation)	108	106	107	103	99	94	86
Ventilation shafts	117	114	116	110	108	107	104

The blasting noise levels were calculated according to the SANS 10357:2004 - The calculation of sound propagation by the Concawe method. Table 20.6 below represents the power levels used in the calculation.

Table 20.6 Sound Power Levels from Blasting Activities

Noise Source	**Sound Power Levels dB**						
Octave band frequencies, Hz	63	125	250	500	1000	2000	4000
Blasting	124	126	127	125	123	120	117

The noise dispersion modelling software as well as the Concawe method was used to assess whether the noise from the proposed mining activities will impact on the relevant noise sensitive receivers, by comparing the predicted propagating noise levels with the current ambient baseline noise levels.

According to the noise dispersion model for the construction phase, the noise from the construction of either TSF site 2 or site 3 and the plant will not measure above the current ambient daytime noise levels at the surrounding communities respectively. The noise levels from the above mentioned activities will also not measure above the SANS daytime suburban rating limits of 50dBA at any of the surrounding communities.

The blasting propagation was calculated separately because it will occur intermittently compared to the other construction activities. The calculation was performed according to the SANS 10357:2004 - The calculation of sound propagation by the Concawe method. Table 20.7 below represent the noise levels from the blasting at the surrounding communities.

Table 20.7 Blasting Noise Levels at the Surrounding Communities

Community	Baseline Level dBa	Blasting Noise Level dBa			
		Shaft 1	Shaft 2 (vent)	Shaft 3 (vent)	Shaft 4 (vent)
Ga-Magongoa	51	58	60	64	52
Tshamahansi	47	54	54	52	53
Mzombane	48	52	50	54	60
Kgobudi	46	58	55	56	58

The noise dispersion model for the operational phase indicates that the noise from the proposed vent shafts and processing activities is expected not to measure above the current ambient noise levels at the surrounding communities.

20.5.11 Soils, Land Use and Land Capability

20.5.11.1 Land Type Soil Information

Existing land type data was used to obtain generalised soil information and terrain types for the Project area. Land type data exists in the form of published 1:250 000 maps. These maps indicate delineated areas of similar terrain types, pedosystems (uniform terrain and soil pattern) and climate (Land Type Survey Staff, 1989).

The Project area is undulating and is located within the dominant Ae, Ah and Ib land types of the 2328 Pietersburg and 2428 Nylstroom land type maps (Land Type Survey Staff, 1989). These land types indicate that the underlying geology consist mainly of hornfels, shale, quartzite, conglomerate, granite and biotite granite. The Ae land type covers most of the southern part of the Project site while land types Ah and Ib cover the northern part of the Project site.

The Ae land type is flat with slopes of 1% – 5% while the Ah and Ib land types are undulating containing slopes of 5% – 10% and 10% – 100% respectively. The Ib land type is easily recognised as rocky outcrops within the Project area.

Crest landscape positions are indicated as 1, scarp landscape positions as 2, mid slope positions as 3, while foot slope and valley bottom positions are indicated as the 4 and 5 landscape positions respectively.

20.5.11.2 Major Soil Types Occupying the Project Area

The steep crest landscape positions are generally occupied by shallow rocky soil. Lower lying mid slope areas on the old flood plain, are dominated by well drained red and yellow soil such as Hutton, Oakleaf and sandy Clovelly soil types.

Hutton soils consist of an orthic A horizon overlying a red brown B horizon. The Clovelly soil consists of an orthic A horizon overlying a yellow brown B horizon while the Oakleaf soil consists of an orthic A horizon, overlying a neocutanic brown apedal B horizon. The A and B horizons have good internal drainage properties, and therefore well drained.

The lower lying areas in the foot slope and valley bottom positions are dominated by heavy clay soils such as the Valsrivier and Arcadia soil forms. The Valsrivier soil consists of an orthic A horizon overlying a structured pedocutanic B horizon. The Arcadia soil consists of a vertic A horizon.

The Katspruit (Ks) soil is a true wetland soil and is permanently wet. This soil type is found at the lowest landscape positions such as in the valley bottom landscape position. The Ks soil consists of an orthic A horizon overlying a G horizon. The G horizon is characterised high clay content and green and grey colours due to the anaerobic soil conditions caused by waterlogging.

The agricultural potential of the dominant well drained soils, for example Oakleaf and Hutton soils in the surveyed area are determined by the combination of soil depth and favourable climatic conditions. The average rainfall in the area is medium to high (650 mm per annum) and in combination with good soil, results in high arable agricultural potential as indicated in Table 20.8.

Table 20.8 Dominant Cultivated Soil Forms Found in the Project Area During Soil Survey

Soil Form	Average Depth (m)	General Characteristics	Agricultural Potential
Clovelly (Found near stream bed cultivated crop is maize)	1.5	Orthic topsoil A horizon overlying a deep, red, well drained, structureless, B horizon underlain by hard or weathered rock.	Low due to very sandy nature and low soil fertility conditions.
Oakleaf	0.8 – 1.5	Orthic topsoil A horizon overlying a deep, neocutanic, brown, well drained, structured B horizon.	High due to high rainfall in the region well drained status and high water holding capacity of the soil.
Hutton	0.8 – 1.5	Orthic topsoil A horizon overlying a deep, red, well drained, structureless, B horizon underlain by hard or weathered rock.	High due to medium to high rainfall in the region well drained status and high water holding capacity of the soil.
Valsrivier	0.75	Orthic topsoil A horizon overlying a pedocutanic B horizon underlain by unspecified material.	Low due to clayey nature and potential water logging conditions.

20.5.11.3 Land Capability and Land Use

Land capability is determined by a combination of soil, terrain and climatic features. Land capability is defined by the most intensive long term sustainable use of land under rain-fed conditions. Simultaneously an indication is included in the definition about the permanent limitations associated with the different land use classes (Schoeman et al, 2000).

Table 20.9 contains a summary of the land capability classes and present land use of the Project area. The site is dominated by the Ae land type indicating that arable agriculture is potentially possible but used presently for sustainable agriculture, specifically mixed arable and grazing (cattle) but dominated by grazing.

Table 20.9 Summary of Land Capability and Dominating Land Use

Land Type	Dominating Soil Capability Class	Dominating Land Capability Class	Dominating Land Use	Agricultural Potential
Ae224	iii	iii	Housing/Grazing	Arable
Ah28	vi	vi	Housing/Grazing	Grazing
Ib447	viii	viii	Grazing	Wildlife

Fertility

Organic carbon in the topsoil ranges from 0.65% – 1.07%. Generally South African cultivated soils contain an organic carbon content of around 1%. An organic carbon content of 1% is considered to be low but expected for cultivated soil under South African climatic conditions.

The phosphorus status is very low for the Project area. Phosphorus is an important macro nutrient and the phosphorus content with a low record of 0.96 and a high record of 4.6 mg kg-1 is very low and indicative of poor phosphorus soil status. Natural low fertility status is deteriorated even further through loss of phosphate by fixation. Phosphate fixation is a common problem in red soils thereby depleting plant available phosphate.

The soil pH is in the order of 5.8 – 6.2. This pH range is indicative of normal soil conditions not only in the topsoil but also in the subsoil.

The soils in this area are considered to have a low Cation Exchange Capacity (CEC). A low CEC reflects low soil clay and organic matter content, because CEC is a property of both clay and organic material. The CEC ranges from 5.9 to 10.8 cmol(+)kg-1 for the topsoils. Low CEC implies low nutrient content while the opposite is true for high CEC.

The size limits for sand, silt and clay used in the determination of soil texture classes are sand: 2.0–0.05 mm, silt: 0.05–0.002 mm and clay: < 0.002 mm. The clay content range is from 6% to 22% in the topsoil while the subsoil has a clay content ranging from 24% to 36%. This type of soil texture indicates that the soils can be cultivated easily using normal farm machinery. The texture properties of the soils analysed allow the cultivated soils to be classed as sandy clay loam soils. Sandy clay loam soils are easily cultivated using normal farming equipment.

20.5.12 Traffic Assessment

A Traffic Assessment was undertaken and the following intersections formed part of the Project area:

- Intersection of N11 and D3502.
- Intersection of N11 and Village Access Road A.
- Intersection of N11 and Road B.
- Intersection of N11 and Road R518.

The following is evident from visual observations and traffic survey data:

- Fairly high traffic volumes were observed on the N11 and the R518 in the morning and afternoon peak hours;
- Low volumes of traffic were observed on D3502, Village Access Road A and Road B during both the peak hours; and
- Overall no capacity problems were evident in the morning and afternoon peaks at the intersections.

20.5.13 Community Health Assessment

Access to the Healthcare facilities is a challenge for the rural communities in the proposed Project area as many reside more than 5 km from a health service point and have to rely on public or private transport to access care. In the area surrounding the proposed Project footprint healthcare provision is mainly in the form of mobile clinics which visit the communities once a week. Emergency services are limited, especially after clinic operating hours (4 pm). Services are free substantiated by more than 90% of respondents claiming not to pay for medical services. The communities have a relatively high dependency ratio due to the high levels of poverty and unemployment.

Under the light of Healthcare services and infrastructure, the proposed Project impacts need to be considered in two tangents. One, being a positive impact whereby there is the potential for the proposed Project to support the development of improved health services through direct and indirect interventions; and the second, being a negative impact whereby the proposed Project may stretch the already burdened capacity of the Healthcare services in the Mokopane and communities in the vicinity of the proposed Project area.

An influx of people into the proposed Project area can be expected and may have specific health impacts. The spontaneous migration and settlement of labourers and their families may introduce a wide range of concerns into the proposed Project area. These include:

- Increased use of and demand for already inadequate community housing, water, sanitation, food, and medical services can mean that health needs go unmet and new health challenges arise (with a likely increase in cost). At this point in time, there is only one informal settlement in the area (Mzombane) and there is a concern that more settlements of this form and nature could proliferate with related health and social concerns.
- Housing inflation and potential increase in communicable diseases like tuberculosis (TB) and Human Immunodeficiency Virus/ Acquired Immune Deficiency Syndrome (HIV/AIDS). This can, however, be mitigated by Health Systems Strengthening (HSS) to improve TB case detection and case management in local dispensaries; developing and maintain site based TB policies and programmes; as well as outbreak preparedness and response plans.
- Emergency services are already limited in the area and increase trauma and accidents will place added burdens on the health infrastructure.
- The potential to increase accidents and injuries due to changes in road traffic may significantly and adversely affect levels of accidents in the area.

Poverty and high levels of illiteracy and unemployment play a key role in local social challenges. The youth are especially at major risk for social ills such as alcoholism and drug abuse. These in turn play a major role in domestic violence and high risk sexual behaviour. There is a high degree of hopelessness in communities, which is especially pronounced amongst the youth.

There is the possible impact due to increased demand on limited services and increased potential for environmental contamination. A number of determinants can influence the potential for an increase in HIV/AIDS in the proposed Project Area. These are generally as an indirect influence of the project but some direct impacts from the workforce do exist. Some mitigation measures to abate these include: developing a community based HIV and Sexually Transmitted Illness (STI) strategy; HIV/AIDS education programmes; and implementing comprehensive HIV and STI management programmes in the workforce.

An influx of people during construction and operational phases of the proposed Project may result in food inflation, increasing food deprivation, nutrition-related diseases. If long term food inflation occurs, food deprivation may affect susceptible sub-populations such as the children and marginalised groups. Poor food hygiene practices may also increase food related illnesses. More consumption of fast food related to increased income may increase non-communicable (lifestyle) diseases such as obesity and diabetes. This can be mitigated through curbing food inflation and assisting with food and sanitation awareness materials to local district environmental health officers for educational sessions with food handlers and slaughterhouses, particularly vendors who sell food to construction workers and employees. Education on lifestyle behaviours including eating habits, exercise, etc. would also lessen the health impacts thereof. Share educational materials for use in local clinics.

The proposed Project may lead to increased traffic loads on primary and access roads and has thus the potential to increase the number of traffic accidents. This can be abated through improving road safety by collaborating with the district road-safety unit to establish and maintain pictorial road-safety signage near the site in local language (either SePedi or Shangaan) and English language (if needed); clearly demarcated pedestrian crossings in appropriate places etc.

While vector borne diseases are not common in the proposed Project Area, uncontrolled digging and the influx of people coupled with poor environmental management may lead to establishment of vector breeding sites in the proposed Project Area, a situation that may lead to emergence and increase in prevalence of vector-borne diseases. Assist in the controlling of vector breeding sites. Vector control in the local communities using Indoor Residual Spraying (IRS) is possible, however, sustainability issues are extremely important and best practice guidelines should be implemented. Efficient environmental management of surface water is essential, particularly during construction. Coordination with the relevant government departments (i.e. health and social development) in establishing vector awareness programs is also essential.

With regards to the social determinant of health, the expected influx of people and increased income may result in illegal substances being available more freely. It is difficult to speculate whether the prevalence of tobacco smoking and or substance abuse will increase due to the presence of the proposed Project. However, it is likely that it will increase as there will be an increase in the number of young people with decent incomes, who will be in a position to afford these commodities.

When discussing the exposure of people to potentially hazardous materials, noise and malodours, one needs to be cognisant of the in-migration of people. An influx of people into the area may increase domestic activities, including the use of domestic fuels. This may result in an increase in air pollution exposure, followed by associated increases in the prevalence of related respiratory illnesses. The clearing of the site (construction phase) and vehicular movement are the main activities and may have potential impacts on the ambient noise levels. Increased activity of vehicles and heavy machinery, and the drilling of rock will all contribute to the increased local noise levels. There is sufficient evidence that noise causes adverse health effects such as cardiovascular effects.

20.6 Rehabilitation and Closure

Internationally and in the South African context, the broad rehabilitation objectives include three schools of thought, explained below:

- Restoration of previous land use capability.
- No net loss of biodiversity.
- What the affected community wants, the affected community gets.

Rehabilitation objectives need to be tailored to the Project at hand and be aligned with the EMP and Mine Closure Plan. And thus, the overall rehabilitation objectives for the Project are as follows:

- Provide for a sustainable post-mining land use and re-establishment of the pre-mining land use/capability;
- Maintain and minimise impacts to the functioning wetlands and water bodies within the area;
- Implement progressive rehabilitation measures where possible (i.e. contractor's camps and areas used during the construction phase)
- Prevent soil, surface water and groundwater contamination;
- Comply with the relevant local and national regulatory requirements; and
- Maintain and monitor the rehabilitated areas.

The conceptual Rehabilitation Plan provides a description of the management and rehabilitation of the area to be affected by the proposed mining activities. The conceptual Rehabilitation Plan focuses on the following:

- Land Preparation;
- Soil Management Plan and Amelioration;
- Infrastructure (demolition and future use);
- Vegetation and Fertiliser Management Plan;
- Weed Control and Alien Invasive Control Plan;
- Monitoring and Maintenance of receiving environment; and
- Wetland Rehabilitation.

Digby Well was appointed to calculate the environmental closure liability for Platreef's Mine Works Programme in support of the Mining Right Application. The cost required for the first 10 years of mining according to the DMR methodology is US$3.9M. The closure cost is paid as a bond in the form of a bank guarantee. The closure liability will be assessed and updated on an annual basis and reported on to the DMR.

20.7 Summary of Monitoring Plans

Environmental monitoring programmes have been developed for the aspects listed below and will be implemented upon commencement of the construction phase of the project:

- Aquatic ecology – biomonitoring.
- Air quality.
- Noise.
- Surface water.
- Groundwater.

20.8 Summary of Environmental Authorisation Conditions

Environmental Authorisation was received from the LEDET for the listed activities applied for under the NEM Act on 27 June 2014, which specified various conditions Platreef must adhere to during the Project. An application to submit some of these conditions was subsequently submitted to LEDET and an amendment to the Environmental Authorisation was received on 19 September 2014. Platreef is still in the process to amend some aspects in the Environmental Authorisation.

The list below gives a summary of the conditions in the Environmental Authorisation:

- The bulk sampling shaft that was approved by the Department on 27 March 2014 has been incorporated in the Authorisation received on 27 June 2014 and will be used as the Shaft #1 on the mine.
- The proposed mine and its associated infrastructure must not encroach into the Witvinger Nature Reserve or any other sensitive landscapes such as intact ridges, bushveld habitat and riparian areas.
- A 300m buffer must be observed from the boundaries of all the mining infrastructure and water courses, drainage lines and wetland areas identified within the proposed mine development site.
- The protected plant species identified within the proposed mining site must not be removed, cut and/or destroyed unless the necessary permission is granted by the Department of Agriculture, Forestry and Fishery in terms of the National Forests Act, 1998 (Act 84 of 1998).
- The natural areas within the proposed site must be conserved as far as possible and an alien invasive plant eradication plan must be implemented throughout the life of the mine.
- The Red Data species, Oreochromis mossambicus, which was identified in the river systems must be monitored and managed according to its status as included in the Red Data Book and biomonitoring should be done on an annual basis.

- All heritage resources identified on site must be managed as per the approval of the South African Heritage Resources Agency (SAHRA). No graves may be removed without permission from SAHRA.
- Geotechnical requirements and monitoring during the shaft sinking process must be adhered to.
- All other licences must be in place prior to the commencement of those activities for example waste and water use licences.
- The conditions of the conveyor belts must be monitored and managed.
- Infrastructure should be painted in natural colours which will blend into the surrounding landscape and specifically designed lighting equipment that minimises the upward spread of light near to and above the horizontal must be used.
- Effluent sampling, testing and reporting must be undertaken and record of such activities be documented.
- Berms, cut-off trenches and surface water management systems must be constructed to divert dirty water to the pollution control dams and to prevent pollution within the site.
- Only tested and uncontaminated waste rock must be re-used for construction purposes within the mine and all contaminated must be used for backfilling or alternatively be disposed of on the waste rock dump. The waste rock dump must be monitored for stability and potential acid mine drainage, and the associated surface runoff must be diverted to the return water dam.
- Surface and groundwater resources must be monitored and reported on to detect and mitigate impacts from mining activities early on.
- Filtration systems must be used to filter air from underground to remove underground air pollutants.
- All dirty water must be stored in lined pollution control dams and recycled for process water.
- The stoping methodology must be adhered to and the mine shafts must be monitored annually for the risk of subsidence.
- Air quality monitoring (dust fallout), soil erosion assessments and remediation should be undertaken.
- A 500m safe blasting distance from the boundaries of the affected communities must be observed.
- All road upgrades must be approved by the Roads Agency of Limpopo prior to the commencement thereof.
- Support must be provided to local health authorities.
- Approval for the provision of electricity must be obtained from Eskom.
- All social issues must be addressed in the Social and Labour Plan.

21 CAPITAL AND OPERATING COSTS

This section has not been changed from the Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

21.1 Introduction

The capital and operating costs have been divided into functional costs areas as follows:

- Capital
  - Mining
  - Concentrator and Infrastructure
  - Tailings Storage Facility (TSF)
- Operating Costs
  - Mining
  - Concentrator
  - TSF
  - General and Administration

Indirects including Owners Costs, contingency and closure costs have been included within each area. The estimate base date has been defined as January 2014. All operating costs exclude VAT. The base currency for the Platreef Project is in United States Dollars ($). Costs estimated in in South African Rand (R) are converted to $ at the following rate: R11=$1.00.

21.2 Capital Cost Summary

The estimated value covers the direct and indirect costs for all equipment, temporary facilities, materials and labour required to construct and complete the permanent works. An allowance for mining development has been made as well as sustaining capital over the 30 year mine life (excluding pre-production).

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 21.1.

Table 21.1 Capital Expenditure Summaries

US$M	Pre-Production	Sustaining	Total
Mining			
Underground	542	956	1,498
Surface Infrastructure	64	-	64
Backfill Plant	21	14	34
Capitalised Operating Costs	35	-	35
Subtotal	**661**	**970**	**1,631**
Processing & Tailings			
Concentrator	93	181	274
Rietfontein TSF	30	30	59
Subtotal	**123**	**211**	**334**
Infrastructure			
General Infrastructure	115	63	178
Site Pre-Production	8	1	9
Closure Costs	-	18	18
Subtotal	**123**	**83**	**206**
Indirects			
Exploration & Geology	3	0.4	4
Engineering Procurement Contract Management (EPCM)	59	17	75
Capitalised G&A & Other Costs	15	6	21
Subtotal	**77**	**23**	**100**
Owners Cost	71	4	75
Capex Before Contingency	**1,054**	**1,291**	**2,345**
Contingency	114	110	224
Capex After Contingency	**1,168**	**1,401**	**2,569**

1. Sustaining capital expenditure also includes 2019 construction capital expenditure
2. Totals vary due to rounding.

21.3 Operating Cost Summary

The operating costs and revenues are summarised in Table 21.2. Mine site cash costs denominated in $/oz Payable 3PE+Au are presented in Table 21.3.

Table 21.2 Total Operating Costs and Revenues

	Life of Mine TOTAL US$M	US$/t milled		
		5-Year Average	10-Year Average	Life of Mine Average
Gross Sales Revenue	22,981	188.18	189.54	191.19
Less: Realisation Costs				
Transport Costs	195	1.55	1.59	1.63
Treatment & Refining Charges	4,137	33.87	34.12	34.41
Royalties	908	5.61	6.82	7.55
Total Realisation Costs	**5,240**	**41.03**	**42.52**	**43.59**
Net Sales Revenue	**17,742**	**147.15**	**147.02**	**147.60**
Site Operating Costs				
Mining	3,585	32.71	31.44	29.83
Processing & Tailings	1,372	11.88	11.58	11.42
General & Administration	434	3.86	3.70	3.61
Total	**5,392**	**48.45**	**46.72**	**44.86**
Operating Margin	**12,349**	**98.70**	**100.30**	**102.74**
Operating Margin (%)	**54%**	**52%**	**53%**	**54%**

Totals vary due to rounding.

Table 21.3 Mine Site Cash Costs

	US$/oz Payable 3PE+Au		
	Life-of-Mine Average	5-Year Average	10-Year Average
Mine Site Cash Cost	401	454	429
Realisation	390	384	390
Total Cash Costs Before Credits	**792**	**838**	**819**
Nickel Credits	389	438	419
Copper Credits	80	90	86
Total Cash Costs After Credits	**322**	**310**	**314**

Totals vary due to rounding.

21.4 Mining Capital Costs

This section describes the parameters, basis, and exclusions for capital estimates for mining the Platreef Mineral Reserve. The overall cost estimate was compiled to include pre-production capital, sustaining capital, and mine operating costs. The capital costs are estimated at a prefeasibility-level of accuracy. For the pre-production period, Owner's Costs; Contingency; and Engineering, Procurement, and Construction Management (EPCM) have been captured in the overall project capital cost summary.

21.4.1.1 Contractor Direct Costs

Contractor costs include cost elements listed below. Key aspects of each cost component are also noted.

- Labour
  - Labour includes a combination of direct and indirect labour required to complete the specified task. Labour can include hourly and staff personnel depending on the type of activity.
- Permanent Materials
  - Permanent materials include all materials installed or consumed while performing the specified task, such as concrete, timber, support steel, etc. It is assumed that the contractor provides all permanent materials.
- Direct Charge Equipment
  - Direct charge equipment includes specialised equipment written off by the contractor while performing the specified task. Rental rates are not applied to this equipment; it is either entirely written off or salvaged at work completion. Items that fall into this category include work stages, concrete forms, etc.
- Equipment Rentals
  - Equipment rentals include rental costs for contractor-owned equipment used to complete the specified task. Equipment rentals are charged at a monthly rate, which is typically assessed as a percentage of the equipment cost. Contractors provide mobile equipment during pre-production, based on a rental rate of 5%.
- Equipment Operating Costs
  - Equipment operating costs includes costs associated with operating all equipment owned or operated by the contractor. Operating costs typically include fuel, lubrication, repair parts, overhaul parts, tire replacement, and ground engaging components (if applicable), but excludes electrical power (refer to Section 21.4.1.3).
- Service and Supplies
  - Service and supplies includes consumable items such as explosives, drilling costs, pipelines, ventilation duct, small tools, etc., associated with the specific task.
- Subcontractors
  - Subcontractors includes subcontractor costs associated with the specific task, such as drain hole drilling, diamond drilling, assaying, etc.

21.4.1.2 Contractor Indirect Costs

Contractor indirect costs include costs incurred by the contractor to complete specific mine development and construction activities but are not included in any direct capital cost items.

An assessment was made of contractor indirect costs during pre-production. The costs are estimated on a daily basis and extended by the activity duration. The contractor indirect costs include the items listed below.

- Supervision Labour.
- Mechanical and Electrical Maintenance Labour.
- Temporary Surface Support Equipment Rentals (compressors, generators, fans, wash / dry trailers, office trailer, pick-up trucks, etc.).
- Operating Costs for the Temporary Support Equipment.
- Service and Supply Costs (sustaining freight, phone / fax, safety and supplies, sewage / garbage disposal, etc.).

21.4.1.3 Mining Owner Costs

Permanent Capital Equipment

Permanent capital equipment includes the costs associated with purchasing fixed and mobile equipment. In addition, rebuild and replacement costs are assessed against mobile equipment. Data from other recent projects was used to develop permanent capital equipment costs.

To assess permanent capital costs, equipment lists are developed from infrastructure designs and operating parameters. Once an equipment list is compiled, purchase, rebuild, and replacement costs are estimated. Following are the key elements used to develop the unit cost database.

- Item Description – Identifies and sometimes provides a brief technical description of the equipment duty requirements or capacity.
- Base Cost – Quoted by a vendor or taken from a historical cost database, including the cost for options.
- Development Allowance – A 5% allowance to cover the cost of miscellaneous components, fuels, lubricants, and services required to commission a piece of equipment.
- Spares Allowance – A cost allowance for spare parts required on site. When provided, the cost of spares recommended by the vendor is included. Similarly, engineering judgment is used to reduce or eliminate spares allowances for identical units that appear in multiple equipment list areas. The following guidelines are used to calculate spare parts costs.
  - Single Units – 5%
  - Pairs of Units – 5%
  - Multiple Units – 5%

- Freight Allowance – A cost allowance for delivering equipment to site. An average of 5% of the base cost is used when a vendor quotation is not provided.
- Total Unit Cost – A summation of the base cost, development allowance, spares allowance, and freight allowance all escalated into current terms, if necessary. The total cost excludes sales tax and contingency.

Once base unit costs are developed, mobile equipment rebuild and replacement unit costs are estimated. In general, these costs are based on annual operating hours and estimates of the average life to rebuild and replace, which varies to suit the type of equipment. For the purposes of this evaluation, an annual allowance was estimated at 15% of new equipment value for all mobile equipment.

Engineering, Procurement, and Construction Management

Engineering, procurement, and construction management (EPCM) costs for the pre-production period are determined as a 7.5% allowance assessed against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs."

Owner's Team

The owner's team cost may include a manager, mine clerk, and mine engineer. Costs include labour, surface pickup operating costs, permanent equipment costs, and miscellaneous costs to support the owner's team from the start of pre-production to the start of production. The cost represents an assessment of a 3% allowance against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs." During the pre-production period owner's team costs have been captured in the overall project capital cost summary.

Electric Power Consumption

Power costs associated with pre-production mine development and construction are assessed at a calculated rate of US$0.061 per kWh.

Water Supply

Water supply costs associated with pre-production mine development and construction are assessed at a rate of US$1.70 per m^3.

Contingency

A capital contingency of 30% is assessed against the total contractor's and owner's costs with the exception of mobile equipment related costs. A contingency of 10% is assessed against mobile equipment costs. The contingency provides additional project capital for expenditures that are anticipated but not defined due to the level of engineering detail in the study. During the pre-production period contingency costs have been captured in the overall project capital cost summary.

21.4.1.4 Capital Cost Criteria and Assumptions

Capital mine development work performed during pre-production is performed by a contractor. After production begins, owner's personnel perform the work.

The estimates are based on the following qualifications.

Pre-production Costs

- Costs are based on constant 2014 US$.
- Pre-production contractor crews will work 7 days per week, 2 shifts per day, and 11 hours per shift.
- There will be an allowance of 5 days of non-production per year.
- Contractor will provide all underground equipment during the pre-production period except mobile equipment, which will be purchased by the owner and operated by the contractor during the pre-production period.
- Contractor's margins / insurances are assessed at 17.5%.

21.5 Concentrator Plant Capital Cost Estimate

21.5.1 General

The prefeasibility capital cost estimate in this section covers the concentrator plant and associated in-plant infrastructure as per the selected flowsheet illustrated in Section 17 of this report. The concentrator plant capital estimate is based on a modular approach using two milling-flotation modules with a capacity of 2 Mtpa to achieve 4 Mtpa normal throughput.

21.5.2 Pre-Production Capital

21.5.2.1 Basis of Estimate

The base date for the process plant capital cost estimate is January 2014. South African Rands (R) was used as the base currency for the estimate, with an exchange rate of R11 = US$1.

The general approach to estimating was to measure and quantify each cost element from the engineering block plans, general arrangement and layout drawings, PFD's and mechanical equipment list. Firm, or budget quotations from vendors were obtained for the major process/mechanical equipment whereas minor items of mechanical equipment were single sourced, or taken from the DRA database.

Earthworks and civil works quantities were taken from the block plan, general arrangement and layout drawings. Bills of Quantities was produced in order to quantify all the items of earthworks and civil works in a standard SANS format Bill of Quantities. Rates from the in-house DRA database have been used to populate the estimate.

The building works were identified from the project infrastructure list as per the work breakdown structure and block plans and quantified from the general arrangement and typical layout drawings and a schedule of square meterage was produced for estimation.

Rates from the in-house DRA database have been used to populate the estimate.

The mechanical equipment and motor lists were used to derive a factorised estimate for the electrical, instrumentation and control engineering disciplines.

The mechanical equipment supply costs were used as the basis to derive a factorised estimate for the piping discipline.

Preliminary and general costs have been estimated per discipline by using the DRA database factors for most recent projects executed and ranges between 40% for bulk earthworks up to 65% for E&I installation.

Project services capital costs have been calculated based on estimates for various project services, including EPCM costs and owners cost allowed at 14% and 3% of total direct cost excluding contingency respectively.

A project contingency, risk and estimating inaccuracy allowance of 20% has been included. Escalation cost has been excluded and the estimate is stated in real money terms.

The Platreef Project process plant estimate has been benchmarked against similar platinum projects executed by DRA in the Southern Africa region. Ratios and factors were compared to the respective engineering disciplines, also taking into consideration the Platreef Project specific details for battery limits, exclusions, founding and ground conditions and the proposed location of the Platreef Project.

21.5.2.2 Input Documents

The following documents were used as the main engineering inputs to the prefeasibility study capital estimate for the concentrator plant:

- Mechanical equipment list based on the updated process design and selected flowsheet.
- Surface geotechnical and hydrology specialist study reports.
- Plot and block plans, engineering design criteria, conveyor designs, structural and conveyor layouts and general arrangement drawings.
- Electrical motor list, MV single line diagram.
- Work breakdown structure and project implementation plan.

21.5.3 Sustaining Capital

Sustaining capital was allowed per year of operation, based on an industry norm that Sustaining capital is on average 2.5 % of plant capital expenditure per year, excluding in plant infrastructure.

No Sustaining capital is allowed for year 1 of operation as it is assumed that initial pre-production capital will still cover for this year and no trigger event for the use of sustaining capital will occur. No sustaining capital is allowed for the final three years of operation, as it is assumed that within these three years no need will arise to provide for sustaining capital.

21.5.4 Closure Costs

The concentrator plant closure cost has been calculated based on the estimated quantities of concrete and tonnages of steel and pipe work erected. Contractor rates for the demolishment and removal from Site have been applied to these quantities. Mechanical and electrical equipment will be sold or removed from Site without additional cost. The closure cost for the concentrator plant amounts to US$9.09 million. No contingency has been allowed for closure cost.

21.5.5 Concentrator Plant Capital Summary Sheets and Graphs

Table 21.7, Table 21.8, and Table 21.9 summarises the values included into this estimate.

Figures 21.9 and 21.10 show the concentrator plant pre-production and total capital distributions respectively.

Table 21.4 Concentrator Plant Pre-Production Capital Summary (area view)

Area	Unit	Total
Crushing		
ROM Storage and Reclaim	US$M	10.7
Crushing	US$M	15.0
Screening	US$M	8.0
Mill Feed Storage and Conveying	US$M	5.4
Subtotal	**US$M**	**39.1**
Milling		
Milling Circuit Stream #1	US$M	11.8
Milling Circuit Stream #2	US$M	9.9
Pre-Conditioning	US$M	3.0
Subtotal	**US$M**	**24.8**
Flotation		
Rougher Flotation	US$M	7.3
Cleaner Flotation	US$M	5.5
Scavenger Cleaner Flotation	US$M	3.0
Subtotal	**US$M**	**15.7**
Tailings		
Tailings Thickener	US$M	2.8
Tailings Disposal	US$M	1.0
Subtotal	**US$M**	**3.8**
Concentrate Handling		
Combined Conc Handling	US$M	8.1
Subtotal	**US$M**	**8.1**
Utilities and Services		
Water Circuit	US$M	1.0
Air Services	US$M	1.2
Subtotal	**US$M**	**2.1**
Reagents		
Xanthate Make-up and Dosing	US$M	2.2
Depressant Make-up and Dosing	US$M	0.4
AERO 3477 Make-up and Dosing	US$M	0.03
Frother Make-up and Dosing	US$M	0.03
Thiourea Make-up and Dosing	US$M	0.1
Oxalic Acid Make-up and Dosing	US$M	0.1
Flocculant Make-up and Dosing	US$M	0.4
Subtotal	**US$M**	**3.2**
In Plant Infrastructure		
Roads and Earthworks	US$M	5.5
Services	US$M	7.2
EC&I	US$M	16.9
Buildings	US$M	10.1
Waste Management	US$M	0.003
Fencing and Access Control	US$M	0.5
Subtotal	**US$M**	**40.2**
P&G's	US$M	30.3
TOTAL	**US$M**	**167.4**

Table 21.5 Concentrator Plant Pre-Production Summary (category view)

Area	Unit	Total
Concentrator - See Table 21.4	US$M	167.4
EPCM Contractor	US$M	23.6
Project Services		
Financials	US$M	-
Owners Project Team	US$M	5.5
Future Studies	US$M	2.7
Project Implementation	US$M	9.4
Consultants	US$M	0.7
Logistics and Freight	US$M	0.1
Commissioning	US$M	1.2
Subtotal	**US$M**	**19.6**
Spares and Consumables	US$M	10.6
Contingency	US$M	43.6
TOTAL	**US$M**	**264.8**

Table 21.6 Concentrator Plant Sustainable Capex and Closure Cost

Area	Unit	Total
Concentrator Sustainable Capex	US$M	85.8
Concentrator Closure Cost	US$M	9.1

Figure 21.1 Pre-Production Capital Distribution (category view)

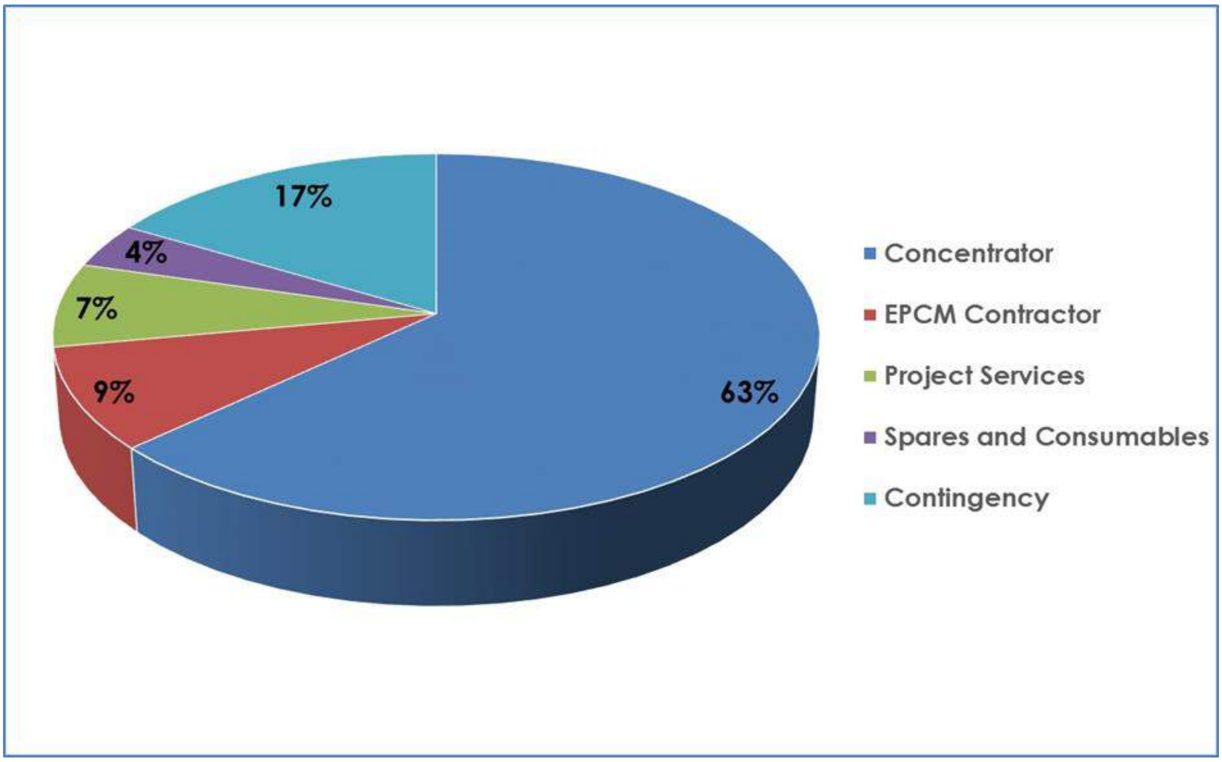
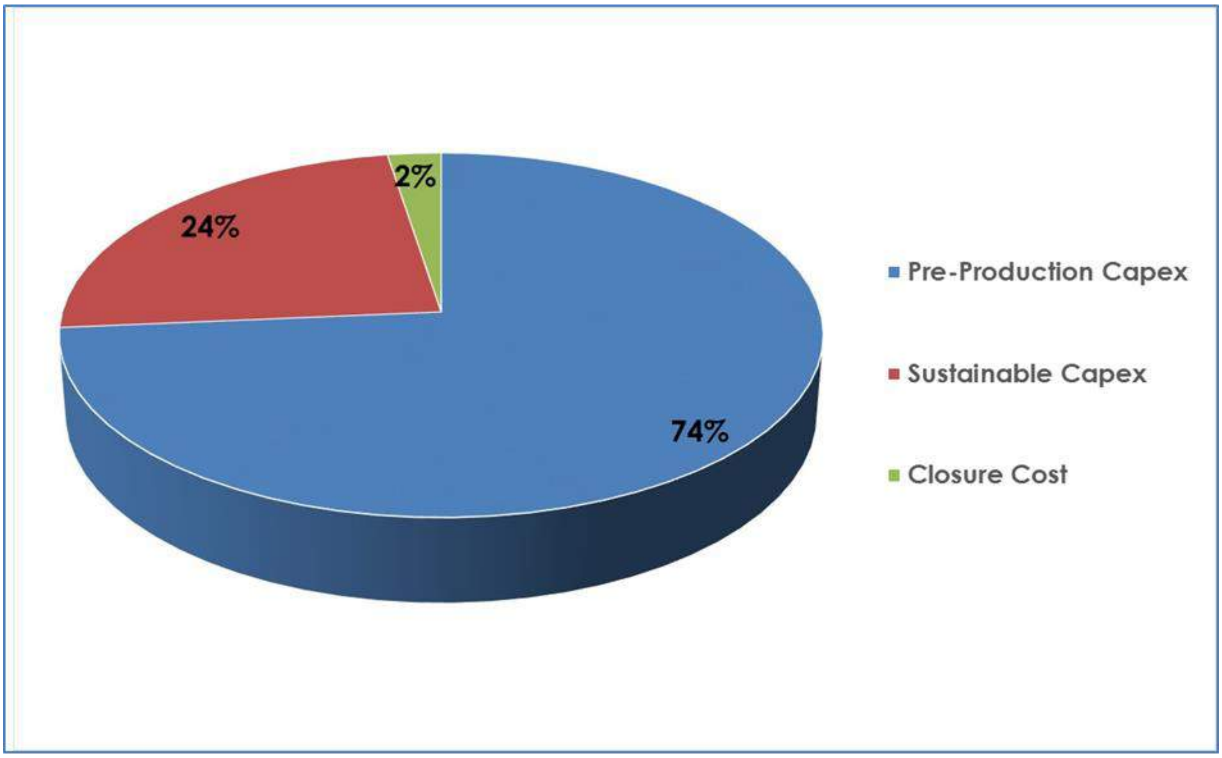


Figure 21.2 Concentrator Plant Total Capital Distribution



21.6 General Infrastructure Capital Cost Estimate

21.6.1 General

The prefeasibility capital cost estimate in this section covers the general infrastructure for the Platreef Project including bulk power and water supply, but excludes the specific in-plant infrastructure at the concentrator and mining shaft areas, as well as the tailings storage facility (TSF).

21.6.2 Pre-Production Capital

21.6.2.1 Basis of Estimate

The base date for the general infrastructure capital cost estimate is January 2014. South African Rands (R) was used as the base currency for the estimate, with an exchange rate of R11 = US$1.

The general approach to estimating was to measure and quantify each cost element from the engineering block plans, general arrangement and layout drawings, utility diagrams and mechanical equipment list. Budget quotations from vendors were obtained for the mechanical equipment whereas minor items of mechanical equipment were taken from the DRA database.

Earthworks and civil works quantities were taken from the block plan, general arrangement and layout drawings. Bills of Quantities were produced in order to quantify all the items of earthworks and civil works in a standard SANS format Bill of Quantities. Rates from the in-house DRA database have been used to populate the estimate.

Building works were identified from the project infrastructure list as per the work breakdown structure and block plans. Quantities were obtained from general arrangements and typical layout drawings and a schedule of square meterage was produced for estimation. Rates from the in-house DRA database were used to populate the estimate.

Piping take-offs were done for all major surface run utility and buried services, including tailings and return water lines between concentrator and TSF. Block plans and utility diagrams per utility service identified the items for take-offs.

The mechanical equipment and motor lists were used to derive a factorised estimate for the electrical, instrumentation and control engineering disciplines.

Preliminary and general costs have been estimated per discipline by using the DRA database factors for most recent projects executed and ranges between 40% for bulk earthworks up to 65% for E&I installation.

Project services capital costs have been calculated based on estimates for various project services, including EPCM costs and owners cost allowed at 10% and 3% respectively of total direct cost, excluding bulk power and water supply and contingency.

A project contingency, risk and estimating inaccuracy allowance of 20% has been included.

Escalation cost has been excluded and the estimate is stated in real money terms.

The Platreef Project specific details for battery limits, exclusions, founding and ground conditions and the proposed location of the Platreef Project have been taken into consideration.

21.6.2.2 Input Documents

The following documents were used as the main engineering inputs to the prefeasibility study capital estimate for general infrastructure:

- Surface geotechnical and hydrology specialist study reports.
- Plot and block plans, engineering design criteria, layout and general arrangement drawings.
- Mechanical equipment list.
- Electrical motor list, MV single line diagram.
- Work breakdown structure and project implementation plan.

21.6.3 Sustaining Capital

Sustaining capital for Infrastructure was developed from first principles, and taken as three of complete tailings line replacements.

21.6.4 Closure Costs

Closure costs have been included in the respective Mining, Concentrator Plant and Tailings Storage Facility (next section) estimates. No additional closure costs have been allowed for the general infrastructure area.

21.6.5 General Infrastructure Capital Summary Sheets

Table 21.10, Table 21.11, and Table 21.12 summarises the values included into this estimate. Two views are provided, an Area view and a Category view. The Area view does not include contingency and services, whereas the Category view includes contingency and services.

Table 21.7 General Infrastructure Pre-Production Capital Summary (area view)

Area	Unit	Total
General Infrastructure		
Earthworks and Roads	US$M	20.5
Services	US$M	63.8
Electrical	US$M	24.6
Substations, Buildings & Workshops	US$M	6.8
Fuel and Lubrications	US$M	0.8
Accommodation	US$M	14.4
Vehicles	US$M	9.8
Waste Management	US$M	2.0
Fencing and Access Control	US$M	2.6
Subtotal	**US$M**	**145.2**
P&G's	US$M	21.4
TOTAL	**US$M**	**166.6**

Table 21.8 General Infrastructure Pre-Production Summary (category view)

Area	Unit	Total
General Infrastructure – See Table 21.7	US$M	166.6
EPCM Contractor	US$M	8.2
Project Services		
Financials	US$M	-
Owners Project Team	US$M	2.5
Future Studies	US$M	1.2
Project Implementation	US$M	2.3
Consultants	US$M	0.2
Subtotal	**US$M**	**6.2**
Spares and Consumables	US$M	-
Contingency	US$M	35.2
TOTAL	**US$M**	**216.2**

Table 21.9 General Infrastructure Sustainable Capex Cost

Area	Unit	Total
General Infrastructure Sustainable Capex	US$M	13.1

21.7 Tailings Storage Facility Capital Cost Estimate

21.7.1 General

The prefeasibility capital cost estimate in this section covers the tailings storage facility (TSF) on Rietfontein 2 KS farm. The Rietfontein TSF design and estimate have been prepared by Geo Tail as the specialist engineering consultant for TSF design.

21.7.2 Pre-Production Capital

21.7.2.1 Basis of Estimate

The base date for the Rietfontein TSF capital cost estimate is January 2014. South African Rand (R) was used as the base currency for the estimate, with an exchange rate of R11 = US$1.

The estimate is based on measured designs and includes a proposed liner system that generally complies with the Class C liner type in the waste classification regulations according to Government Notice R.634 (Government Gazette No. 36784, 23/08/2013) pertaining to the National Environmental Management Waste Act (Act No. 59 of 2008) by the Department of Environmental Affairs.

A project contingency, risk and estimating inaccuracy allowance of 20% has been included.

Escalation cost has been excluded and the estimate is stated in real money terms.

21.7.2.2 Pre-Production Capital Summary

The capital cost for the pre-deposition civil works associated with the Rietfontein TSF can be summarised as follows:

Table 21.10 TSF Pre-Production Capital Cost (area view)

Area	Unit	Total
Tailings Storage Facility	US$M	49.6
P&G's	US$M	9.6
TOTAL	**US$M**	**59.2**

Table 21.11 TSF Pre-Production Capital Cost (category view)

Area	Unit	Total
TSF – See Table 21.10	US$M	59.2
EPCM Contractor	US$M	1.4
Project Services		
Future Studies	US$M	1.1
Subtotal	**US$M**	**1.1**
Contingency	US$M	9.6
TOTAL	**US$M**	**71.3**

21.7.3 Closure Costs

Closure cost is estimated based on benchmarking against similar projects and where necessary on first level quantification and market related rates to a ±30% accuracy level.

Table 21.12 Closure Cost TSFs

Area	Unit	Total
TFS Closure Cost	US$M	8.5

21.8 Mining Operating Costs

Unit operating costs were estimated for ore development, longhole stoping, drift-and-bench stoping, and drift-and-fill mining methods based on annual tonnages produced each year. Costs for paste backfill were provided by Golder and are therefore excluded from the estimate. Total operating costs are summarised in Table 21.13.

Table 21.13 Mining Operating Cost Summary

Description	Unit	Total
Production Direct Costs	US$/t	9.74
Backfill Costs (by Golder)	US$/t	5.53
Indirect Operating Costs	US$/t	8.12
Power Costs	US$/t	4.84
Water Cost	US$/t	0.26
Undefined Allowance	US$/t	1.17
Total Operating Cost	**US$/t**	**29.66**

21.8.1 Operating Cost Estimate Scope Definition

These cost estimates are intended to cover all expenses required to operate the mine and produce ore. The operating costs are subdivided into the following cost centres.

21.8.1.1 Lateral and Vertical Development

Development includes all ore development (drill / draw drift development and drill drift slashing), remaining lateral waste development, stope access, and miscellaneous excavation.

21.8.1.2 Production Direct Costs

Production direct costs includes the costs to drill, blast, ground support, and muck from the stopes into the dedicated haulage trucks or passes and material handling to the surface.

21.8.1.3 Indirect Operating Costs

Indirect operating costs includes the cost of pumping, ventilation, providing compressed air, mine service crews and trainees/spares, mechanics and electricians for mine operations, and site staff personnel for the mine.

21.8.1.4 Power Costs

A detailed annual power load sheet was prepared and annual power usage was estimated based on the yearly production and estimated power consumption. Power load estimates were calculated annually based on the estimated power requirements for the following activities.

- Waste Development (drills and bolters)
- Production Drilling
- Material Handling (hoisting, conveyors, apron feeders, etc.)
- Ventilation
- Mine Air Cooling
- Compressor Plant
- Dewatering
- Miscellaneous (office / dry facility, underground shop, core drills, raisebores, miscellaneous lighting, etc.)

21.8.1.5 Operating Cost – Undefined Allowance

An allowance was included as 5% of the operating costs to allow for unforeseen costs.

21.8.2 Exclusions from Mining Operating Cost Estimate

In addition to capital construction related costs estimated by Stantec, the following operating cost items are included elsewhere in the operating costs:

- Federal or local sales taxes on permanent materials or services.
- Milling, refining, or shipping costs.
- Contingency on operating costs.

- Finance charges and interest charges.
- Land acquisition, rights-of-way, licenses, and royalties.
- Disposal of hazardous materials.
- Escalation.
- Inflation.
- Mineral royalties.
- Costs for titles, title insurance, legal services, and surveying to evaluate, negotiate for, and purchase land for development of the mines.
- All licenses, permits, and maintenance of same, including, but not limited to the following.
  - Requirements for environment.
  - Construction.
  - Explosives purchase, transport, and usage.
  - Mining operations.
  - Water and air discharge.
  - Equipment and supplies importation.
- Value-added tax, customs, excise, duties, sales, or other import taxes.
- Employee housing.
- Flights to and from site.
- Telephone, facsimile, and satellite links provided for the purpose of communications.
- Explosives plant.
- Surface rock haulage from stockpiles.
- Surface rock haulage to low-grade stockpiles and dumps. During shaft sinking and preproduction development, waste and low-grade rock will be hoisted to the surface and dropped on the ground adjacent to head frames. Rock will be loaded and hauled to dumps or stockpiles by others.
- On-site power generation to support regular mining operations. Stand-by service is provided during construction.
- All surface facilities other than those specifically identified in cost estimates.
- Treatment and disposal of hazardous or toxic materials.
- Site restoration required for exploration, construction, operations, and closure.
- Medical service facilities beyond those that are typically provided for shaft sinking and mine development in a developed country. There are no provisions in this study for doctors, nurses, a clinic, ambulance operation, and other medical services to support mine construction and operation.
- Camp facilities.

- Personnel providing service functions in support of underground mining and other site operations including, but not limited to, payroll, accounting, information technology, recruiting, human resources, and site security.
- Export credit financing.
- Finance and interest charges.
- Working capital.
- Basic training in safety, work procedures, equipment operation, and underground tasks as well as surface ore and waste haulage.
- Builder's risk insurance.

21.8.3 Additional Mining Costs

21.8.3.1 Material Allowance

Material waste and undefined additional usage are included in the study at a rate of 5% of the designed requirement. This allowance is not considered in the development or construction schedules and is applied on a cost basis, only. No equipment operating materials or supplies are included in this allowance.

21.8.3.2 Vacation, Sickness, Absenteeism, and Training Allowance

Additional personnel will be required on site to cover direct labour during times when individuals are on vacation, sick, absent, or in training. An allowance of 15% is applied to direct labour costs for this purpose. The allowance does not include coverage for initial training of new employees.

21.8.3.3 Operating Cost – Undefined Allowance

An allowance has been included as 5% of the operating costs for miscellaneous costs that may not be accounted for elsewhere in the estimate. This allowance is only applied to direct production costs.

21.9 Backfill Plant Operating Costs

Annual operating costs were estimated by Golder Associates Africa Ltd. (GAA), in collaboration with Golder Paste Engineering and Design of Golder Associates Ltd. (collectively referred to as Golder).

21.9.1 Backfill Operating Cost Summary

A summary of the cemented rock fill operating cost is in Table 21.14 and a summary of the cemented paste fill operating cost is in Table 21.15.

Table 21.14 Cemented Rock Fill Operating Cost Summary

Description	Cost as per Q3 2019
	US$
Electrical Power	19,467
Binder – CRF	878,604
Hydration Retarder	55,267
Misc. equipment and instrumentation	5,000
Total Labour	67,762
Parts and supplies for maintenance	155,619
CRF delivery to stope	95,733
RF delivery to stope	95,733
Barricade Construction	4,920
Total Operating Cost	$1,378,100
Operating Cost per Tonne Cemented Rock Fill	$10.18
Operating Cost per Tonne of all Rock Fill	$5.47
Operating Cost per Tonne Ore	$3.90

Table 21.15 Cemented Paste Fill Operating Cost Summary

Description	Cost as per Q3 2019
	US$
Electrical Power	1,186,500
Flocculant @ 20 g/t	185,858
Binder	18,836,030
Sublevel Boreholes	131,250
Sublevel main line pipe (steel)	447,740
Stope access pipe (HDPE)	126,000
Barricade Construction	1,050,690
Misc. equipment and instrumentation	40,000
Total Labour	1,802,514
Total Annual Operating Cost	**$24,451,710**
Operating Cost per Tonne Cemented Paste Fill	**$10.26**
Operating Cost per Tonne Ore	**$6.11**

21.9.2 Basis of Backfill Operating Cost Estimate

- Electrical and instrumentation costs were derived from the motor lists and prices from similar projects.
- All costs were based on available equipment in South Africa and aligned to South African prices and conditions. The CRF plant however, was priced as a package imported into South Africa.
- Operating labour costs were either estimated or based on actual values obtain from Platreef.
- The cost of consumable items such as cement, electrical power, flocculant, etc., were obtained from vendor quotes or provided by Platreef.

The cost estimate does not include the owner's cost or the contingency.

21.10 Plant Operating Cost Estimate

21.10.1 Summary

The operating cost estimate is based on a metallurgical processing facility with a design throughput of 4 Mtpa.

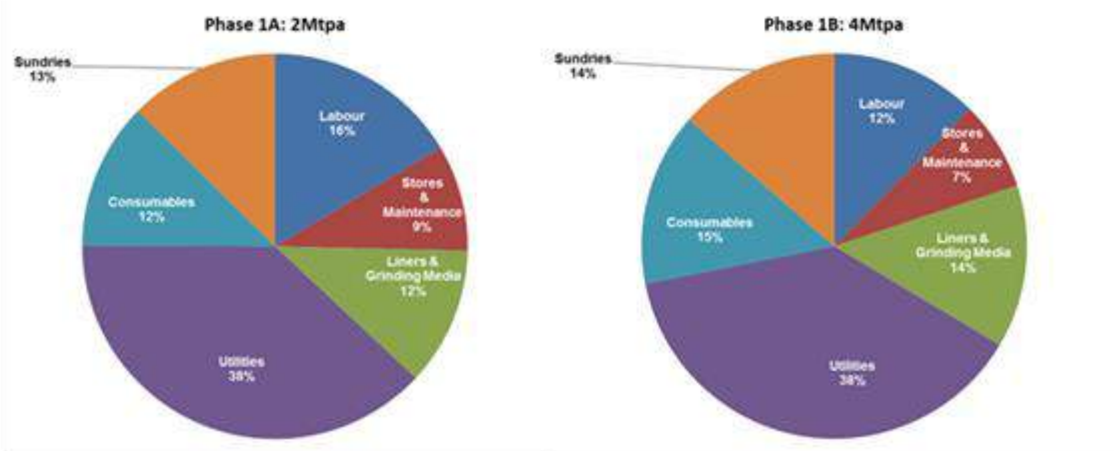
Table 21.16 below summarises the estimated annual operational costs for the Platreef Project concentrator plant Phase 1.

Table 21.16 Plant Operational Cost Estimate

Description	Phase 1A	Phase 1B
	2 Mtpa (US$M)	4 Mtpa (US$M)
Labour Cost	4.4	5.6
Stores & Maintenance	2.4	3.4
Stockpile Reclamation	0.8	0.3
Utilities	10.1	17.3
Consumables	6.4	12.8
Total Annual Cost	**24.0**	**39.4**
Total Unit Cost (US$/t) Excluding contingency	**12.0**	**9.84**

Refer to Figure 21.3 for an illustration of the plant operational cost estimate splits for each phase.

Figure 21.3 Process Plant Operational Cost Splits



Basis of Plant Operating Cost Estimate

The process plant operating cost estimate is based on steady state operating conditions of 2 Mtpa for Phase 1A and 4 Mtpa for Phase 1B. The opex estimate is based on prefeasibility testwork and engineering input and is expected to have an accuracy of ±15 to ±25%. The base currency of the Platreef Project is in United States Dollars (US$). All costs have been estimated in ZAR and converted to US$ at the following rate: ZAR 11.00 = US$1.00. A base date of October 2014 was used.

21.10.2 Process Plant Operating Cost Exclusions

The following are specific exclusions from process plant operating cost estimate:

- General & administration related costs (Covered separately):
  - All owner's budget costs, head office, administration charges, payroll, marketing, training, medicals etc.
  - Insurance; and
  - Contributions to social programs, rehabilitation funds, environmental monitoring and conformance to environmental requirements.
- No allowance is made in the operating cost estimate for ramp-up of fixed costs prior to the start of hot commissioning – i.e. on-boarding of operating personnel etc.
- Concentrate off take agreement costs / penalties.
- Concentrate Transport cost is accounted for in section Concentrate Transport 21.13.
- Laboratory labour and assay costs as these are accounted for in Section 21.12.
- Tailings storage facility operation.
- Waste rock handling (Mining Opex).

- Backfill plant operation (Mining Opex).
- Site closure and rehabilitation.
- All VAT, import duties and / or any other statutory taxation, levies.

No contingency has been allowed for in the operating cost estimate.

21.10.3 Plant Operation Costs Inputs

21.10.3.1 Labour

The process plant labour costs are based on a staffing model of similar platinum processing plants of a similar size and complexity and labour rates as supplied by the Platreef team.

Operating personnel compliments are based on rotational shift teams. Allowances have been made for leave and absenteeism within the process and engineering teams.

21.10.3.2 Power

The power cost used is based on the 2013/2014 Eskom Megaflex Non-local authority tariffs. These tariffs were used in conjunction with the Eskom defined time periods to calculate a time weighted average annual rate of $0.055/kWh. Additional administration costs were added based on the Eskom tariffs booklet.

Power was calculated on the basis of the average continuous power demand for each duty drive. The mechanical equipment list was used to identify all duty drives from equipment sizing calculations to which the utilization and mechanical efficiency factors were applied in order to determine the estimated power consumed.

Emergency power is supplied to critical drives during a power outage by means of a dedicated diesel generator.

21.10.3.3 Water

The monthly water cost estimate is based on a rate of $ 0.91m3, typical costs for the supply from Lebalelo water user association. Raw water consumption figures are based on the plant mass balance and expected losses (evaporation and interstitial losses) at the tailings storage facility. Water consumptions figures used do not account for periodic return water volumes from pollution control dams.

21.10.3.4 Consumables

The consumption figures used in determining the operating cost estimate are based on testwork data and supported by the plant mass balances. Grinding media consumption was based on the abrasion indices obtained from testwork for the specified ore type. The reagent and grinding media costs used in the operating cost estimate are based on supply rates obtained from reputable reagent suppliers to the platinum industry in South Africa.

A single supply price was obtained for all reagents, with the exceptions of oxalic acid and thiourea where three check prices were obtained for each. The reagent prices were obtained in late 2014 and are consistent with the rates currently being paid by other platinum producers. The price differences for oxalic acid and thiourea between the different vendors are due to the selected vendor sourcing the reagents from India and China, respectively. The vendor is of the opinion that the oxalic acid and thiourea supply sources are reliable. The vendor also commented that they keep approximately two months' stock spare in South Africa.

The crusher and mill liner costs were based on supply rates obtained from reputable vendors. Abrasion indexes for the specified ore type were used to calculate the average life span of the crusher and mill liners.

21.10.3.5 Stores & Maintenance

The costs included for stores and maintenance are based on factors applied to the relevant capital supply rates. A 70:30 ratio was applied to the phased capital expenditure for phase 1A: phase 1B, based on similarly phased projects.

21.10.3.6 Ore Stockpile Management

It was assumed that a total of 30 000 tpm of ore would be reclaimed in Phase 1A, and an average of 10 000 tpm would be reclaimed in Phase 1B. A rate of US$2.09 /t was used for the stockpile movement, based on rates obtained from earth moving contractors.

21.11 TSF Operating Costs

The TSF operating cost has been developed by Geotail and is estimated to be US$ 587 273 per annum.

This number has been based on rand per ton value of US$ 0.155 per dry tonne deposited. This has been established by benchmarking the proposed TSF against simular Platinum tailings storage facilities currently being operated in South Africa

This estimate was based on a contractor operated facility opposed to an owner operated facility.

21.12 Laboratory Operating Costs

The laboratory operating cost estimate is based on steady state operating conditions of 2 Mtpa for Phase 1A and 4 Mtpa for Phase 1B. The laboratory operating cost estimate is summarised in Table 21.17.

Table 21.17 Laboratory Operational Cost Estimate

Description	Phase 1A 2 Mtpa (US$M)	Phase 1B 4 Mtpa (US$M)
Labour Cost	0.8	1.0
Process Plant Samples	0.4	0.7
Mining Samples	1.3	1.3
Environmental Samples	0.01	0.01
Maintenance Costs	0.07	0.07
Total Annual Cost	**2.6**	**3.2**
Total Unit Cost (US$/t)	1.31	0.80

Taking into account escalation for subsequent project phases in terms of staffing requirements, quantity of samples processed and maintenance requirements, the following laboratory costs are estimated:

Table 21.18 Phased Laboratory Operational Costs

Description	Phase 1A 2 Mtpa (US$M)	Phase 1B 4 Mtpa (US$M)
Total Annual Cost	**2.6**	**3.2**
Total Unit Cost (US$/t)	1.31	0.80

21.12.1 Basis of Laboratory Cost Estimate

The laboratory operating cost estimate is based on steady state operating conditions 4 Mtpa for Phase 1B. The estimate has been calculated based on the sample requirements as indicated by geology, mining and the processing plant. A base date of March 2014 was used.

21.12.2 Assays

A sampling matrix was generated to estimate the number of annual samples required daily for mining, plant and environmental purposes. Using a cost per sample type, this was converted into an annual variable laboratory cost. The laboratory fixed costs were then generated from the annual laboratory labour compliment costs and the laboratory maintenance costs.

21.13 Concentrate Transport

Concentrate transport costs were obtained from a reputable logistics company based on a transported distance of 350 km from site to the Rustenburg area. A rate of US$31.82 per ton of concentrate filtercake has been used for the estimate. The concentrate filter cake nominal moisture content of 12% (by mass) has been used along with the calculated mass pull required to maintain a concentrate grade of approximately 85 g/t 3PE+Au. A base date of October 2014 was used.

Table 21.19 below summarises the estimated annual concentrate transport costs for the Platreef Project concentrator plant Phase 1.

Table 21.19 Concentrate Transport Cost Estimate

Description	Unit	Phase 1A 2 Mtpa	Phase 1B 4 Mtpa
Annual Concentrate Transport	US$M	2.6	5.9
Total Unit Cost Excluding contingency	US$/t	**1.30**	**1.48**

Based on the assumption that mass pull will be similar to phase 1 in subsequent expansion phases, the following concentrate transport costs are predicted:

21.14 General and Administration Operating Costs

General and administrative costs are those costs necessary for sustaining the operation but that are not directly associated with the output of the desired product. General and administrative costs may include: licensing fees, insurances, costs for catering, security, employee transport services, communications costs, legal fees, corporate office management charges, stationery costs, donations, postage, petties and recruitment and relocation costs, bonuses.

The general and administration costs estimate is based on the variable and fixed costs as summarised in Table 21.20. The total general and administration costs are presented in Table 21.21.

Table 21.20 General and Administration Cost Assumptions

Description	Unit	G&A Costs
G&A Variable	US$/t	0.46
G&A Per Annum Fixed	$Mpa	12.4

Table 21.21 General and Administration Costs

LOM Opex Estimate	Unit	Total
Total G&A Costs	US$M	434
G&A Unit Costs	US$/t Milled	3.61

22 ECONOMIC ANALYSIS

The Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. This section has not been changed except to add additional price and cost sensitivities using the Platreef 2014 PFS results.

22.1 Summary of Financial Results

The key features of the Platreef 2014 PFS include:

- Development of a large, mechanised, underground mine is planned at an initial 4 Mtpa throughput scenario.
- Planned average annual production rate of 433 koz of platinum, palladium, rhodium, and gold (3PE+Au).
- Estimated pre-production capital requirement of approximately US$1.2 billion, including US$114 million in contingencies.
- After-tax Net Present Value (NPV) of US$972 million, at an 8% discount rate.
- After-tax Internal Rate of Return (IRR) of 13%.
- The Platreef 2014 PFS 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 2014 PFS are shown in Table 22.1.

Figure 22.1 Mining Production

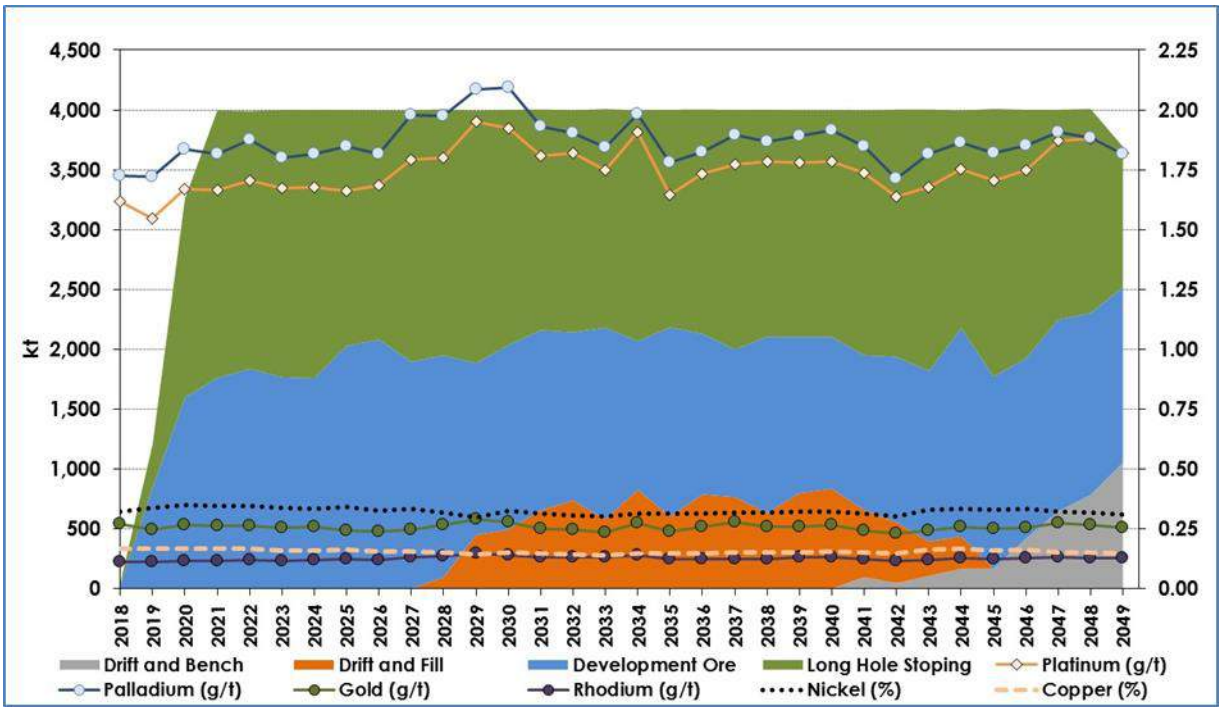


Figure 22.2 Cumulative Cash Flow After Tax

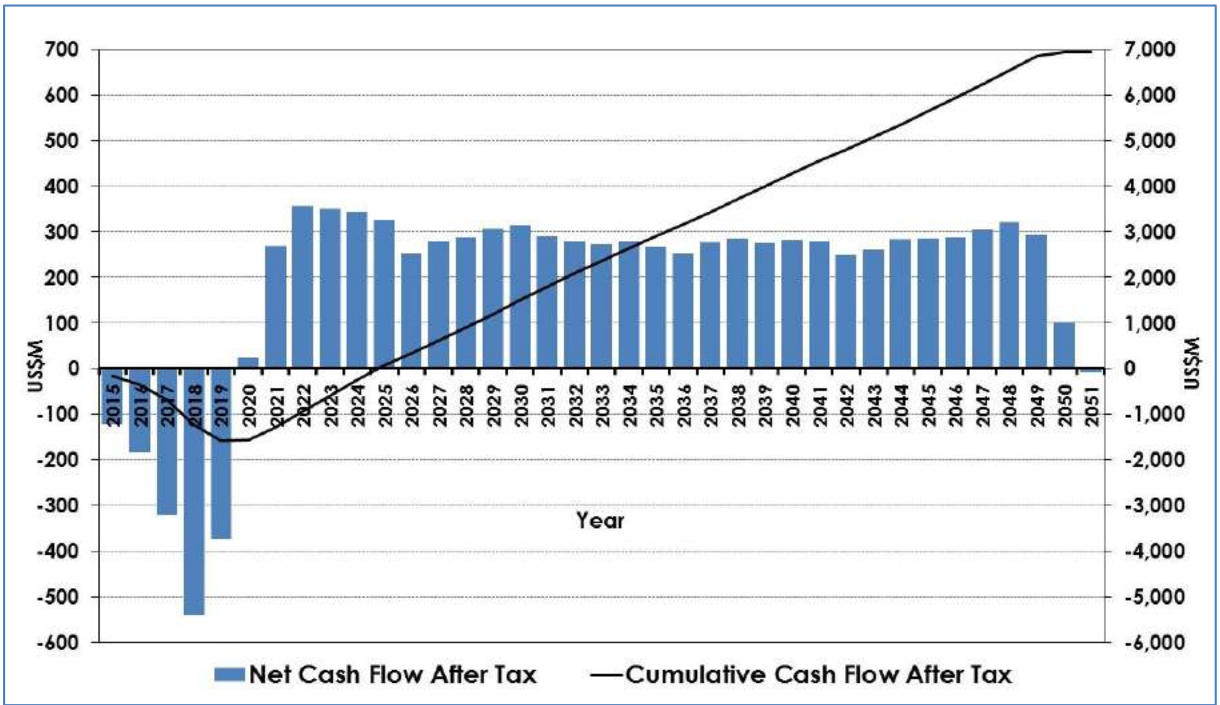


Table 22.1 Platreef 2014 PFS Results

Item	Units	Total
Mined and Processed		
Mineral Reserve	Mt	120
Platinum	g/t	1.76
Palladium	g/t	1.87
Gold	g/t	0.26
Rhodium	g/t	0.13
3PE+Au	g/t	4.02
Copper	%	0.15
Nickel	%	0.32
Concentrate Produced		
Concentrate	kt	4,915
Platinum	g/t	37.5
Palladium	g/t	39.8
Gold	g/t	4.8
Rhodium	g/t	2.8
3PE+Au	g/t	85
Copper	%	3.3
Nickel	%	5.4
Recovered Metal		
Platinum	koz	5,927
Palladium	koz	6,295
Gold	koz	761
Rhodium	koz	448
3PE+Au	koz	13,431
Copper	Mlb	358
Nickel	Mlb	588
Key financial results		
Life of Mine	years	31
Pre-Production Capital	US$M	1,168
Mine Site Cash Cost	US$/oz 3PE+Au	401
Total Cash Costs After Credits	US$/oz 3PE+Au	322
Site Operating Costs	US$/t Milled	44.86
After Tax NPV8	US$M	972
After Tax IRR	%	13
Project Payback Period	years	7

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.
3. 3PE+Au = (Pt + Pd + Rh) + Au (g/t)
4. Metal prices used in the reserve estimate are as follows.US$1,699/oz Pt, US$667/oz Pd, US$1,315/oz Au, US$1,250/oz Rh, US$8.81/lb Ni, US$2.73/lb Cu
5. A declining Net Smelter Return (NSR) cut-off of US$100/t–$80/t was used in the mineral reserve estimates.
6. Metal price assumptions used for the base case economic analysis are: US$1,630/oz Pt, US$815/oz Pd, US$1,300/oz Au, US$2,000/oz Rh, US$8.90/lb Ni, US$3.00/lb Cu.
7. Tonnage and grade estimates include dilution and recovery allowances.

8. Totals may not add due to the rounding.

22.2 Model Assumptions

22.2.1 Pricing and Discount Rate Assumptions

The Platreef Project level financial model begins on 1 January 2015. It is presented in 2015 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 economic analysis uses price assumptions of US$1,630/oz Pt, US$815/oz Pd, US$1,300/oz Au, US$2,000/oz Rh, US$8.90/lb Ni, and US$3.00/lb Cu. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. Realisation costs are described in Section 19. Costs estimated in ZAR have been converted to US$ at an exchange rate of 11 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$/oz	1,630
Palladium	US$/oz	815
Gold	US$/oz	1,300
Rhodium	US$/oz	2,000
Nickel	US$/lb	8.90
Copper	US$/lb	3.00
Base Metals Refining Charge	% 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 * 12.5)] * 100, with a maximum of 5%, for production of refined 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 $45M.

22.4 Project Results

The results of the financial analysis show an After Tax NPV8% of US$972M. The case exhibits an after tax IRR of around 13% and a payback period of around seven years. The estimates of cash flows have been prepared on a real basis as at 1 January 2015 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 and Figure 22.3.

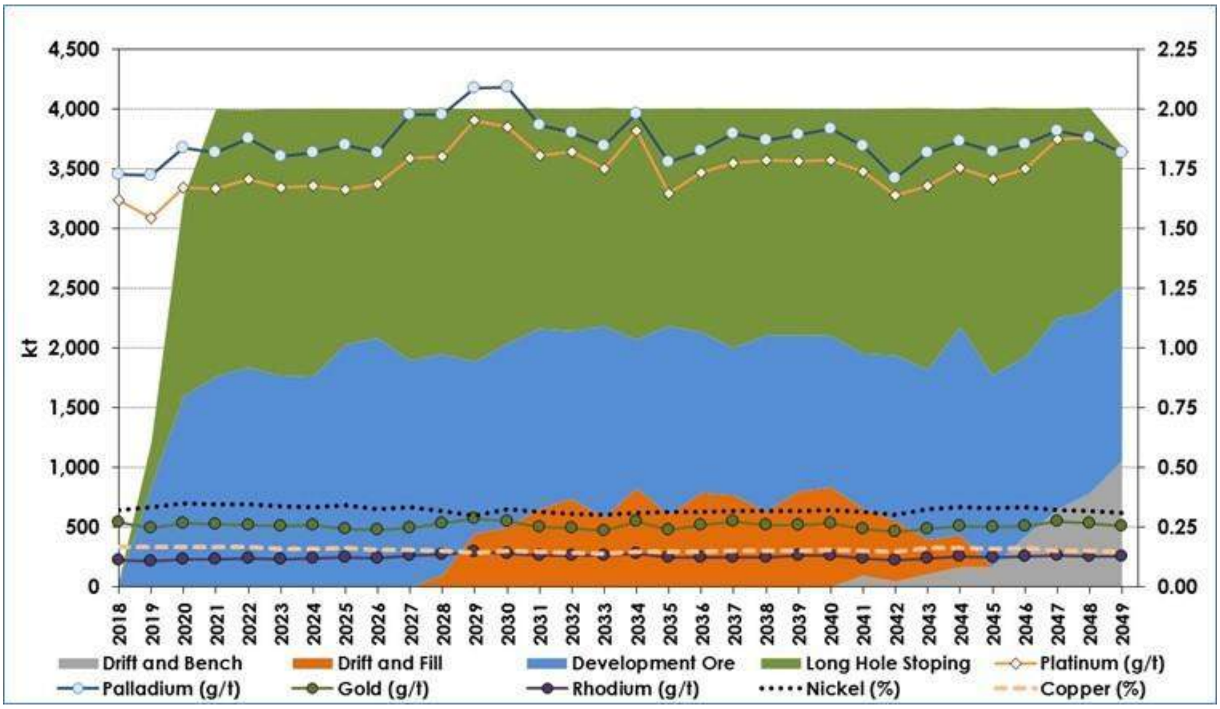
Table 22.3 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	9,619	6,981
	5.0%	3,024	2,113
	8.0%	**1,491**	**972**
	10.0%	885	519
	12.0%	473	210
	15.0%	80	-86
	20.0%	-254	-336
Internal Rate of Return		15%	13%
Project Payback Period (Years)		7	7

Table 22.4 Mining Production Statistics

	Unit	TOTAL LOM	5 YR AVG	10 YR AVG	LOM AVG
Ore Production					
Mineral Reserve	Mt	120	3.29	3.64	3.88
Ore Milled	Mt	120	3.29	3.64	3.88
Platinum	g/t	1.76	1.67	1.70	1.76
Palladium	g/t	1.87	1.82	1.86	1.87
Gold	g/t	0.26	0.26	0.25	0.26
Rhodium	g/t	0.13	0.12	0.12	0.13
3PE + Au	g/t	4.02	3.87	3.93	4.02
Copper	%	0.15	0.17	0.16	0.15
Nickel	%	0.32	0.34	0.34	0.32
Recoveries					
Platinum	%	87.2	86.1	86.5	87.2
Palladium	%	86.9	86.5	86.8	86.9
Gold	%	76.7	76.9	76.6	76.7
Rhodium	%	92.0	91.8	91.9	92.0
Copper	%	87.7	87.6	87.6	87.7
Nickel	%	68.8	69.4	69.2	68.8
Concentrate Produced					
Concentrate	kt	4,915	129	145	159
Platinum	g/t	37.51	36.78	36.87	37.51
Palladium	g/t	39.84	40.39	40.44	39.84
Gold	g/t	4.81	5.11	4.89	4.81
Rhodium	g/t	2.83	2.72	2.80	2.83
3PE + Au	g/t	85.00	85.00	85.00	85.00
Copper	%	3.30	3.71	3.54	3.30
Nickel	%	5.42	6.10	5.84	5.42
Recovered Metal					
Platinum	koz	5,927	152	172	191
Palladium	koz	6,295	167	189	203
Gold	koz	761	21	23	25
Rhodium	koz	448	11	13	14
3PE + Au	koz	13,431	351	397	433
Copper	Mlb	358	11	11	12
Nickel	Mlb	588	17	19	19

Figure 22.3 Mining Production



22.5 Capital and Operating Cost Summary

Mine site cash costs are summarised in Table 22.5. The revenues and operating costs are presented in Table 22.6.

The higher nickel and copper grades contribute to lower operating cash costs for the Northern Limb as illustrated by Figure 22.4. Among the current and future Northern Limb producers, Platreef's estimated cash cost of US$322 per 3PE+Au ounce, 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 Payable 3PE+Au		
	Life-of-Mine Average	**5-Year Average**	**10-Year Average**
Mine Site Cash Cost	401	454	429
Realisation	390	384	390
Total Cash Costs Before Credits	**792**	**838**	**819**
Nickel Credits	389	438	419
Copper Credits	80	90	86
Total Cash Costs After Credits	**322**	**310**	**314**

Figure 22.4 Cash Cost Comparison Platreef and 2013 Producers

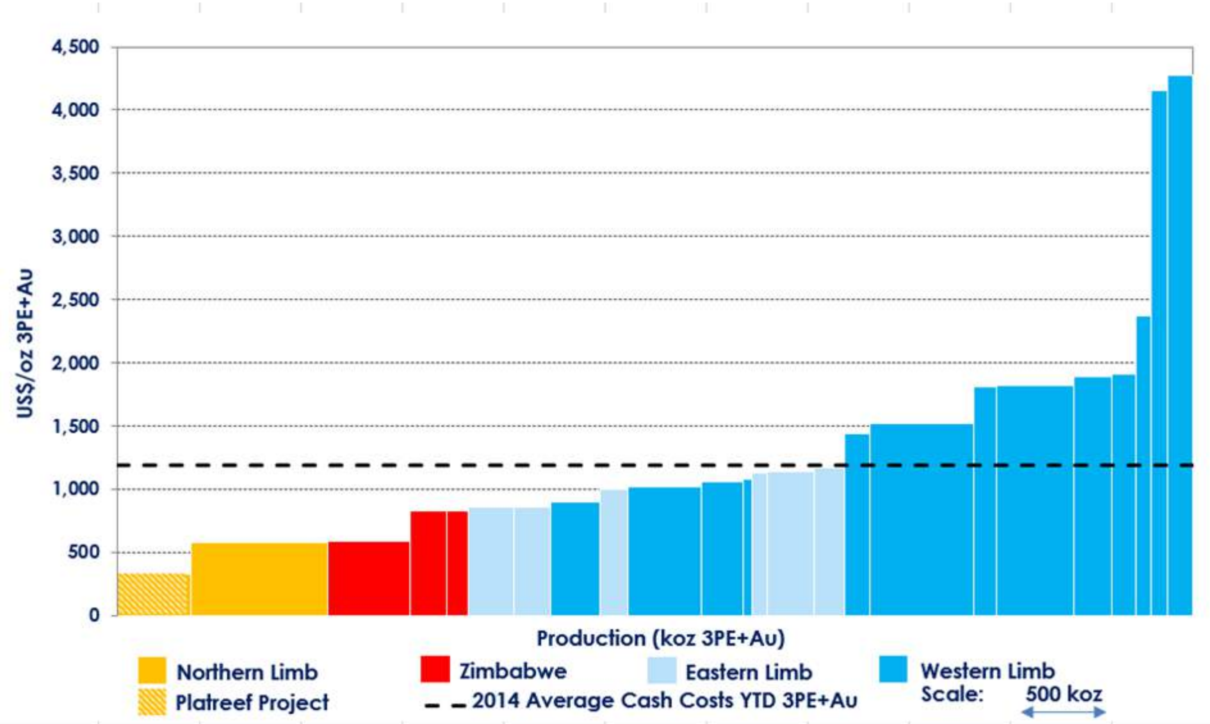


Figure by OreWin Producer cost data Source: SFA (Oxford).

Table 22.6 Operating Costs and Revenues

	Life of Mine TOTAL US$M	US$/t milled		
		5-Year Average	10-Year Average	Life of Mine Average
Gross Sales Revenue	22,981	188.18	189.54	191.19
Less: Realisation Costs				
Transport Costs	195	1.55	1.59	1.63
Treatment & Refining Charges	4,137	33.87	34.12	34.41
Royalties	908	5.61	6.82	7.55
Total Realisation Costs	**5,240**	**41.03**	**42.52**	**43.59**
Net Sales Revenue	**17,742**	**147.15**	**147.02**	**147.60**
Site Operating Costs				
Mining	3,585	32.71	31.44	29.83
Processing & Tailings	1,372	11.88	11.58	11.42
General & Administration	434	3.86	3.70	3.61
Total	**5,392**	**48.45**	**46.72**	**44.86**
Operating Margin	**12,349**	**98.70**	**100.30**	**102.74**
Operating Margin (%)	**54%**	**52%**	**53%**	**54%**

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			
Underground	542	956	1,498
Surface Infrastructure	64	-	64
Backfill Plant	21	14	34
Capitalised Operating Costs	35	-	35
Subtotal	**661**	**970**	**1,631**
Processing & Tailings			
Concentrator	93	181	274
Rietfontein TSF	30	30	59
Subtotal	**123**	**211**	**334**
Infrastructure			
General Infrastructure	115	63	178
Site Pre-Production	8	1	9
Closure Costs	-	18	18
Subtotal	**123**	**83**	**206**
Indirects			
Exploration & Geology	3	0.4	4
Engineering Procurement Contract Management (EPCM)	59	17	75
Capitalised G&A & Other Costs	15	6	21
Subtotal	**77**	**23**	**100**
Owners Cost	71	4	75
Capex Before Contingency	**1,054**	**1,291**	**2,345**
Contingency	114	110	224
Capex After Contingency	**1,168**	**1,401**	**2,569**

1. Sustaining capital expenditure also includes 2019 construction capital expenditure
2. Totals vary due to rounding.

22.6 Project Cash Flows

Cumulative cash flow after tax is depicted in Figure 22.5 and a complete cash flow is provided in Table 22.8.

Figure 22.5 Cumulative Cash Flow After Tax

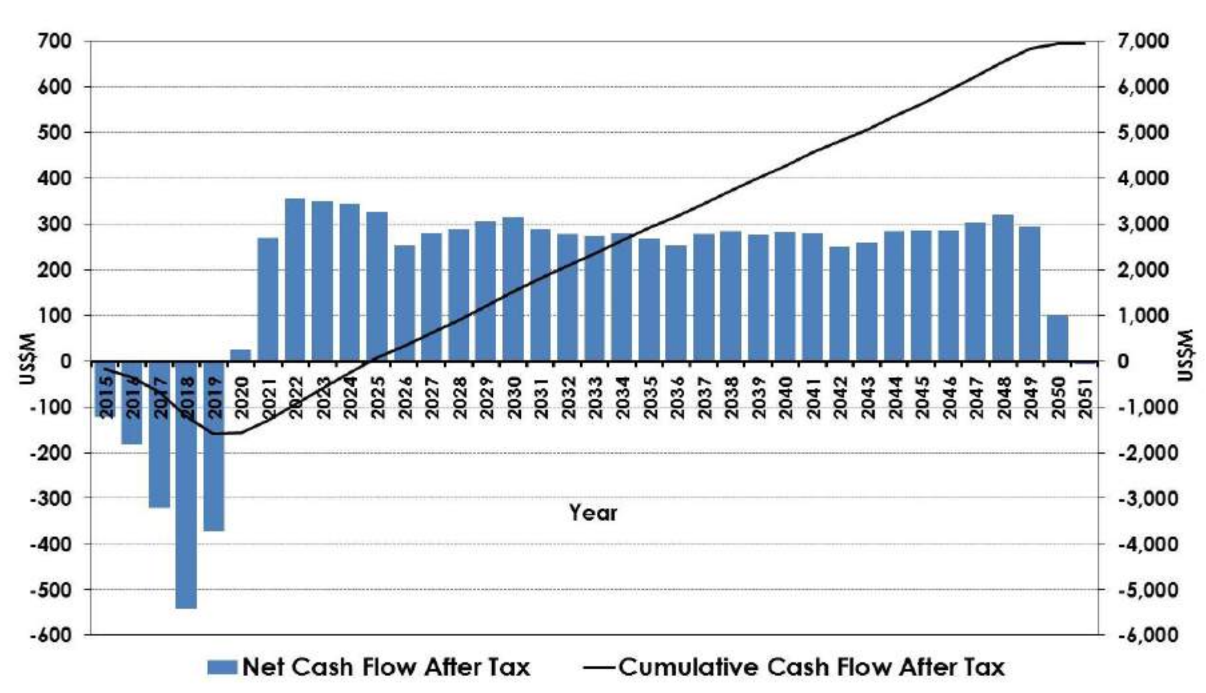


Table 22.8 Cumulative Cash Flow

US$M	YEAR												TOTAL
Description	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6-10**	**11-20**	**21-LOM**	
Gross Revenue	-	-	-	-	214	612	755	769	746	3,813	7,743	8,329	22,981
Realisation Costs	-	-	-	-	40	123	167	175	170	875	1,778	1,911	5,240
Net Sales Revenue	**-**	**-**	**-**	**-**	**174**	**489**	**588**	**594**	**576**	**2,938**	**5,965**	**6,418**	**17,742**
Site Operating Costs													-
Mining	-	-	-	-	67	97	124	126	124	608	1,218	1,222	3,585
Processing & Tailings	-	-	-	-	20	39	45	45	45	227	453	497	1,372
General & Administration	-	-	-	-	7	14	14	14	14	71	143	157	434
Total Site Operating Costs	**-**	**-**	**-**	**-**	**95**	**150**	**183**	**185**	**183**	**906**	**1,814**	**1,876**	**5,392**
Operating Surplus / (Deficit)	**-**	**-**	**-**	**-**	**79**	**338**	**405**	**408**	**393**	**2,032**	**4,151**	**4,542**	**12,349**
Indirect Costs	-	-	-	-	1,354	281	168	105	102	291	240	239	2,779
Net Profit Before Income Tax	**-**	**-**	**-**	**-**	**-1,275**	**58**	**237**	**303**	**291**	**1,741**	**3,912**	**4,303**	**9,570**
Income Tax Expense	-	-	-	-	-	-	-	-	-	333	1,095	1,210	2,638
Net Profit After Income Tax	**-**	**-**	**-**	**-**	**-1,275**	**58**	**237**	**303**	**291**	**1,408**	**2,816**	**3,093**	**6,932**
Capital Expenditure	-122	-184	-320	-542	-421	-245	-108	-45	-42	-177	-183	-180	-2,569
Depreciation Less Working Capital	-	-	-	-	1,322	212	140	97	100	257	189	300	2,617
Net Cash Flow After Tax	**-122**	**-184**	**-320**	**-542**	**-373**	**25**	**269**	**355**	**349**	**1,489**	**2,822**	**3,213**	**6,981**

22.7 Price Sensitivity Analysis

A price sensitivity to platinum and nickel prices analysis of the NPV8% was performed on the financial model and the results are shown in Table 22.9, Table 22.10, Figure 22.6 and Figure 22.7. Platinum and nickel are the major contributors to project revenue.

Table 22.9 After Tax NPV Platinum and Nickel Price Sensitivity

After Tax NPV8 US$M	Platinum Price - US$/lb				
Nickel Price - US / lb	1,330	1,530	1,630	1,730	1,930
7.40	572	748	837	924	1,099
8.40	663	839	927	1,014	1,189
8.90	708	884	972	1,059	1,234
9.40	753	929	1,017	1,104	1,279
10.40	844	1,019	1,106	1,194	1,369

Table 22.10 After Tax NPV Sensitivity to Platinum and Nickel Price Proportional Change

After Tax NPV8 US$M	Platinum Price - US$/lb				
Nickel Price - US / lb	1,330	1,530	1,630	1,730	1,930
7.40	-41%	-23%	-14%	-5%	13%
8.40	-32%	-14%	-5%	4%	22%
8.90	-27%	-9%	0%	9%	27%
9.40	-22%	-4%	5%	14%	32%
10.40	-13%	5%	14%	23%	41%

Figure 22.6 Platinum Price Sensitivity

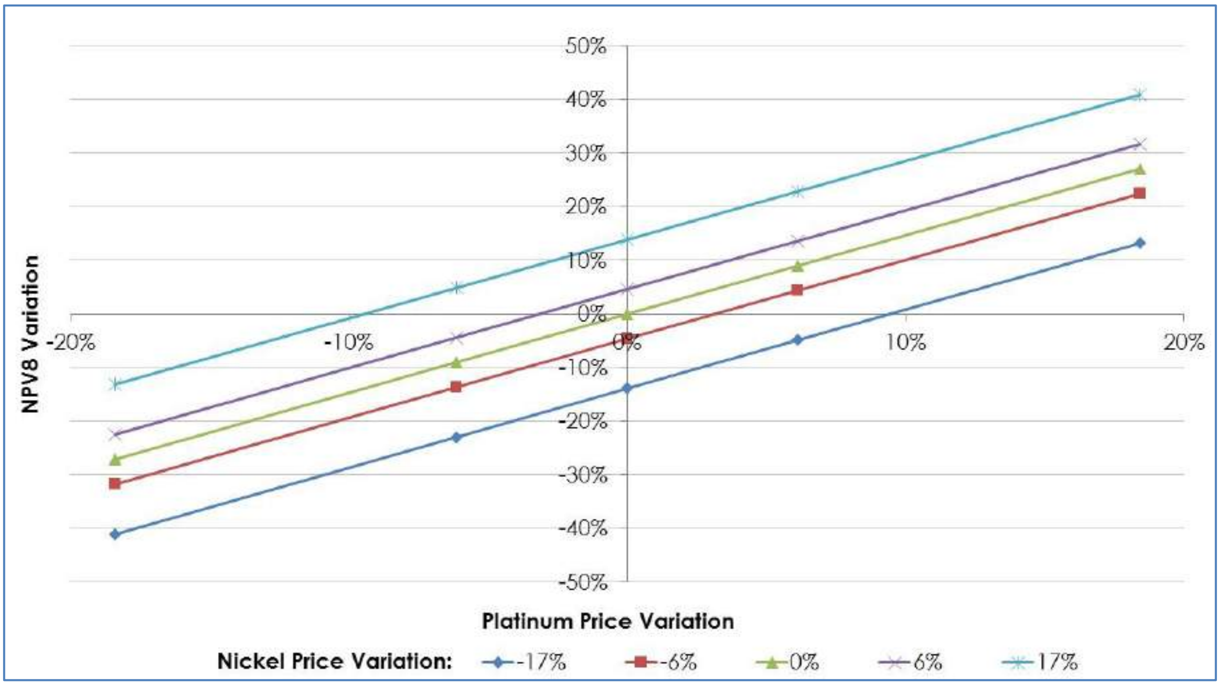


Figure by OreWin 2014.

Figure 22.7 Nickel Price Sensitivity

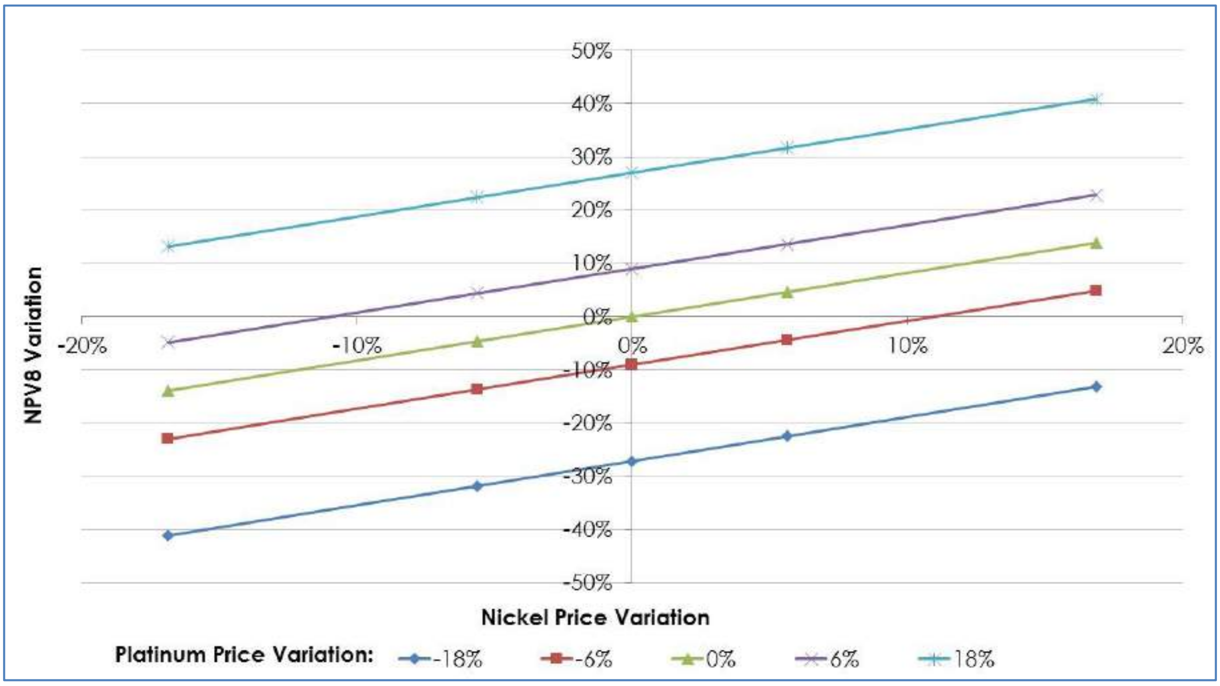


Figure by OreWin 2014.

22.8 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 R11 to US$1. A currency exchange rate sensitivity analysis was performed on the financial model and the results are shown in Table 22.11 and Figure 22.8. The analysis indicates that an increase in the exchange rate to 12 Rand/US$ results in a 9% increase in the After Tax NPV8.

Table 22.11 Exchange Rate Sensitivity

South African Rand to US$	**7.00**	**8.00**	**9.00**	**10.00**	**11.00**	**12.00**	**13.00**	**14.00**	**15.00**
After Tax NPV8% US$M	395	595	749	872	**972**	1,055	1,125	1,186	1,238
Difference US$M	-577	-377	-223	-100	-	83	154	214	266
Difference %	-59	-39	-23	-10	-	9	16	22	27

Figure 22.8 Exchange Rate Sensitivity

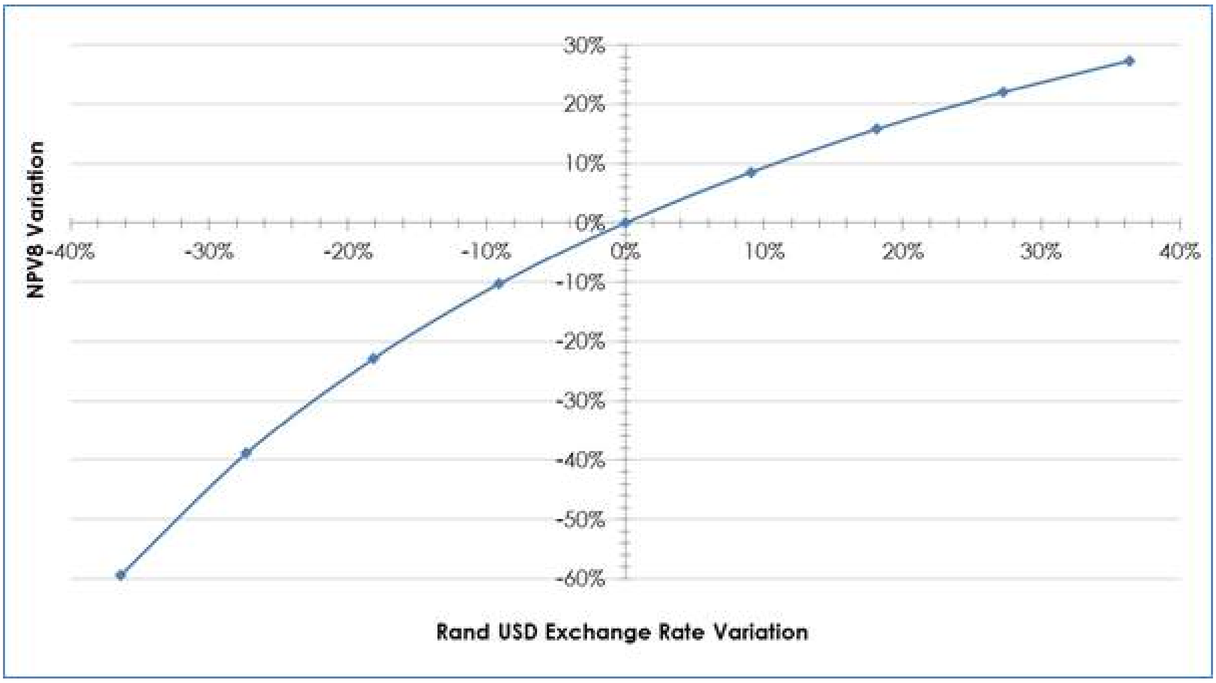


Figure by OreWin 2014.

22.9 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$1,212 million at a 25% decrease in the pre-production capital cost and the After Tax NPV8% of US$732 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 22.12 and Figure 22.9.

Table 22.12 Pre-production Capital Cost Sensitivity

Pre-production Capital Cost	-25%	-10%	0%	10%	25%
After Tax NPV8% US$M	1,212	1,068	**972**	876	732
Difference US$M	240	96	**-**	-96	-240
Difference %	25	10	-	-10	-25

Figure 22.9 Exchange Rate Sensitivity

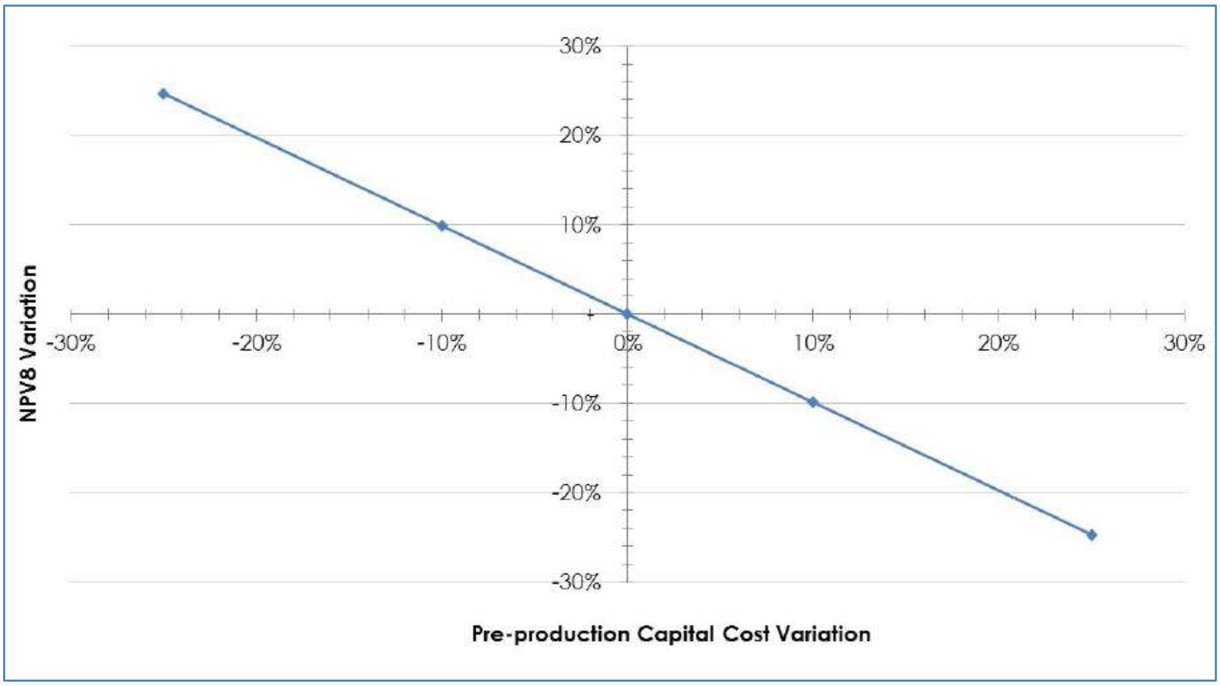


Figure by OreWin 2014.

The sustaining capital cost sensitivity analysis shows the After Tax NPV8% of US$1,149 million at a 25% decrease in the sustaining capital cost and the After Tax NPV8% of US$795 million at a 25% increase in the sustaining capital cost. The results of the sustaining capital cost sensitivity analysis are shown in Table 22.13 and Figure 22.10.

Table 22.13 Sustaining Capital Cost Sensitivity

Sustaining Capital Cost	**-25%**	**-10%**	**0%**	**10%**	**25%**
After Tax NPV8% US$M	1,149	1,042	**972**	901	795
Difference US$M	177	71	**-**	-71	-177
Difference %	18	7	-	-7	-18

Figure 22.10 Sustaining Capital Cost Sensitivity

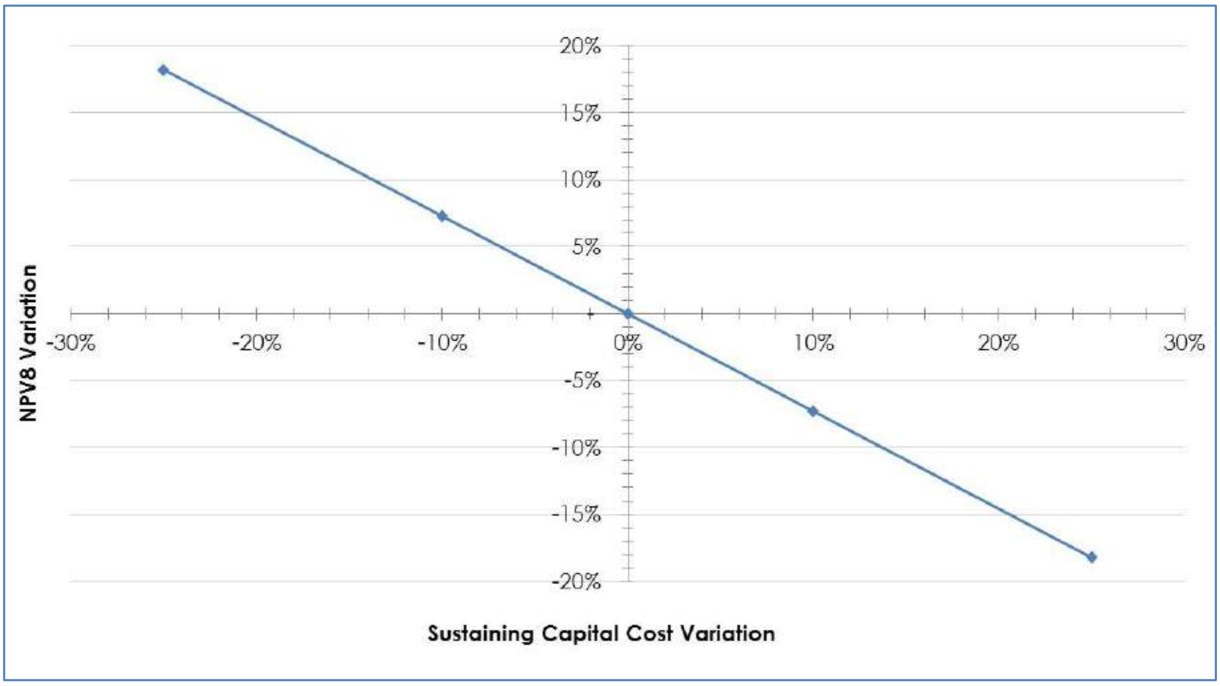


Figure by OreWin 2014.

The operating cost sensitivity analysis shows the After Tax NPV8% of US$1,249 million at a 25% decrease in the operating cost and the After Tax NPV8% of US$697 million at a 25% increase in the operating cost. The results of the operating cost sensitivity analysis are shown in Table 22.14 and Figure 22.11.

Table 22.14 Operating Cost Sensitivity

Operating Cost	**-25%**	**-10%**	**0%**	**10%**	**25%**
After Tax NPV8% US$M	1,249	1,082	**972**	861	697
Difference US$M	278	110	**-**	-110	-275
Difference %	29	11	-	-11	-28

Figure 22.11 Operating Cost Sensitivity

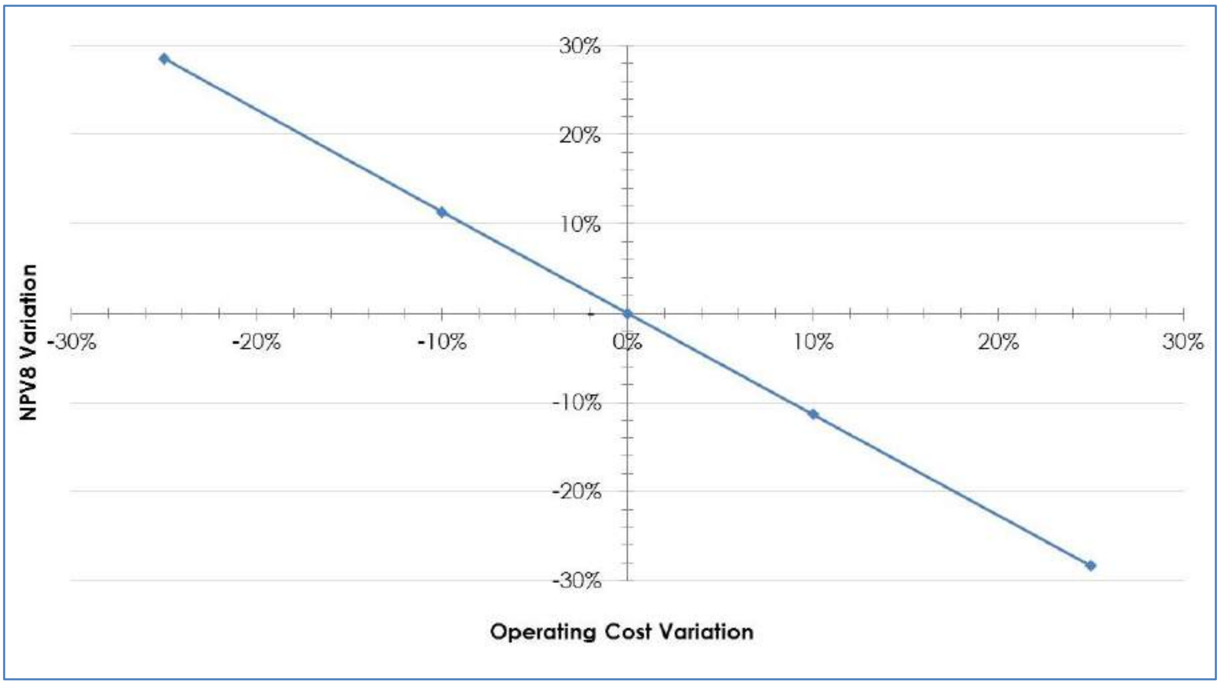


Figure by OreWin 2014.

22.10 Additional Sensitivity Analysis

The Platreef 2014 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS. This section has not been changed except to add additional price and cost sensitivities using the Platreef 2014 PFS results. Since the completion of the Platreef 2014 PFS there have been reductions in the spot and long term metal prices, exchange rates and costs. Changes to the exchange rate and Chinese pricing of materials have indicated potential cost reductions, while some costs have remained stable there are fewer costs items that have increased during the period since the Platreef 2014 PFS was completed. In order to analyse potential changes a sensitivity using modified metal prices and a 5% decrease in costs was prepared. The results of this demonstrated that the Platreef 2014 PFS Mineral reserve was still viable.

The key economic assumptions for both the Platreef 2014 PFS and the additional sensitivity base analysis are shown in Table 22.15. The results of the additional sensitivity base analysis indicate that the After Tax NPV8 is $511 M. The sensitivity of the results for a 55 decrease in costs and a range of nickel and platinum prices is shown in Table 22.16 and graphically in Figure 22.12.

Table 22.15 Sensitivity and Platreef 2014 PFS Economic Assumptions

Parameter	Unit	Platreef 2014 PFS Assumptions	Additional Sensitivity Base Assumptions
Platinum	US$/oz	1,630	1,222
Palladium	US$/oz	815	761
Gold	US$/oz	1,300	1,235
Rhodium	US$/oz	2,000	1,097
Nickel	US$/lb	8.90	7.67
Copper	US$/lb	3.00	2.83
Base Metals Refining Charge	% Gross Sales	18%	18%
Precious Metals Refining Charge	% Gross Sales	18%	18%
Change in Costs	%	100%	95%

Table 22.16 Metal Price Sensitivity for 5% Decrease in Costs

After Tax NPV8 US$M	Platinum Price - US / oz					
Nickel Price - US / lb	**800**	**1,000**	**1,222**	**1,422**	**1,630**	**1,800**
4.50	-166	17	218	398	583	734
5.50	-71	110	311	490	675	824
6.50	22	204	403	581	765	915
7.67	132	313	511	688	872	1,021
8.00	162	343	540	717	901	1,050
8.90	246	426	622	799	982	1,131
9.90	339	517	713	890	1,072	1,221

Figure 22.12 Metal Price Sensitivity for 5% Decrease in Costs

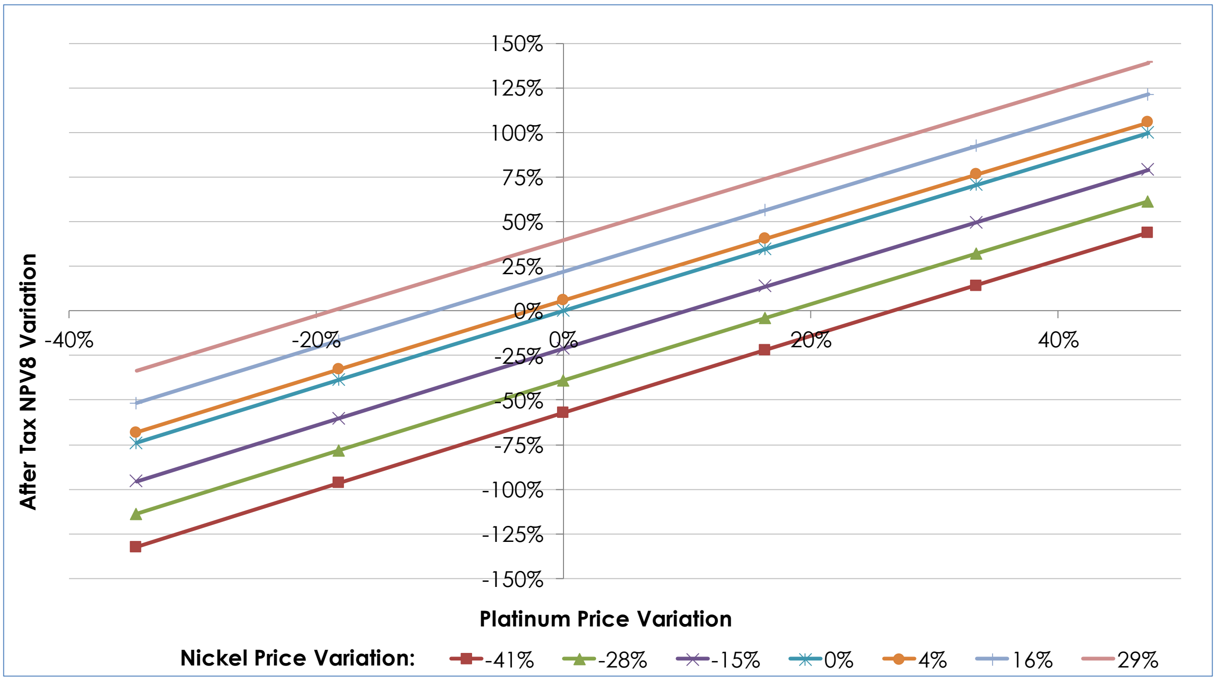


Figure by OreWin 2016.

22.11 Comparison to Platreef 2014 PEA

A comparison of the Platreef 2014 PFS and Platreef 2014 PEA financial models was carried out. 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.

The PFS estimates of cash flows were prepared on a real basis as at 1 January 2015 to calculate NPV and the PEA estimates cash flows were prepared on a real basis as at 1 January 2014. 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 PFS accuracy. Some delays were experienced in commencing the sinking of Shaft 1 and there was some 2014 expenditure planned that was not incurred. Although the comparison between the two studies in not an exact match, the analysis indicates that the Platreef 2014 PFS and the Platreef 2014 PEA are similar. The after tax financial results in 2014 PFS and 2014 PEA is shown in Table 22.17.

Table 22.17 After Tax Financial Results Comparison

	Discount Rate	2014 PFS	2014 PEA
Net Present Value ($US M's)	Undiscounted	6,981	6,992
	5.0%	2,113	2,040
	8.0%	**972**	**897**
	10.0%	519	449
	12.0%	210	149
	15.0%	-86	-133
Internal Rate of Return		13%	13%
Project Payback Period (Years)		7	6

The 2014 PFS and 2014 PEA 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 2014 PFS and 2014 PEA is presented in Table 22.18

Table 22.18 Metal Prices Comparison

Parameter	Unit	2014 PFS	2014 PEA
Platinum	US$/oz	1,630	1,700
Palladium	US$/oz	815	820
Gold	US$/oz	1,300	1,300
Rhodium	US$/oz	2,000	1,700
Copper	US$/lb	3.00	3.00
Nickel	US$/lb	8.90	8.35

The key production summary comparison is shown in Table 22.19. A comparison of the Pre-Production Capital Costs, Sustaining Capital Costs and total Capital costs in 2014 PFS and 2014 PEA are shown in Table 22.17 to Table 22.19

A comparison of the mine site cash costs in 2014 PFS and 2014 PEA is summarised in Table 22.23

Table 22.19 Production Summary Comparison

Item	Unit	2014 PFS	2014 PEA
Ore Production			
Mineral Reserve	Mt	120	117
Ore Milled	Mt	120	117
Platinum	g/t	1.76	1.84
Palladium	g/t	1.87	1.93
Gold	g/t	0.26	0.27
Rhodium	g/t	0.13	0.13
3PE + Au	g/t	4.02	4.17
Copper	%	0.15	0.16
Nickel	%	0.32	0.34
Recoveries			
Platinum	%	87.2	88.2
Palladium	%	86.9	87.6
Gold	%	76.7	76.7
Rhodium	%	92.0	85.9
Copper	%	87.7	87.9
Nickel	%	68.8	69.1
Concentrate Produced			
Concentrate	kt	4,915	4,665
Platinum	g/t	37.51	40.50
Palladium	g/t	39.84	42.41
Gold	g/t	4.81	5.26
Rhodium	g/t	2.83	2.76
3PE + Au	g/t	85.00	90.92
Copper	%	3.30	3.57
Nickel	%	5.42	5.82
Contained Metal			
Platinum	koz	5,927	6,075
Palladium	koz	6,295	6,362
Gold	koz	761	789
Rhodium	koz	448	413
3PE + Au	koz	13,431	13,638
Copper	Mlb	358	368
Nickel	Mlb	588	599

Table 22.20 Pre-Production Capital Costs comparison

Item	Unit	2014 PFS	2014 PEA
Mining			
Underground	US$M	542	595
Surface Infrastructure	US$M	64	-
Backfill Plant	US$M	21	-
Capitalised Operating Costs	US$M	35	24
Subtotal	**US$M**	**661**	**619**
Processing & Tailings			
Concentrator	US$M	93	201
Rietfontein TSF	US$M	30	39
Subtotal	**US$M**	**123**	**241**
Infrastructure			
General Infrastructure	US$M	115	105
Site Pre-Production	US$M	8	-
Subtotal	**US$M**	**123**	**105**
Indirects			
Exploration & Geology	US$M	3	-
EPCM	US$M	59	117
Capitalised G&A & Other Costs	US$M	15	26
Subtotal	**US$M**	**77**	**143**
Owners Cost	US$M	71	77
Capex Before Contingency	**US$M**	**1,054**	**1,185**
Contingency	US$M	114	340
Capex After Contingency	**US$M**	**1,168**	**1,525**

Table 22.21 Sustaining Capital Costs comparison

Item	Unit	2014 PFS	2014 PEA
Mining			
Underground	US$M	956	679
Backfill Plant	US$M	14	-
Subtotal	**US$M**	**970**	**679**
Processing & Tailings			
Concentrator	US$M	181	103
Rietfontein TSF	US$M	30	-
Subtotal	**US$M**	**211**	**103**
Infrastructure			
General Infrastructure	US$M	63	12
Site Pre-Production	US$M	1	-
Closure Costs	US$M	18	14
Subtotal	**US$M**	**83**	**26**
Indirects			
Exploration & Geology	US$M	0.4	-
EPCM	US$M	17	4
Capitalised G&A & Other Costs	US$M	6	-
Subtotal	**US$M**	**23**	**4**
Owners Cost	US$M	4	2
Capex Before Contingency	**US$M**	**1,291**	**814**
Contingency	US$M	110	160
Capex After Contingency	**US$M**	**1,401**	**974**

Table 22.22 Total Capital Costs comparison

Item	Unit	2014 PFS	2014 PEA
Mining			
Underground	US$M	1,498	1,274
Surface Infrastructure	US$M	64	-
Backfill Plant	US$M	34	-
Capitalised Operating Costs	US$M	35	24
Subtotal	**US$M**	**1,631**	**1,298**
Processing & Tailings			
Concentrator	US$M	274	305
Rietfontein TSF	US$M	59	39
Subtotal	**US$M**	**334**	**344**
Infrastructure			
General Infrastructure	US$M	178	116
Site Pre-Production	US$M	9	-
Closure Costs	US$M	18	14
Subtotal	**US$M**	**206**	**130**
Indirects			
Exploration & Geology	US$M	4	-
EPCM	US$M	75	121
Capitalised G&A & Other Costs	US$M	21	26
Subtotal	**US$M**	**100**	**148**
Owners Cost	US$M	75	79
Capex Before Contingency	**US$M**	**2,345**	**1,998**
Contingency	US$M	224	500
Capex After Contingency	**US$M**	**2,569**	**2,499**

Table 22.23 Cash Costs Comparison

	Unit	LOM AVG	
		2014 PFS	2014 PEA
Mine Site Cash Cost	$ / oz Payable 3PE+Au	401	412
Realisation	$ / oz Payable 3PE+Au	390	402
Total Cash Costs Before Credits	**$ / oz Payable 3PE+Au**	**792**	**814**
Nickel Credits	$ / oz Payable 3PE+Au	389	367
Copper Credits	$ / oz Payable 3PE+Au	80	81
Total Cash Costs After Credits	**$ / oz Payable 3PE+Au**	**322**	**367**

22.8 Project Optimisation Planning

Ivanhoe has retained Whittle Consulting (Whittle) of Melbourne, Australia to conduct an optimisation study using as a base the Platreef 2014 PFS production schedules, revenues and costs. The work has progressed but is yet to be finalised. The recommendations from the study are intended to provide guidance for the feasibility study.

The study uses of Whittle's proprietary Enterprise Optimisation methodology to focus on the financial metrics and integrate with the SUSOP process offered by JKTech of Brisbane, Australia to investigate the sustainability of proposed projects.

The Enterprise Optimisation methodology is used to optimise the major elements of the Platreef value chain in order to maximise project NPV. JKTech's SUSOP approach is to assess the implications of the optimised financial solution on the manufactured, social, human and natural capital of a project and make comparisons with the initial base case. The combination of the Whittle/JKTech will allow efficient comparison of multiple project scenarios that consider environmental, community and social issues in order to provide conclusions and recommendations for the sustainable development of Platreef to the benefit of stakeholders.

The mine designs from the Platreef 2014 PFS have been modified using alternative cut-off grades to create alternative shapes. Whittle have developed an iteration process to test combinations of cut-off grade and produce revised mining and processing schedules.

The opportunities identified for study from the Whittle work are:

- Net value per bottleneck unit to identify the location of the most profitable ore;
- Bring forward revenue and cash flow using enhanced scheduling techniques;
- Examination of initial smaller scale operation to bring forward the start date of operations and generate earlier cash flow;
- Analysis of dynamic grind to increase processing rates and dynamic mass pull to flotation concentrate to increase recovery to increase early cash flow;
- Increased cut-off grade in the initial years to increase revenue and cash flow;
- Alternative use of the planned mining infrastructure.

The SUSOP process has identified the opportunities to strengthen workforce performance, assisting in developing local enterprises, improving health facilities and road safety.

Two specific scenarios that have been identified so far in the optimisation study are:

- Conversion of the development shaft to a production shaft to supplement the main shaft and increase the overall mining capacity with relatively small capital cost implications. By matching the ore processing capacity of the concentrator to the total shaft hoisting capacity throughput and recovery could be increased relative to the Platreef 2014 PFS production.
- Construction of an initial smaller scale operation followed by a second similar capacity phase could reduce peak funding requirements and maintain NPV relative to the Platreef 2014 PFS.

23 ADJACENT PROPERTIES

This section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

25.1 Platreef 2014 PFS

The Platreef 2016 Resource Technical Report provides an update of the Platreef Project Mineral Resource, with the Mineral Reserve from the Platreef 2014 Prefeasibility Study (Platreef 2014 PFS) remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Platreef 2014 PFS.

The Platreef 2014 PFS presents the Mineral Reserve for Phase 1 of the Platreef development. Further work and studies should be undertaken to bring the Phase 1 to a Feasibility Study level. Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of Phases 2 and 3 will be evaluated at a later date, and the decision to expand can be deferred or brought forward as markets dictate and funding permits.

25.2 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) totaling 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. 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 km2 of exploration targets and an additional 48 km2 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 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 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 recognized a definitative 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 and the NSR values assigned to the model 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 long lead times for delivery of fixed and mobile mining equipment.
- There could be a lack of available work force with sufficient skills to meet the specified performance rates at the stated labour rates.
- Expanding the bulk sampling program during Shaft No. 1 sinking will result in an extended preproduction schedule and associated cost.
- There could be ground conditions below 1,100 m that limit shaft depths.
- There is limited underground surge capacity.
- Additional ore passes may be required for transfer of production to truck haulage levels.
- There is a potential of sterilizing lower grade resources by initial mining of higher grade resources.
- The ore and waste passes may not last until the end of the mine's life.

25.4.2 Opportunities and Recommendations

The following is a list of potential opportunities Platreef may experience:

- In the current design, Shaft No. 2 has a 6 Mtpa capacity. This, combined with the conversion of Shaft No. 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).
- Optimisation of stope access designs to reduce the amount of waste development.
- Continued optimisation of the stoping sequence to improve the grade profile during the early years of production.
- Automation of production LHDs and trucks.
- Remote operation of fixed rock breakers at ore and waste truck dumps.
- Potential for a second underground crusher and dedicated conveyor to transport ore from deeper resources to Shaft No. 2 for hoisting to surface.
- Potential to incorporate electric-powered mobile equipment in an effort to reduce ventilation and associated refrigeration requirements.
- Further analysis of equipment utilization to reduce fleet size.
- Handling of waste, including the following.
- Opportunity to use production ore passes as waste passes during the development phases.
- Optimisation of the mine air cooling system.
- Reduction of the development width of stope drill drifts and extraction drifts by fan drilling and extending the depth of production holes.

- 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 recognized a definitive answer may have to await exposures in underground workings.
- 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 prefeasibility-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 PFS, Mr Val Coetzee, that an acceptable metallurgical testwork programme was conducted on the samples provided. The range of samples tested appears to span the limits of the mineralised material from a grade perspective and includes the main domains identified by the geological team; however, further variability testwork will be required in the feasibility study phase to delineate variability of recovery response and cost to grade, domain and spatial location in the deposit of the material to be processed.

The mineralogical analysis provided invaluable information on the mineralisation within the mineralised zone and the mode of occurrence of the PGEs. This aided in the selection of the degree of comminution and the subsequent flotation process for the recovery of the platinum group metals and associated base metals (Copper and Nickel).

The testwork confirmed that the mineralised material can be classified as a very hard material that will require careful consideration from a crushing and milling perspective. The selected comminution circuit is considered as robust and conservative, and mitigates most risks associated with processing competent material. Further testwork in the next stages of the Platreef Project should be considered to evaluate alternative comminution technologies as are already being applied in other metallurgical operations.

The flotation testwork confirmed that the mineralised material contains various gangue mineral species that will challenge the operation in achieving saleable grade concentrate. The testwork was however successful in overcoming the PGE grade challenge without a significant loss in PGE recovery. The testwork results were confirmed at two independent metallurgical laboratories and therefore the risk of poor inaccurate results has been mitigated.

The current study is based on a flotation reagent suite (oxalic acid and thiourea) which is novel to the Platinum industry in South Africa. A conventional flotation reagent suite (employed in an MF2 circuit configuration) and an alternate reagent suite containing a targeted copper collector (MF1 configuration) achieved similar metal recoveries in preliminary testwork campaigns. Sufficient options therefore exist to mitigate the risks and provide the necessary operational flexibility.

Observations and conclusions from the testwork are accurately interpreted and included in the proposed concentrator design. Further testwork is required to test the robustness of the design and to optimise the various stages in the process.

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 program will have to be closely monitored to ensure that the correct skills are developed in time to support the construction and operational requirements of the Platreef Project.

25.6.1 Infrastructure Risks

The availability of sufficient water for the sinking and construction phase 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 70MVA 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 2014 PFS

26.1.1 Platreef 2014 PFS

The Platreef 2014 PFS presents the Mineral Reserve for Phase 1 of the Platreef development. Further work and studies should be undertaken to bring the Phase 1 to a Feasibility Study level. Additional studies should be undertaken to update the development scenarios. The development scenario expansions will require additional capital and may change the processing and refining route. The timing of Phases 2 and 3 will be evaluated at a later date and the decision to expand can be deferred or brought forward as markets dictate and funding permits.

26.1.2 Drill Programme

Ivanhoe is planning for an underground drill programme for delineation/grade control purposes. The focus is on the planned mining areas and not on the exploration targets.

Expansion drilling in Targets 1 and 2 would occur from underground would likely occur in 5 to 10 years; no drill plans have been finalized.

26.1.3 Resource Estimate

Amec Foster Wheeler recommends Ivanhoe continue to investigate the FW mineralization for the purpose of expanding the Platreef Mineral resoruces.

Amec Foster Wheeler recommends the assay tables for the ATS and, AMK drill hole 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.

26.2 Mineral Reserve Recommendations

The results and conclusions from the prefeasibility mining study are positive; as a result, Stantec recommends the advancement of the Platreef study to the feasibility level. Following are Stantec's recommendations regarding the mineral reserve for the feasibility study:

- Continue surface delineation drilling to further improve the level of geological knowledge and confidence in the Platreef block model and to update resource and mineral reserve classifications.
- Improve the spatial definition of the rock type and ore type boundaries in the block model.

26.3 Mining Recommendations

Following are Stantec's recommendations regarding for additional work and modifications to the current mine plans during the feasibility study:

- Mine layouts, ore and waste pass system, and designs should be refined and optimised to the extent possible in the feasibility study to enable more accurate scheduling and cost estimates.

- Stoping layouts should be prepared in greater detail with top cuts and bottom cuts as part of the optimization process.
- Shaft sinking and other development rates should be reviewed and modified as necessary in order to ensure that the preproduction development targets (milestones) are reasonable in the feasibility study.
- Alternative types and sources for backfill should be evaluated for the time period between production start-up and commissioning of the paste backfill plant.

26.4 Metallurgical Recommendations

The metallurgical testwork thus far has been sufficient to meet the requirements of the pre-feasibility study. A number of metallurgical areas require further definition and should be addressed in the subsequent stages of the Platreef Project. These include:

- Variability testwork must be conducted to delineate variability of recovery and processing costs to grade, domain and spatial location in the deposit of the material to be processed.
- The mineralised material is considered to be very hard. Comminution variability testing for Year 6+ is required.
- The complexity of the oxalic acid and thiourea reagent suite and the condition time requirement in order to be able to meet the recovery and grade targets.
- Risk mitigation for the oxalic acid and thiourea reagent suite with additional testwork on a conventional reagent suite with a targeted copper collector. This will ensure that existing alternative choices also provide operational flexibility.
- Flotation open circuit variability testing.
- Flotation closed circuit variability testing.
- Optimise the concentrate quality and grade. The need to reduce the sulphide gangue component reporting to the concentrate and the effect of chasing lower sulphur grades on the PGE recovery.
- Operation of a mini pilot plant.
- Concentrate thickening and filtration testwork.
- Rhodium recovery from locked cycle tests with residue analysis by means of nickel sulphide collection methods.

Further testwork is recommended on both variability point and composite samples to address the aspects identified above.

A better understanding of the mineralised material characteristics may open up avenues to introduce alternative technologies from a comminution, flotation and solid-liquid separation perspective.

The presence of floatable gangue species and the effect of these minerals on the grade-recovery profile is sufficient motivation for the commissioning of a pilot plant campaign to understand the interaction and potential build-up of floatable contaminants in the flotation circuit.

26.5 Infrastructure

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

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

Negotiations with Eskom 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. Alternative power sources need to be further investigated during the feasibility study.

26.6 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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